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OF SOUTH AFRICA.

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THE JOURNAL

OF THE

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OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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Vol. XII. JULY, 1911. No. 1.

Proceedings
AT
Ordinary General Meeting, July 15, 1911.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, July 15th, Mr. H. A. White (Vice-President) in the chair. There were also present:—

44 Members: Messrs. W. R. Dowling, R. Allen, M. H. Coombe, K. L. Graham, Tom Johnson, J. E. Thomas, Prof. J. A. Wilkinson, F. W. Watson, Dr. W. A. Caldecott, A. F. Crosse, J. R. Williams (Members of Council), A. Avent, J. E. R. Adendorff, W. Beaver, W. M. Coulter, G. G. Ferris, R. Fiander, R. Gascoyne, B. C. Gullachsen, A. J. Johnson, A. M. Kratz, R. Lindsay, H. F. Lofts, H. Lea, J. Lea, L. Marks, J. T. Milligan, G. Melvill, J. E. Metcalf, P. T. Morrisby, T. Graham Martyn, S. Newton, E. A. Osterloh, H. W. Pidsley, S. C. Quin, G. A. Robertson, A. L. Spoor, G. Hildick Smith, A. Thomas, W. E. Thorpe, Prof. G. R. Thompson, W. A. C. Tayler and J. Watson.

14 Associates and Students: Messrs. M. S. E. Archibald, A. R. Adams, E. G. Baskett, W. E. Burrows, R. Davis, C. L. Dewar, J. Davidson, D. C. Edington, B. W. Holman, W. J. R. Hunter, A. King, E. H. Tamplin, J. Thorlund and S. A. Woolf.

19 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as published in the June *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. Beaver and S. Newton were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been duly elected, as follows:—

Braidwood, James Quarton, Transvaal Coal Owners' Association, P. O. Box 1197, Johannesburg. Engineer and Analyst.

Currie, Walter, Willoughby's Consolidated Co., Ltd., P. O. Box 220, Bulawayo. Consulting Mining Engineer.

Ellis, Lionel Livingstone, Ginsberg G. M. Co., Ltd., P. O. Box 16, Knights. Assayer.

Ferdinand, Paul Werner, Mynbouw-Maatschappij, Lebong Soelit, via Benkoelen, W. Sumatra, Dutch East Indies. Mining Engineer.

Finch, Edward William, Yambulla Gold Mine, Yambulla, New South Wales. Mining Engineer.

Kirsch, George Hans, Consolidated Goldfields of S. A., Ltd., P. O. Box 1167, Johannesburg. Mechanical Engineer.

Noble, Edward, Lebong Soelit, via Benkoelen, W. Sumatra, Dutch East Indies. Mill Manager.

Pearce, Frederick Stewart, Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Assayer.

Stockett, Alfred Walton, M.I.M.M., M.Am.I.M.E., Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. General Manager. *(Transfer from Associate Roll.)*

Wood, Henry, Knights Deep, Ltd., P. O. Box 143, Germiston. Assayer.

The Secretary notified that since the last meeting of the Society the following Associates and Students had been admitted by the Council:—

As Associates—

Arundale, Francis Robert, P. O. Box 523, Kimberley, C.C. Managing Director, Abels Kop Gold Mines, Ltd.

Barrett, Francis Cecil, *c/o* B. Smallie, Esq., per Private Bag, Bulawayo, Rhodesia. Amalgamator.

Evans, Garnet Chesterfield, *c/o* Oriental Consolidated Mining Co., Ltd., Unsan, Korea. Cyanide Foreman.

Gieser, Herbert S., Brakpan Mines, Ltd., P. O. Box 3, Brakpan. Mining Engineer.

Heathcote, Charles Francis, M. I. M. M., A.M.I.C.E., F.G S., Tasmanian Gold Mine, Beaconsfield, Tasmania. Manager.

Lovelock, John Edward Jones, A.I.M.M., Progress Battery, Crushington, Reefton, New Zealand. Mill Superintendent.

Macdonald, William, Waihi G. M. Co., Ltd., Waikino, New Zealand. Metallurgist.

Mackey, E. Alexander, Crown Mines, Ltd. (A. Section), P. O. Box 156, Fordsburg. Assistant Assayer.

Parnell, Frederick Leopold, French Bobs Mine, Louws Creek, Mining Assistant. *(Transfer from Student Roll.)*

ROTHKUGEL, MAX, Ferreira Deep, Ltd., 46, Standard Buildings, Johannesburg. Assistant Surveyor.

STOCKDALE, CLAUDE, Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Cyanide Foreman.

As Students—

BULMAN, GERALD, Village Deep, Ltd., P. O. Box 1064, Johannesburg Assay Learner.

DAMANT, CECIL ATHERSTONE, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

DICKS, IVAN BERNARD, City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Assay Learner.

JOHNSON, ERNEST LEONARD, Consolidated Goldfields Laboratory, P. O. Box 108, Germiston. Assay Learner.

LITTLE, GEORGE HEARD, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

PHILLIPS, ROBERT MALDWYN, Nourse Mines, Ltd., P O Box 32, Denver. Reduction Works Learner.

TWYNAM, CYRIL DOUGLAS, Simmer Deep, Ltd., P. O. Box 178, Germiston. Reduction Works Learner.

WOOLF, SAMUEL ADOLF, S.A. School of Mines and Technology, 100, Van Beek Street, Doornfontein, Johannesburg. Student.

GENERAL BUSINESS.

The Secretary: I have to announce that a cable was received this morning from our President, Mr. C. B. Saner, which reads "Hearty thanks for the honour of election." Mr. Saner informs me by letter that he will leave London for South Africa on the 5th August.

I have also received a letter from the Hon. Secretary of the British Organising Committee drawing the attention of members of this Society to the fact that the Eighth International Congress of Applied Chemistry will be held in Washington and New York, U.S.A., from September 4th to September 13th, 1912, under the patronage of His Excellency the President of the United States. The Hon. President of the Congress is Prof. Edward W. Morley, Sc.D., Ph.D., LL.D.; the President, Mr. Wm. H. Nichols, Sc.D., LL.D., and the Secretary, Dr. Bernhard C. Hesse, 25, Broad Street, New York City.

The Congress is divided into the following sections and sub-sections :—

I. Analytical Chemistry.
II. Inorganic Chemistry.
IIIa. Metallurgy and Mining.
IIIb. Explosives.
IIIc. Silicate Industries.
IV. Organic Chemistry.
IVa. Coal Tar Colours and Dyestuffs.
Va. Industry and Chemistry of Sugar.
Vb. India Rubber and other Plastics.
Vc. Fuels and Asphalt.
Vd. Fats, Fatty Oils and Soap.
Ve. Paints, Drying Oils and Varnishes.
VIa. Starch, Cellulose and Paper.
VIb. Fermentation.
VII. Agricultural Chemistry.
VIIIa. Hygiene.
VIIIb. Pharmaceutical Chemistry.
VIIIc. Bromatology and Pharmacology.
VIIId. Physiological Chemistry.
IX. Photo-chemistry.
Xa. Electro-chemistry.
Xb. Physical Chemistry.
XIa. Law and Legislation affecting Chemical Industry.
XIb. Political Economy and Conservation of Natural Resources.

Papers will be accepted for reading and discussion in all the above sections, preference being given to those specially adapted for international discussion. They must be sent in not later than July 1st, 1912.

A preliminary pamphlet has been issued and may be obtained from the Hon. Secretary of the British Organising Committee, Charles G. Cresswell, Society of Chemical Industry, Palace Chambers, Westminster, S.W. Copies of the pamphlet may also be seen in the Society's office at any time and in the Seymour Memorial Library.

The Chairman: I should like to offer a hearty welcome to Mr. J. R. Williams and to express our sincere pleasure at seeing him once more in our midst. I have also to congratulate you on the presence of three of our Past Presidents which shows that passing the Presidential chair is not in every case like passing through the waters of oblivion.

Mr. John R. Williams (*Past-President*): I have to thank you very heartily and all the members present, for the kind greeting you have given me. I may say that it gives me very great pleasure to be back in Johannesburg and particularly to be at a meeting of this Society with which I have been connected from the commencement.

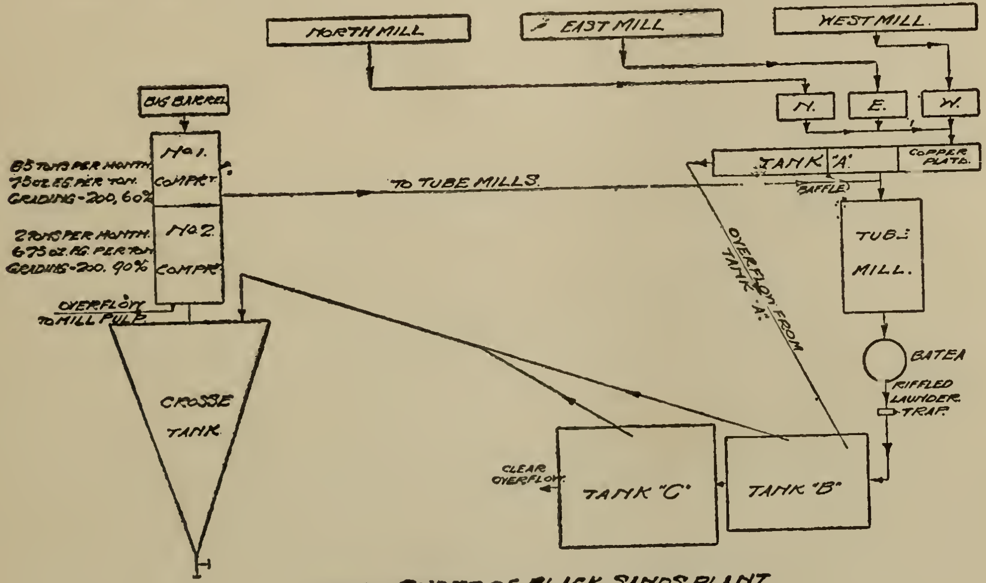
NOTES ON THE TREATMENT OF MILL CONCENTRATE.

By R. LINDSAY (Member).

As much attention has been given of late to the more careful collection and treatment of mill concentrate, a few notes on the procedure adopted at the Geldenhuis Deep, Ltd., may not be without interest to members of this Society. It will be understood that the following remarks apply more particularly to reduction works where operations are conducted on a fairly large scale, or to amalgamated mines working as sectional units over a reasonably limited area. The concentrate treated here results from the daily scrape from the mills at north, east and west sections, also the big barrel residue from the respective mills; the concentrate retained on the plates with the amalgam being taken off together and transported to the central amalgam room by 11 a.m. in locked amalgam safes provided for the purpose. The scrape is then ground in barrels (a separate barrel being provided for each section) for 4 or 5 hours, and cleaned in the usual way over a stationary copper plate, the black sand settling in a tank "A," 4 ft. 6 in. × 2 ft. × 2 ft., fitted with a baffle at 18 in. from the end adjacent to the plate, the heavy material, grading 45% − 200 (·003″), settling behind the baffle; what remains on the further side overflowing through a 2 in. pipe to tank "B" outside the building, the grading of the fine product carried over being 98% − 200. A reference to the following flow sheet will make the procedure adopted a little more clear.

oz. of mercury being maintained therein. The batea acts most effectively as a trap, 7 oz. of mercury and 50 oz. of amalgam per ton being regularly caught. This 50 oz. of amalgam yields 11·28 oz. fine gold at a cost per ton of 3s. for power and 5s. for labour.

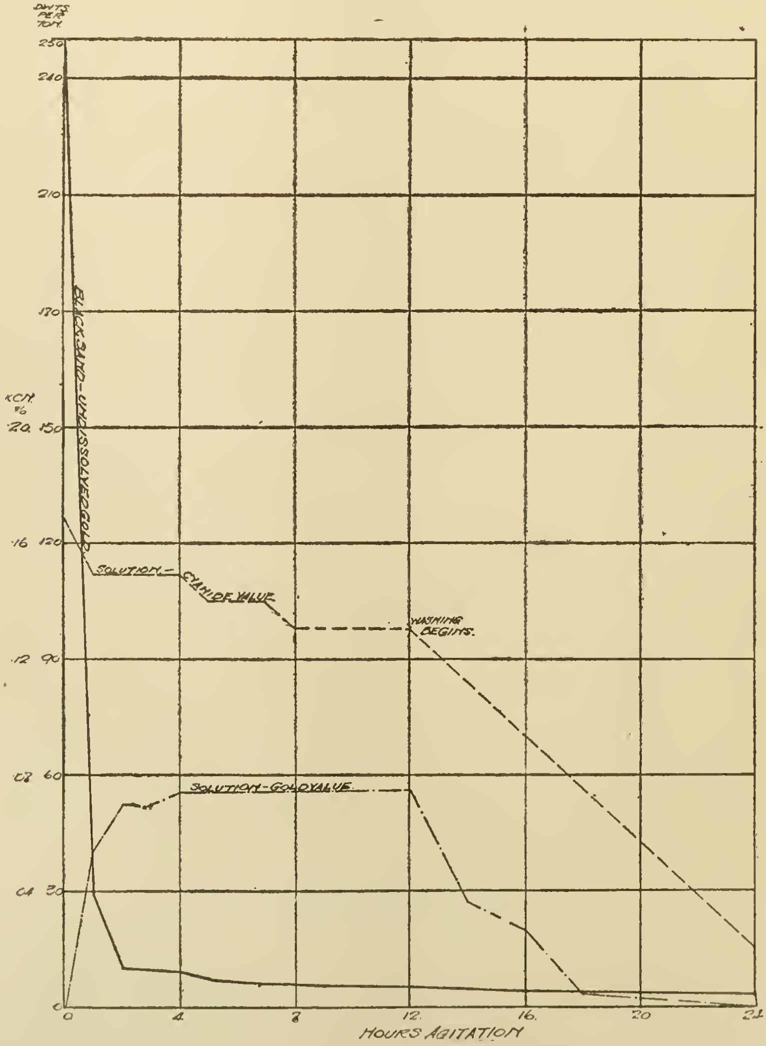
From the batea the concentrate pulp passes over a riffled launder and trap to the tank "B," thus joining the overflow from tank "A." The overflow from tank "B" passes into a larger settling tank "C," the overflow from which is perfectly clear and is allowed to run to waste. The product in the "C" tank grades very fine, being all − 200, and assays 12 to 14 oz. fine gold per ton, and represents about 5% of the total solids caught in the tank "B." The concentrate from the big barrels in the respective mills is washed over a stationary plate in the usual way and is caught in a large tank divided into two by a partition crosswise in the centre. The product from the first compartment, amounting to 3·5 tons per month, grading 60% − 200 and assaying 7·5 oz. fine gold per ton is transported to the central amalgam room every month



FLOW SHEET OF BLACK SANDS PLANT.
GELDENHUIS DEEP LTD.

The tank "A" is cleaned out daily and produces about 250 lb. dry weight of concentrate, grading 45% − 200 as already stated, and assaying about 30 oz. fine gold per ton. This product is then fed to a tube mill 4 ft. 6 in. × 3 ft. 6 in., lined with 6 in. × 6 in. × 3 in. silex blocks and running at 26 r.p.m. using pebbles as the grinding medium. The product from the tube mill, grading 98% − 200, now flows into a batea running at 100 r.p.m. with a 3 in. stroke, a bath of 250 end and is tube milled. The product from the second compartment, amounting to 2 tons per month, grading a little over 90% − 200, and assaying 6·75 oz. fine gold per ton is transported direct to the reduction works at east section for cyanide treatment.

Cyaniding the Mill Concentrate.—For cyaniding the finely ground concentrate, the conical tank designed by Mr. Andrew F. Crosse and described by him in this *Journal* (Nov. 1909), has been adopted and has proved most satisfactory. The tank, 7 ft. 6 in. in diameter and 8 ft. 4 in. deep and having a content of about 122 cub. ft., treats comfortably 1,800 lb. dry weight of material, and there is no trouble whatever in obtaining a clear solution for decanting, agitation with air going on meanwhile. As one would expect in treating material in so fine a state of division, the gold goes rapidly into solution, a charge assaying 250 dwt. fine gold per ton being reduced to under 30 dwt. after one hour's agitation as shown in the diagram.



The practice here is to agitate the charge with air for 12 hours before commencing to wash with ·02% cyanide solution. At the end of 12 hours agitation the undissolved gold in the concentrate is down to 6 dwt. per ton, while the cyanide solution has a gold value of 56 dwt. per ton. Washing is then proceeded with at the rate of half a ton of solution per hour, 18% of the solution in the tank being thus displaced every hour, agitation going on meanwhile. The quantity of solution required for treating the 1,800 lb. charge is 6 tons. This circulation on the Crosse principle with simultaneous decantation when washing, effects a considerable saving of time over settlement and decantation, considerably increasing the capacity of the conical tank. The washing is maintained for a period of 12 hours, making the total time of treatment 24 hours, at the end of which the concentrate residue assays 4 dwt. per ton, and the solution ·08 dwt. per ton. This is at present the economic limit to which treatment may be carried. The cyanide consumption at 2·75 lb. of 120% KCN is rather high due to the refractory nature of the material dealt with, but it has gradually been reduced from double the present consumption, and it is hoped that it will be still further reduced. Possibly there may be a useful field here for one of the various kinds of magnetic separators to eliminate fine iron.

The air for agitation is obtained from a little Ingersoll Rand compressor on the accumulated slime plant; a $\frac{3}{4}$ in. branch pipe from it is allowed to enter a receiver fitted with a pressure gauge, just before entering the Crosse tank, thus allowing the consumption of air to be measured and the cost calculated. The consumption is 7 cub. ft. of free air per minute costing 5d. per ton of concentrate treated. The quantity of mercury recovered from the Crosse tank averages up to the present only about 1 oz. per ton, thus showing the effectiveness of the batea as a trap as already mentioned. The total cost ot transporting, grinding and cyaniding is 26s. per ton as follows :—

		Per Ton.	
Tube Milling.	Transport and labour	8s.	
	Power	3s.	
	Pebbles	4s.	15s.
Cyaniding.	Transport and labour	8s.	
	Cyanide	2/3	
	Lime	4d.	
	Air	5d.	11s.
			26s. per ton.

The average value of the concentrate produced at the three sections of the Geldenhuis Deep is 16 oz. fine gold per ton, having a value of about £67. Taking the average residue at 4 dwts., and cost of treatment at 26s., the profit per ton stands at £64 18s. The economic advantage of treating the concentrate on the mines where it is produced instead of selling it to customs works is apparent even when the basis of payment is as high as 94% of the gold content at 84s. per fine oz., less £6 per ton for treatment charges. Taking the concentrate at 16 oz. per ton as above, the profit on selling would be £56 13s 4d., allowing 10s. per ton for bagging and transporting, the mines profit being thus reduced by £8 4s. 8d. The concentrate plant installed here towards the end of last year at a cost of £576, including royalty on the Crosse tank, has already handsomely paid for itself, gold to the value of over £2,000 having been recovered up to the end of last May.

I have to thank Mr. H. M. Thomas, General Manager, and Mr. W. M. Coulter, Reduction Officer for permission to publish these notes and figures.

The Chairman : This is one of those short papers dealing with practical matters in a practical way, which are always very welcome in this Society. I declare it open for discussion.

Mr. K. L. Graham (*Member of Council*): Before proposing a vote of thanks to Mr. Lindsay for his interesting and practical note, which, to be in the fashion, I might describe as the last word on the treatment of battery concentrate, I should like to add a few remarks, rather in amplification than in a spirit of criticism. I may say that the investigation into the best methods of handling this by-product, began on the Geldenhuis Deep some months ago and has entailed considerably more work than appears from the condensed figures submitted by the author this evening. It may not be generally realized, nevertheless it is the case, that an extraordinary want of uniformity exists in the treatment of this material on our mines; in many cases the methods in vogue are crude to a degree and undoubtedly result in considerable losses to the companies victimized in this respect. In looking over the cost figures, transport and labour seem high; this, however, is largely due to the arrangement of plant, it having been necessary to erect the Crosse tank at some considerable distance from the central smelting room, in which the preliminary grinding work is done; were this otherwise, this item could probably be reduced by 50%. An interesting figure is that given for air, which it will be seen is almost negligible. At the time the tank was erected, no data were available from which the amount of free air required to agitate and aerate a given weight of current battery concentrate in a tank of these dimensions, in the shortest possible time, could be calculated. The source of supply was the mine compressor, a $1\frac{1}{4}$ in. pipe, reduced near the tank to $\frac{3}{4}$ in. bringing air at pressures varying from 40 to 70 lb. per sq. in. It was apparent after working the tank for a short time that this arrangement afforded a serious loop-hole for waste, as by careless manipulation of the controlling valve 300 to 400 cub. ft. of free air could be blown through the pulp to no purpose. By inserting a 3 in. chamber at a point close up to the tank, fitted with a gauge, and experimenting with blank flanges drilled with different sized orifices, it was comparatively easy to compute the amount of free air passed in a given time. By plotting a series of curves it was found that this could be reduced to less than 10 cub. ft. per minute without affecting the rate of dissolution of the gold or the ultimate residue. As the variable pressure made it impossible to satisfactorily control the air supply, a connection was made to a small Ingersoll Rand compressor, as stated by Mr. Lindsay, the air being delivered at about 5 lb. pressure. This has proved to be a much more satisfactory and economical arrangement in every respect.

One other specially interesting feature in the flow sheet put before you is the Batea after the tube mill instead of the usual copper plate, shaking or otherwise. From the figures given it will be seen that the work done by this machine is excellent. I may say about half the direct recovery is due to regrinding in the tube mill, the balance being finely divided mercury and amalgam which have escaped the stationary copper plate. This Batea which runs particularly smoothly and without jar, is suspended at two points, the lip being supported by a ball bearing of most excellent design obtained from Mr. Harland at the Robinson G. M. Co., whom I think I might describe as the most successful exponent of Batea work on the fields. In conclusion I should like to congratulate Mr. Lindsay on the thoroughness of his work and the completeness of his results, and I now ask you to accord him a hearty vote of thanks for his interesting and valuable paper.

Mr. Andrew F. Crosse (*Past-President*): It is naturally very gratifying to me to find that the author's contribution read at this meeting, proves that the statements made in my paper on my slime process, read before this Society in 1909, are perfectly correct.

The clear overflow is easily maintained and the amount is 18 % of the conical vat capacity per hour, which was the exact proportion which I had mentioned in my paper. For slime or finely crushed ore having a lower value. half the time of treatment would be possible, namely 12 hours. In fact for very poor slime only carrying a few pennyweights I am convinced that three treatments could be carried out in one vat in 24 hours. One of the most important conditions necessary for obtaining a high extraction from pyritic concentrate is that the concentrate should be reduced to the finest possible state of division. I do not consider a small tube-mill to be the most efficient mechanical device for the purpose, as it is easily possible for the pyritic pulp to have such a high specific gravity as almost to neutralize the grinding action of the pebbles. I have lately invented a new fine grinder in which the old bucking-board principle of grinding has been converted into a rotary acting machine with an oscillating muller. Even in my new grinder (which is only a small working model) I can reduce 600 lbs. of pyritic concentrate so that it will all pass through a screen having 200 holes to the lineal in. per 24 hours. This is an absolutely ideal product for amalgamation and cyaniding. I shall be very pleased to shew the new grinder to anyone it may interest at my laboratory.

The London mining people have often remarked to me that we are very much too conservative out here, and one of the Past-Presidents said to me that when an inventor comes along out here there seems to be a feeling as if every one was up in arms crying out "Here's a head, let us hit it." The first stage of a new process is very difficult, then when a process is successfully proved every one wants to use it for nothing although it is patented. For instance a mine in Mexico is sliming all its ore and recovering its gold by my process which is patented in Mexico, and so far I have not received one cent from the Company.

EXPERIMENTS IN THE TREATMENT OF ACCUMULATED ORE SLIME BY AIR LIFT AGITATION.

By J. E. R. Adendorff, F.C.S. (Member).

It is a well known fact that accumulated acid ore slime is not so amenable to the usual cyanide treatment as neutral slime; that is to say, the consumption of free cyanide is greater, and the residue contains more gold. With a view to determining the best method of treating such accumulated slime, and to find out, if possible, the rate at which the gold dissolves in the cyanide solution the following experiments were undertaken. The cyanicides in accumulated slime from banket ore appear to be free sulphuric acid and partly oxidised iron-sulphur compounds. It appears that, during treatment, it is these partly oxidised compounds and free acid that destroy the free cyanide, and, by absorbing a large amount of oxygen in order to complete oxidation, interfere with the solution of the gold, according to Elsner's formula, thus accounting for poor extraction. In the first experiment the action of the slime in regard to ordinary treatment was noticed. A sample of about half a ton was collected, carefully mixed, and bagged in 100 lb. lots. On examination one sample was found to contain :—

Moisture ...		22·70%
Acidity as H_2SO_4 ...		·024%
Gold ...		2·31 dwt. p. t.
Grading ...	+ ·01 in.	2·10%
,,	− ·01 + ·006 in.	9·23%
	− ·006 in.	88·67%

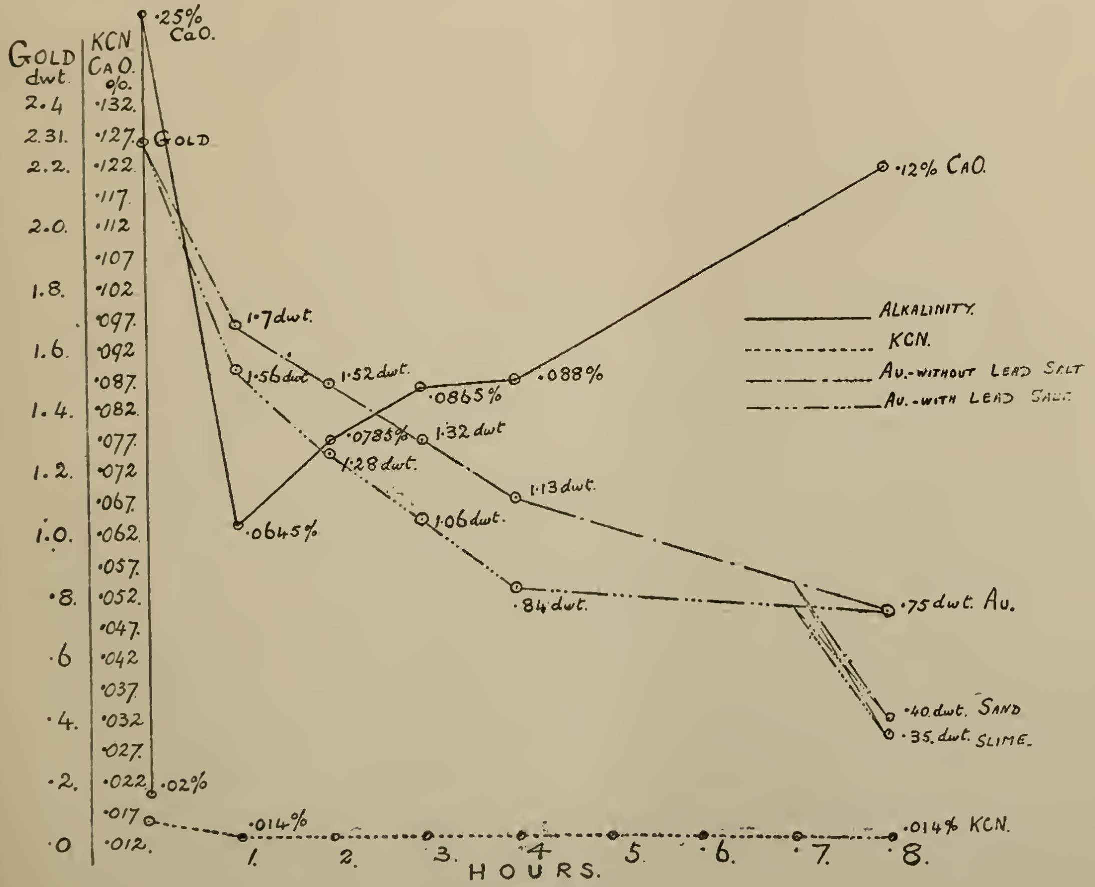
A precipitated works slime solution was used as a solution in the proportion of two of solution to one of slime by weight. This solution contained ·017% "free" KCN ("total" ·024% KCN) and had an alkalinity of ·02% CaO. Ten charges were prepared for bottle agitation, and each had finely powdered lime added at the rate of 5 lb. per ton. Six charges had lead acetate added at the rate of $\frac{1}{4}$ lb. per ton of ore slime. These charges were then well agitated in pairs for periods of 1, 2, 3, 4 and 8 hours respectively. After each period of agitation the charge was allowed to settle for about fifteen minutes to allow sufficient of the solution to be drawn off for the purpose of testing alkalinity and free cyanide strengths; while the residues, after being well washed, were dried and assayed for undissolved gold. In the eight hours sample both charges had lead acetate added, and the residues were separated into slime proper (–200 mesh, 0·003 in. aper.) and sand before assay. The following results were obtained from the ten charges:

Hours.	Cyanide Solution. CaO.	KCN.	Residues. Assay Values.
	per cent.	per cent.	
1a	·064	·014	1·70 dwt.
1b	·065	·014	1·56 dwt.
2a	·075	·014	1·52 dwt.
2b	·082	·014	1·28 dwt.
3a	·088	·014	1·32 dwt.
3b	·085	·014	1·06 dwt.
4a	·084	·014	1·13 dwt.
4b	·092	·014	·84 dwt.
8b aver	·120	·014	·35 dwt. slime ·40 dwt. sand

Note.—The letter "a" denotes without lead acetate.
The letter "b" denotes with lead acetate.

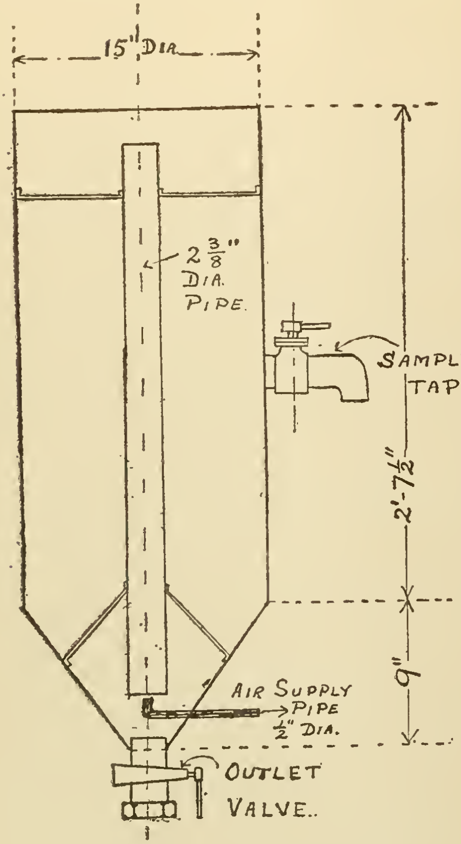
This experiment showed clearly that those charges containing lead acetate yielded better extraction of the gold than those without; also that the free cyanide was well protected by the lime present, while the cyanide consumed for dissolving the gold must have come from the cyanide reported as combined. The eight hours charge showed that the undissolved gold was more or less equally distributed in the slime proper and sand (+200 mesh). The high residue seemed to point to the fact that there was insufficient oxygen in the charges, those containing lead acetate lending proof to the belief.

The further experiment (Trial No. 1) was made in a small Brown or Pachuca vat, made to scale to hold about a 100 lb. slime charge. A



A.—Bottle Shaking Test.

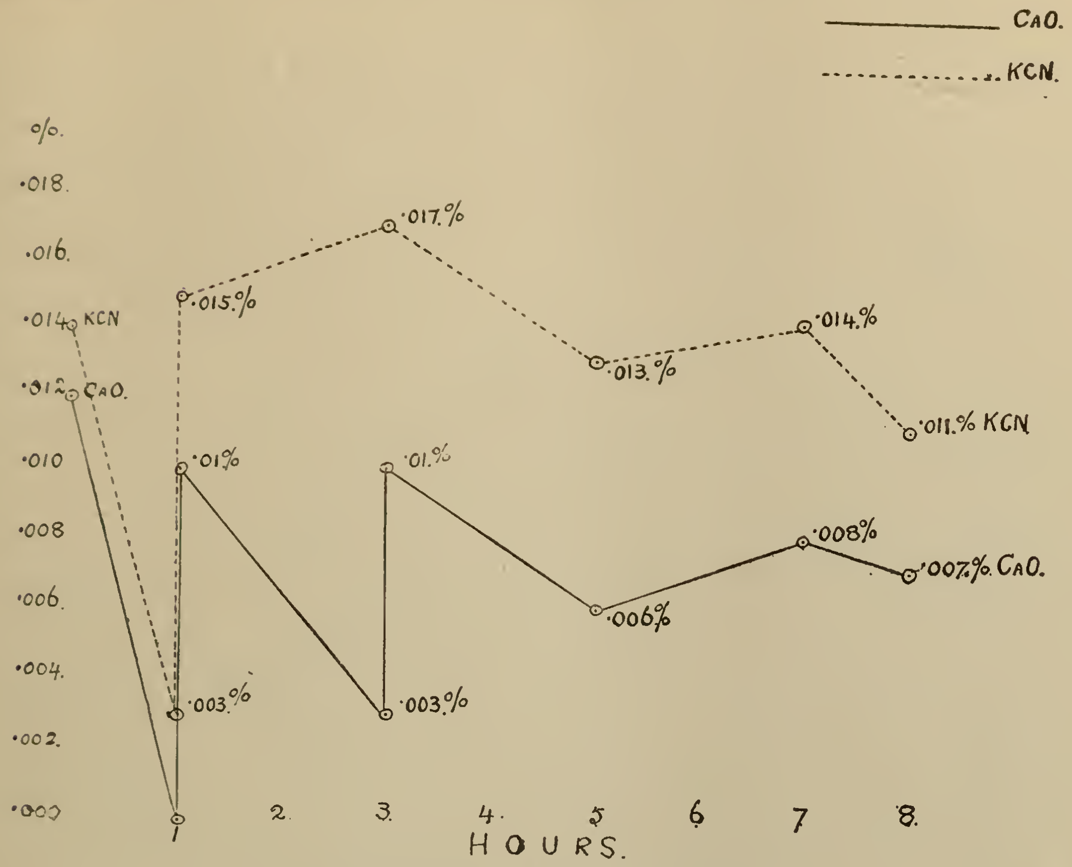
sketch of the vat used, giving actual measurements, will be found below.



Detail of Vat Used for Experiments.
Scale 1½ in. × 1 ft.

A 100 lb. charge of slime was gradually puddled with a works precipitated slime solution and transferred to the vat, samples for moisture, grading and gold content having been previously taken. Care was taken to start the air-jet beforehand with sufficient force to just make a movement of the mass. Soon after starting a sample was taken for specific gravity, which showed the ratio by weight to be 2·47 of solution to 1 of slime. After one hour's run samples were taken for alkalinity and free cyanide strengths at the sample tap at the side of the vat. The precaution was taken of drawing off some of the ore slime before taking the actual sample in order to minimise the chance of any slime concentrating in the tap. After determining the alkali strength of the clear solution, lime was added to raise the strength to ·01% CaO, while the free cyanide was also raised to ·015% KCN Agitation continued for eight hours without interruption, samples being taken after 3, 5, 7 and 8 hours run. The final sample was a large one, so as to allow assays to be made of the cyanide solution, and for undissolved gold in the residue. Care was taken to have the residue washed free from cyanide as soon as possible, after taking off sufficient for assay. The results will be found tabulated below with the results of subsequent tests. This test demonstrated very clearly the acid and oxygen-absorbing nature of the slime, and the great loss of cyanide that would occur if not well protected by alkali. (See B for explanatory diagram.) It will be seen that during the first hour the free cyanide and alkali fell from ·014% and ·012% to ·003% KCN and "faintly acid" respectively. These strengths were therefore brought up again to ·015% KCN and ·01% CaO by the addition of fresh cyanide and lime. The third hour showed the beneficial effect of the added alkali, for the free cyanide strength rose to ·017% KCN, its maximum point, while on the other hand the lime was neutralised to the extent that it only showed ·003% CaO. This strength was again raised to ·01% CaO. The subsequent tests showed more advantageous conditions and needed no further additions. It is interesting to note that after the third hour the strength of free cyanide and alkali appear to be in sympathy with each other, the diagrammatic lines being more or less parallel with each other up to the eighth hour. The original slime assayed 2·48 dwt. gold and the residue ·755 dwt., again showing the need of lead acetate in order to improve the extraction. The large amount of lime used shows that aeration tends to produce more acid compounds than under ordinary circumstances.

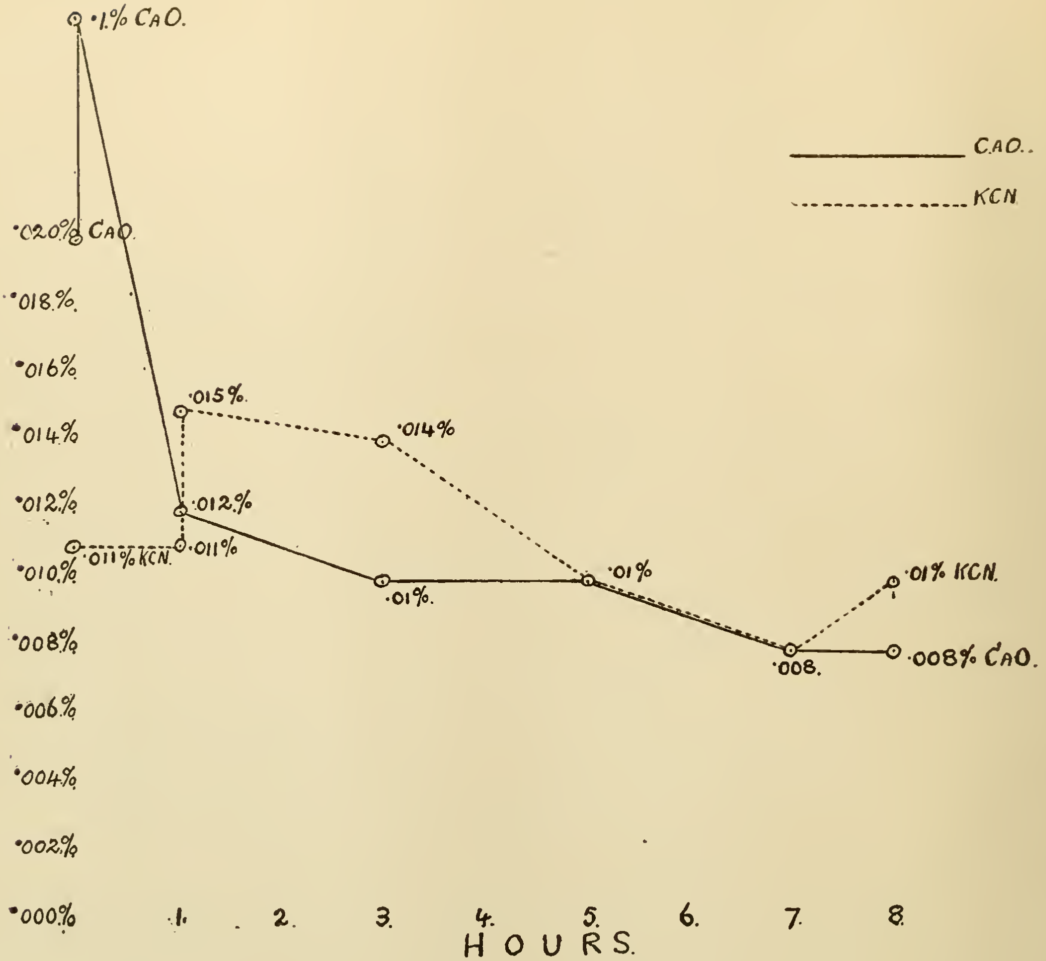
In the next trial (No. 2) finely crushed lime, at the rate of 2 lb. per ton, was thoroughly mixed with the slime before being puddled with the solution, instead of in bulk as before. Lead acetate, at the rate of 1 lb. per ton was added to the working solution before commencing treatment. Samples were taken as in the previous trial, and the results are given in diagram "C" below. This trial showed the advantage of adding lime before commencing treatment, as the free cyanide was well protected. After the first hour it was desired to raise it to ·015% KCN. The addition of lead acetate confirmed the preliminary bottle shaking test, for it will be observed that the undissolved gold in the residue was lower than in the previous trial, being ·28 dwt., while the original slime assayed 2·41 dwt. From the diagram it will be seen that while the alkalinity was high enough to prevent loss of free cyanide during the first hours of treatment, when it fell to ·01% CaO, there was a sympathetic fall in cyanide strength.



B.—Aeration Test in "Brown" or Pachuca Vat.

Another experiment (Trial No. 3) was made under similar conditions to the last, except that the ratio of solution to ore slime was brought as near as possible as 2 : 1, determination proving it to be 2·03 of solution to 1 of ore slime. This change meant a larger weight of slime in the same amount of solution, and a corresponding drop in alkalinity and perhaps free cyanide might be expected. This proved to be the case, for after the first hour the strengths were ·006% KCN and ·008% CaO. As the lime added could not have been all neutralised, or even all dissolved, it was not considered necessary to make further additions. Cyanide, however, was added to bring the strength up to ·02% KCN. After the first hour there is a marked parallelism between the rise and fall in cyanide and alkali strengths. (See diagram D.) The extraction was also good, the original slime assaying 2·53 dwt., while the undissolved gold in the residue was ·20 dwt., the best extraction so far obtained. The tests seemed to show, therefore, that success in treating accumulated acid ore slime depended on maintaining a high alkalinity during the first three hours of aeration and that lime should be added to the slime before treating with cyanide solution, forming a better protection for the free cyanide; also that lead acetate tends to improve the extraction.

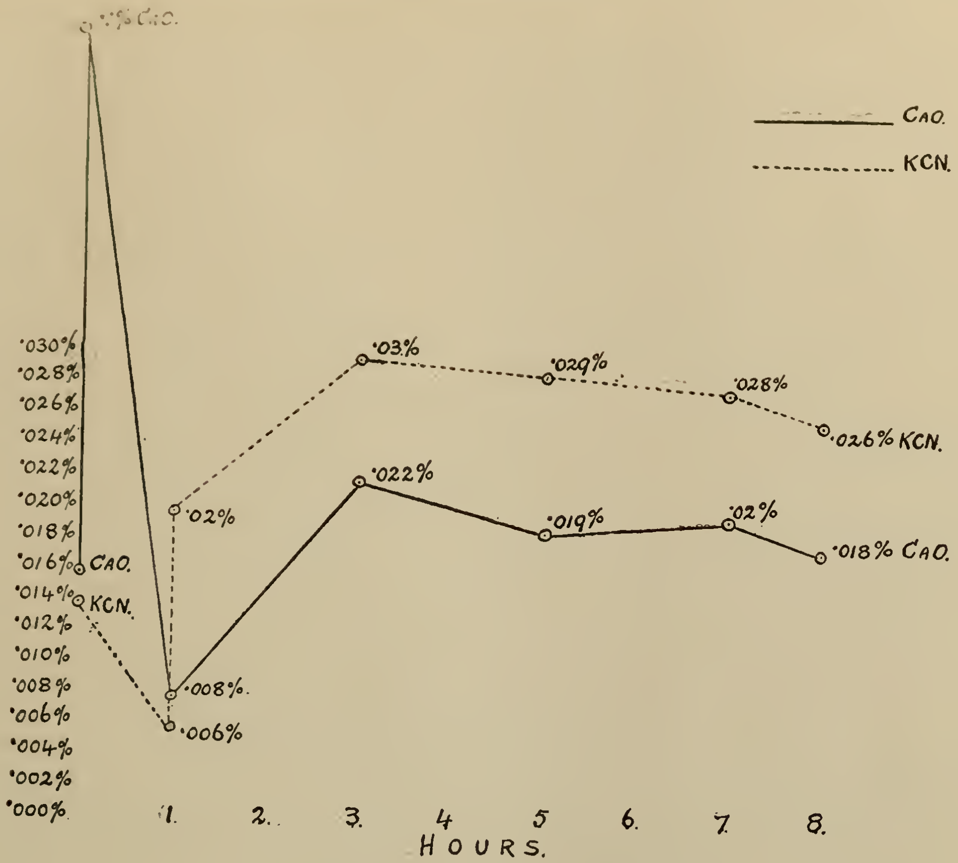
With all the foregoing information a final experiment (No. 4) was made with a view to determine, if possible, the rate of solubility of the gold in the accumulated acid ore slime in cyanide solution. Finely crushed lime was added to a charge of 100 lb. of slime and thoroughly mixed. The free cyanide strength of the working solution was brought up to ·02% KCN, and lead acetate was added at the rate of 1 lb. per ton of slime. The specific gravity of the pulp showed the ratio to be 2·06 of solution to 1 of slime. Samples were taken as in previous tests, after 1, 3, 5, 7 and 8 hours, then every two hours up to 18 hours, and a final sample after 30 hours treatment. In each case, however, besides testing for alkalinity and cyanide strengths, a sufficiently large sample of the pulp was taken for undissolved gold assays. After the first hour it was noticed that the alkali strength had fallen to



C.—Aeration Test in "Brown" or Pachuca Vat. (Lead Acetate at the rate of 1 lb. per ton was added.)

·01% CaO, and, as already experienced, there was a sympathetic loss of free cyanide to the same figure, the latter it was considered necessary to raise to ·02% KCN. Only the final sample of the treatment solution (30 hours) was assayed for its gold content. The results of this test are found hereunder. The undissolved gold in the residue after 18 and 30 hours showed practically no difference, some assays checking absolutely, while others differed only ·02 dwt.

It will be noticed that the value of the slime was different in each trial. This is due to the difficulty of obtaining a perfectly mixed sample when working with half a ton sample. Also it is quite within the bounds of reason that the free acidity of the accumulated slime varies considerably, and no hard and fast rule can be laid down for amount of lime to be added. It is interesting to observe, however, from these tests that about ·012% CaO seems to be the minimum alkalinity that should be maintained during treatment in order to protect the free cyanide, for it will be noticed from the diagrams that when the alkalinity falls below that figure, there is a loss of cyanide. The final experiment shows that during the first hour the various reactions are most active, more than 50% of the gold in the ore slime going into solution. From the eighth to twelfth hours there is a further solution of only ·04 dwt. gold. No doubt it is possible, as shown in the previous tests, that a residue of ·20 dwt. may be obtained in eight hours treatment, instead of 18 hours as in the final experiment. It is also interesting to note that the alkali and free cyanide strengths remain unchanged after sixteen hours to the end of the treatment. In summing



D.—Aeration Test in "Brown" or Pachuca Vat.

up it may be concluded that successful treatment of accumulated acid ore slime depends upon the oxidation and neutralisation of the cyanicides in the slime, and the maintainence of a high alkalinity during the first hours of treatment.

I wish to thank Mr. F. A. G. Maxwell, the Consulting Metallurgist of the Randfontein group, for permission to publish these results.

The Chairman: This is a very useful paper and I am quite sure it will be very welcome to our proceedings. I am bound to point out one serious omission. No reference is made to the preliminary aeration before the addition of any cyanide. Dr. Caldecott, years ago, laid it down as a standard that before cyanide was added at all, sufficient aeration must be given to oxidize all reducers present. This is the first occasion since then when any experimenter has dealt with accumulated slime problems without any reference to this point.

Mr. W. R. Dowling (*Vice-President*): I have much pleasure in seconding the vote of thanks to Mr. Adendorff for his carefully prepared paper. The author sets a good example in following the recommendations of the terms committee, as published in our past proceedings.

The paper represents a good deal of work, and I think it a valuable one to have on record. The Chairman has anticipated me in making the point that accumulated slime should receive preliminary aeration in alkaline water before the addition of cyanide.

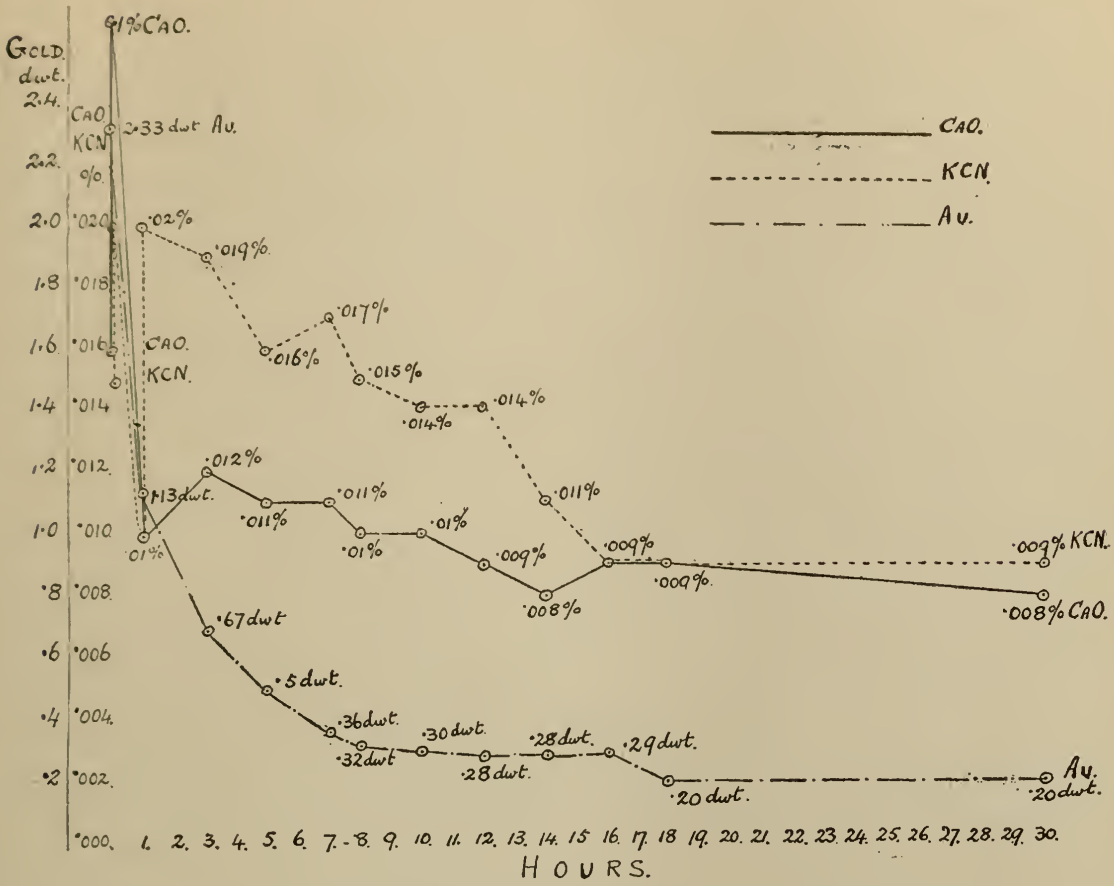
Judging by the gradings given the slime appears more like a well crushed pulp, and it is a question whether it would not have led to better results if the sand had been separated and treated as such.

The following are detailed results of all the experiments (Diagram E (on next page) showing the results of the final test.)

	Trial 1.	Trial 2.	Trial 3.	Trial 4.
Specific gravity	1·2334	1·2250	1·2620	1·2590
Moisture	24·5%	23·5%	18·75%	17·4%
Grading, + 60 (·01 in. aperture) ...	1·91%	2·64%	1·46%	1·31%
,, − 60 + 90 (·006 in.) ...	4·33%	6·52%	5·32%	4·25%
,, − 90 + 200 (·003 in.) ...	12·63%	10·54%	12·97%	11·1%
,, − 200 ,, ...	81·13%	80·3%	80·25%	83·34%
Value of ore slime	2·48 dwt.	2·44 dwt.	2·53 dwt.	2·33 dwt.
Value of solution	·08 dwt.	·03 dwt.	·04 dwt.	·21 dwt.
Value of residue	·755 dwt.	·28 dwt.	·20 dwt.	·20 dwt.
Value of final solution ...	·71 dwt.	·93 dwt.	1·15 dwt.	1·28 dwt.
Free KCN in solution ...	·014%	·011%	·014%	015%
Total KCN in solution ...	·026%	·023%	·023%	·030%
Alkalinity as CaO	·012%	·020%	·016%	·016%
Ratio of solution to ore ...	2·47	2·43	2·03	2·06

STRENGTH OF SOLUTIONS.

Time in Hours.	Trial 1.	Trial 2.	Trial 3.	Trial 4.	Undissolved Gold.
At start.	per cent.	per cent.	per cent.	per cent.	
	CaO ·012	·10	·10	·10	2·33 dwt.
	KCN ·014	·011	·014	·02	
After 1 hour ...	CaO nil.	·012	·008	·010	1·13 dwt.
	KCN ·003	·011	·006	·010	
After 3 hours ...	CaO ·003	·010	·022	·012	·67 dwt.
	KCN ·017	·014	·030	·019	
After 5 hours ...	CaO ·006	·010	·019	·011	·50 dwt.
	KCN ·013	·010	·029	·016	
After 7 hours ...	CaO ·008	·008	·020	·011	·36 dwt.
	KCN ·014	·008	·028	·017	
After 8 hours ...	CaO ·007	·008	·018	·010	·32 dwt.
	KCN ·011	·010	·026	·015	
After 10 hours ...				CaO ·010	·30 dwt.
				KCN ·014	
After 12 hours ...				CaO ·009	·28 dwt.
				KCN ·014	
After 14 hours ...				CaO ·008	·28 dwt.
				KCN ·011	
After 16 hours ...				CaO ·009	·29 dwt.
				KCN ·009	
After 18 hours ..				CaO ·009	·20 dwt.
				KCN ·009	
After 30 hours ...				CaO ·008	·20 dwt.
				KCN ·009	



E.—Diagram showing Curves of Gold Solubility and Strengths of Cyanide and Alkali at Various Periods during Treatment.

THE ASSAY OF USED PLUMBAGO CRUCIBLES FOR GOLD.

(*Read at May Meeting.* 1911.)

By JOHN WATSON, F.I.C (Member).

DISCUSSION.

Mr. Dan Nicholas (*Associate*): I think assayers should appreciate the author's paper, although he does not throw much fresh light on the subject. I agree with him as to the usefulness of the maligned nitre. I am of opinion, that in order to get an accurate assay value of material containing graphite to any extent, it is necessary to work on a larger quantity than a 0·1 or 0·2 A.T. charge. In support of this I take the liberty of mentioning a method which I have found very useful, and as far as my experience goes, accurate, for it has always held its own both for silver and gold in samples, referred to arbitration; the work can also be dove-tailed in with ordinary routine assay work.

Method.—Weigh up 0·5 A.T. of sample (−80 mesh sieve), mix with an excess of flux (the flux I used was a stock flux made up of 7 parts soda, 4 parts borax, and one and one-third parts litharge, by measure). Add silica if necessary. Fuse quietly for about half-an-hour in an ordinary "H" crucible, pour off the fluid slag into a mould, oxidise the unfused contents of the crucible by the addition of sufficient nitre, helping the action by giving the crucible a rotary shake, return the slag poured off previously to crucible, which replace with its contents in the fire, and complete fusion. Charcoal may now be added in form of a wash to make sure of getting lead down. Silver for parting purposes is added to the lead button in required quantity.

PRACTICAL NOTES ON COAL.

(Read at February Meeting, 1911.)

By Michael Dodd (Member).

REPLY TO DISCUSSION.

Mr. Michael Dodd (*Member*): In the discussion which took place on my "Notes" a few points of real importance were raised. One or two I admit to have overlooked: and, possibly, one or two I had purposely left alone.

Information more specific than that which was given as to the relative values of the different coals produced by the various collieries would, doubtless, have been of real interest to those who are purchasers, but it seemed to me well to avoid any institution of "comparisons" which might have been odious, so that no mention was made of the names of individual collieries. My endeavour was to suggest the general principles to be kept in mind when coal was being bought, and to indicate a rough method of assessing different coals, given analysis and calorific value. The following information, however, may not be without interest:—

TRANSVAAL COALS.

	Calorific Value.	Volatile Contents.
Springs District ...	10·3 to 10·5	20 to 31
Vaal River and O.R.C.	do.	about 27
Heidelberg	11	21
Middelburg	12·5 to 13·1	22 to 29
Natal	13·5 to 14	19 to 26

A point of very considerable importance was raised by Mr. McArthur Johnston when he expressed disappointment that the possibility of making coke from the coals of the Transvaal had not been alluded to. On this question we are in want of more information than is at present available. Some time ago the Transvaal Government had a number of samples of coal sent to England to be made into coke, but I am not aware that the results of the tests of those samples have yet been published.

In Natal, coke of fair quality is being made in very small quantities in the Hatting Spruit area, and I have seen from Burnside samples of coal which promise to yield coke of a still higher grade.

In the Transvaal, even the best of the coals yet discovered are, in bulk, too high in ash to produce coke of even moderate quality. The ash contents of the coals from the Middleburg field may be from 14% to 16%. The 60% of the weight of coal which comes out of the ovens in the shape of coke contains all the original ash. And a coke containing 20% or over of ash is not good enough. A small portion of the seam worked in the above-named field has always appeared to me as of quality such as to make a reasonably fair coke, but, for two reasons, the manufacture could not be economical.

1. It is costly, in coal mining, to completely separate a small portion of the seam from the bulk.
2. In these days the coal produced is sold on its calorific value. Any colliery separating the best of its product for a specific purpose would, in the process, reduce the value of the balance. And this would only be justified if a high price were obtainable for the small portion taken out.

To illustrate this: one of the collieries supplies gas coal specially selected from a portion of its seam, and to do so must have a price which is something like four times that obtained for average product.

The sample of Burnside coal referred to likely to give good coke, analysed as follows, viz.:

	Per cent.
Fixed carbon	66·07
Volatiles	24·43
Moisture	0·69
Ash	8·81

The question as to whether the B.T.U. is the unit required to heat a unit of water from 39° F. or from 60° is a point of purely academic interest. I cannot imagine that the measure of heat required at the two points of temperature could by any chance vary because of the difference of density. Nor can one suppose a laboratory test made by the process of heating water one degree only. It should be enough to speak only of one degree. Yet I admit having stumbled into the mention of the 60°.

It was not quite without a smile that I heard Mr. McArthur Johnston's suggestion that a form of contract for the purchase of coal might have been outlined. I have seen many, and my first suggestion after a wide experience would be: "secure essentials, but always aim at conciseness and simplicity."

A point was raised by Mr. Weston which I had not overlooked, that of the importance, when purchasing coal, of an engineer carefully considering his boiler plant. This, of course, must never be neglected, yet I have to admit having seen many plants using unsuitable fuel when the sort which would have been much more economical, was all the time available.

Two points of distinct importance were mentioned by Mr. Laschinger. One with regard to the physical properties of coal, and the other the loss of heat which ought to be generated by the gases. As to the former there are undoubtedly physical attributes to certain coals which are not revealed in laboratory tests. The only comment one can make, from the purely practical side, is to fall back on the trite but sensible adage "there's no test like trial." As to the second, it must always be remembered that coal gases require for their explosion an admixture of air, that the best mixture is probably 10 or 12 of air to one of gas. Either too much air or too little takes from the effectiveness of the burning. Thus a careful engineer will see to the admission of as nearly as possible the correct amount of air into his boiler furnace. Yet how many are there who, changing over from one coal to another, make a point of modifying the air admission?

FAULTING PHENOMENA IN RAND MINES.

(*Read at February Meeting, 1911.*)

By G. H. Beatty (Member).

REPLY TO DISCUSSION.

Mr. G. H. Beatty (*Member*): The paper has failed, probably on its merits, in its primary object, namely to promote a discussion. Like most mining papers it has been received with an apathy, which fortunately is not shared by the metallurgical section of the Society. Mr. E. Pam says, on p. 370, "that owing to the large amount of development work done, compilation plans of districts, and improved methods of keeping records, every fault of importance can be overcome without difficulty." This is in the main true, but very many faults, large enough to upset the best intentioned development schemes, confine themselves to an area of a few claims. It is these smaller faults and the various methods of developing them, which are the cause of future mining and tramming troubles. The permanent works—shafts, main cross-cuts and drives—are beautifully laid out, but the developing drives, cross-cuts, winzes, etc., are congested. To use an analogy the arteries are in perfect order but work under a low load factor because the veins are sluggish. Mr. Coombe says, on p. 370, "that in the olden times, they knew almost to a certainty where to seek a lost reef by employing rule of thumb methods, that is by considering every fault to have a normal throw." Actually, I think, this method would be correct for 70% of the dip faults and about 50% of the strike faults. I am sure Mr. Coombe would not be satisfied if his reduction officer gave him a 70% extraction. I have often admired the work done by the old miners, in opening up a new property, but they too, with admittedly less to guide them, make many unnecessary mistakes. I do not agree with Mr. Tom Johnson when he says, on p. 556, "the one term downthrow would suffice if the direction in which the fault did downthrow was mentioned." The better and more correct method would be to state the nature of the fault (reverse or normal) and give its dip. When a fault is struck the working place should *always* be surveyed and the prospecting cross-cut driven on lines. Mr. Johnson says "the small faults do not always have the same dip or throw the same way as the parent fault" but the small faults he mentions (shown on the sketch on p. 368) as an exception, are not in any way sympathetic or contemporaneous with fault DF2. In my paper I said (p. 367) "almost every fault will be found to have an increasing or diminishing throw, ultimately dying out in both directions." I think it is clear that I meant, in the case of an increasing throw, the fault would, after reaching its maximum, diminish down to zero. Mr. Johnson wants to know why there are no corresponding dip faults on the underlap side of the dyke. So do I. I expected to find this out during the discussion. I agree with him that probably the westerly movement accounts for it. It will be noticed from the sketch (p. 368) that the dip faults are purely local, diminishing very rapidly, thus possibly the huge dyke which caused the overlap and westerly movement, may have destroyed that portion of the sedimentary rocks, which contained their northern origin.

The Chairman: I hope the mining members will take to heart the preliminary remarks in this reply.

SOME ASPECTS OF MINE VENTILATION AND EARTH TEMPERATURES.

By J. Whitehouse, B.Sc., M.I.M.E. (Associate) and W. L. Wotherspoon.

Mr. Tom Johnson (*Member of Council*): The authors give a chart showing the rate of increase of temperature of the rocks in depth, but the text does not agree with this chart.

Marriott's line starts at 200 ft. with 65°F and finishes at 7,000 ft. with 98°F., showing a geo-thermic gradient of 206 ft. not 208 ft. Wotherspoon's line starts at 200 ft with 66°F and finishes at 7,000 ft. with 92·5°F. showing a geo-thermic gradient of 256·6 ft. not 253·9 ft. I assume the starting point to be at the depth of 200 ft. which I expect is the invariable line.

The air temperature in No. 1 shaft has been lowered by 7·9°F. and in No. 2 shaft 8·3°F. This is very good as showing how large quantities of air, circulating through the shafts and drives help, and it points the way to what has to be done in the deeper mines. The rock temperature from the third to twelfth levels has been reduced 3·2°F., but at the same time most of the development work on the main reef leader and the whole of the south reef is as bad as before; it is the development work that must be improved. Would the authors give us the air temperatures, which I think should show up a little better? It is a little strange that the rock temperature does not show up better in the development work, if, as I assume, the same holes were used for the tests before and after the fans started. Why should the percentage of CO_2 go up when the machines have all got started to work? One would expect the opposite to take place.

On page 545, third paragraph, do the authors not make a mistake in saying that "the temperature must be taken from the downcast to be that experienced on the surface during summer?" If this is not a mistake, would the authors kindly show us what amount of air we could expect to be put in circulation under the conditions stated on page 536 where they show a surface temperature (that is a downcast temperature) of 84°F. and a mine temperature of 76·5°F?

In table E, I note that on three occasions the relative humidity increased with an increased temperature. Might not this be due to exhaust steam from the hoisting engines being blown across the post of observation?

On Fig. 6 the Sirocco fan shows a remarkable manometrical efficiency being about 96% to 97% of Murjuc's theoretical, which I think is like some other things too good to be true. Figures of fan tests are only relatively true when used for the same fan running under same conditions—so much depends also on the water gauge, and a water gauge tells many different tales. Are the costs shown based on the Village Deep tonnage alone? It must be remembered that the fan is ventilating the Village Main Reef Mine also. Will the authors give us the manometrical efficiency of the fans tested, as shown in Fig. 5? What are the advantages of placing the fan underground at the Village Deep, and could not the fan have been placed somewhere near the outcrop, and the Village Main Reef Mine ventilated with fresh air from its own shafts, instead of with the return air from the Village Deep? How much of the W. G. is used on each of the mines, and how much of the W. G. is due to the natural ventilation?

Why do the authors tell us that it is desirable that the air should be split into small circuits in order to give each district a cool supply and then show us Fig. 7; is it consistent? would it not be better to have another road through the dyke and put the fan in another place?

The tables A. B. C. D. E. being local observations are of great value.

The geo-thermic chart shows us there is a possibility of being able to mine at 10,000 ft. to 12,000 ft. vertical, as far as temperature is concerned.

FUTURE ECONOMIES ON RAND PRODUCTION PLANTS.

(*Read at April Meeting, 1911.*)

By C. O. Schmitt (Member).

The following contributions to the discussion on this paper were made by members of this Society and the Institution of Mining and Metallurgy at a meeting held in London on May 31st, 1911, and are printed herein by arrangement.

DISCUSSION.

Mr. R. E. Commans having briefly introduced the paper,

Mr. A. C. Claudet said that, in order to be able to follow a paper of this description, the assay values of the gold in the waste rock should have been given. The paper should, in fact, contain such data as would permit a business man, who was not necessarily a mining man, to check the figures for himself and to study in an intelligent fashion the comparative gains in reduction costs obtained by sorting out more or less rock. The author did not state how much gold was contained in the waste rock in any of the cases specified. It was obvious that if 30% of waste rock were separated, instead of 16% the working costs would be reduced; but with what loss of gold, who could say? The paper was full of data, but it seemed to him (the speaker) that some of the essential figures required for a proper understanding of the question were omitted.

Mr. Hugh F. Marriott said that the author had given them a paper overflowing with data and ideas, and covering an era of time from the very commencement of operations on the mineral deposits of the Rand to the eventual cessation of the industry. The title of the paper by no means indicated the wide scope of the subject matter, and it would perhaps have been better had the author confined himself to that portion of the estimated economies of the future which related to the treatment of the ore after it had emerged from the mouth of the shaft and up to the point where it was handed over to the cyanide works. As it was, he had included a prognosis which embraced all the vital principles in the industry, and the statements in this respect called for far more serious criticism than those portions of his paper which dealt more particularly with the reduction plants.

Mr. Schmitt led off with an introductory statement which would single out the mines of the Witwatersrand as the special delight of a pessimist. It certainly was an acknowledged fact—which was unnecessary of emphasis, as it was a platitude—that when working costs equalled the values recovered there was no profit, and if there were then no further prospect in favour of the decrease of the former or an increase of the latter, there was no object in continuing the particular business in hand, But that this particular point should be emphasised in regard to the Rand to-day was erroneous, and was chiefly so for a reason which had been already brought forward in other recent discussions in connection with Rand gold mining, viz., that the basis of comparison was unstable.

The author introduced a further complication into his argument at the very outset, when he used the combination of "working costs *plus* capital charges." The capital charges, apparently, were those which it was the custom of the Rand to segregate from what were called the mine working costs, and to deduct from the total profit at the end of each year. Those capital charges were, in reality, working costs, for without them the mine could not continue to be kept in a state of normal efficiency. This, however, was by the way as regarded the main question that arose on the first page of the paper.

To show that the basis of comparison between the earlier returns from the Rand and those of the present day was erroneous, was a simple task. In the first instance, the values that were obtained chiefly from the Central Rand in the first few hundred feet from the outcrop must be ignored for the purposes of a general argument. The first developed ground in any goldfield was usually the richest. It was, as a rule, the richest portion of the outcrop that attracted attention and was first exploited. In the earlier days people were after as much gold as they could get. Their standard of selection was what would to-day be called "picking the eyes out of a mine." Owing to the higher costs of working, they left behind ore which, when extracted to-day, was often materially richer than the general average of what was taken as pay-ore in the present developments. They took no account and made no record of the large quantities of lower-grade rock which were to-day valued, measured up, and included in published ore reserves. Some good many years ago the speaker had occasion to measure up that portion of the Ferreira Gold Mining Company which was stated to be entirely worked out. He found that 50% of the reef was still left in the mine unaccounted for. This had since been included in the payable reserves of the mine.

It could readily be understood, therefore, that many a mine to-day would give an utterly different presentation of the grade of its ore reserve if these earlier methods were followed and returns were made on the same basis. The result would certainly be, in many cases, to arrive at valuations of the richest sections quite as high as those which were quoted in evidence of former glories. It was chiefly the most central portion of the Rand which was responsible for all this fiction, and one or two richer spots in the east and west extensions, which were also opened out in the earlier days more rapidly than the poorer sections, added their weight.

To take the case as it really stood, attention must be confined to one unit only, and other variations must be left out of the question or reduced to a constant. The values of actual banket were the point under discussion, and they had therefore nothing to do with varying widths worked or tonnages developed in following up this point. On the outcrop of the Robinson, Ferreira, and Wemmer, as had been already stated, there was abnormal richness, but, when the first few hundred feet of the upper levels were eliminated, there was little to choose in actual reef values between the lower sections of these mines, including their immediate neighbour, the Ferreira Deep, and the actual reef values which had been demonstrated by the huge workings considerably to the dip of these three properties in the Robinson Deep. Further to the west, the past record of the Crown Deep had proved a stretch of ground actually richer than its neighbour on the outcrop, and, to the east, the central portion of the Village Main Reef on both reef systems had shown its reef values to be of considerably higher grade than the workings immediately above it or on either side. Further east, the City Deep had opened out a large expanse of rich ore which, if the haphazard chance of tilting and denudation had placed it on the horizon of the outcrop, would have been the actual centre of the Rand, and Johannesburg would probably have been located further to the south-east, in the region of the Natal Spruit.

He (the speaker) would only go so far in these comparisons as would suffice to show that, as far as reef values were concerned, there was no substantial evidence to-day, even for the most confirmed pessimist, to prove a general deterioration. His opinion in this respect was not merely advanced as a benevolent idea, but was based on the most complete evidence it was possible to secure, that of the assay values of samples of actual reef widths taken throughout all the mines and correlated. The richest portions near the outcrop were naturally opened up first, and some day these must inevitably expand into a poorer district. On the other hand, poorer districts which had been shown as such at the outcrop would also pass on further development into richer patches, the extent of which could only be known after the development had conclusively shown their boundaries. It should be thoroughly understood that at the outcrop it was possible to pick and choose which sections of the reef should be opened out, and this had been done to the greater exploitation of the richer sections and the partial exclusion of the low-grade patches. The deep level properties, however, had no such privilege. They had only had the results from the companies above them to draw upon for their estimates of value, and experience had shown that this connection was often misleading.

If any rule at all could be given to-day for Rand values, it was that the different grades occurred in patches of irregular form and extent, and anyone district might be said to have a set value only as a general average and not in detail. It followed, therefore, that the opening out of a deep level property was much more of a speculation than the selection of an outcrop, and if an abnormal grade were first met with there was no other course than to drive east and west until its lateral extent was proved. In the deeper levels the poor patches thus exposed went into the estimates of general values for the depth concerned, whereas, where the lowest values had been opened up at the outcrop, the ground had been left alone and had not been included in general estimates of grade.

To come back to the argument of unsound bases of comparison, it was human agency, not alone reef values, which had provided these decreasing valuations per ton milled on which the author laid so much stress. To take the case of a mine to-day as against that of the early "nineties," the high-grade portions of a mine to-day which were analogous to the selected ore worked in the early days were only a part of the ore reserves. In addition to these blocks, there was also included a huge amount of ore of a value which used not, formerly, to be taken into account at all, and all rock was included down to the pay limit as decided by the estimate of the mine working costs of to-day. The comparison might be considered to be sufficiently nullified by this variation of principle, but the argument was still further clinched when it was pointed out that not only was the high-grade portion of to-day's mine modified by the inclusion of the lower-grade rock immediately associated with it, but in many cases there were also added to this the reserves of low-grade rock which were still left over from those portions of the mines which were originally only estimated on the selected value.

History had shown that this continual and vast encroachment into the lower-grade natural reserves of the deposit had, up to a point, greatly increased the prosperity of the industry, and the main point which confronted the industry to-day was to take care that human agency did not artificially deteriorate the values from the degree of concentration in which they were found more than was absolutely necessary, and that there should be properly applied the most scientific mining that the talent of the world could to-day produce.

This paper was termed "Future Economies in Rand Reduction Plants," but before a discussion of methods of treatment was entered into, it raised a question of much further-reaching importance.

The author prefaced his remarks on the plant in question by stating that, in order to get below the comparatively shallow depth of 5,000 ft., it would be necessary to reduce operating costs and capital charges. He (the speaker) would say rather that it would be necessary to use greater discrimination as to what rock was chosen for the mill and what was left behind, and, after the greatest care possible had been exercised in that respect, and the standard for each mine as to the lowest value that really assisted in paying a dividend had been decided, then the question of reducing operating costs was of next importance, so that still more of the developed ground could be included in the dividend-paying list.

It still held good to-day, as it had done in the past, that where pence could be saved on the surface shillings could be saved below ground, or rather, the extra profit that could be obtained by more economical methods was much greater; and this would always be so, for, while the work done in the surface treatment plants could be made to follow a regular routine, and efficiency could be measured by power and weight meters, and by assay results and gold won, the efficiency below ground depended on that ever-changing factor, the human element, which too often commenced where the previous unit commenced, and not, as in the case of machinery, where the replaced unit left off.

The author assumed, on p. 466, that mining costs would go up. This was a statement which, like his previous dictum regarding the values, could not be passed without direct challenge. All the most recently designed improvements had been arranged with the special object of ensuring that mining costs should go down, and that such items as transporting the rock from the stope face to the level among the main haulage drives and up the shafts should show a higher state of efficiency, and not be subject to excessive costs on account of the many periods of idleness which were sandwiched in among the operations, and so made for uneconomical standing charges.

To take the author's items as enumerated :—

Increased Cost of Skilled and Unskilled Labour.

—As South Africa became a more fully inhabited country, and the local population grew more and more to consider mining as a normal occupation, the altogether abnormal prices that had obtained in the past in the effort to get and retain the limited number of skilled men available should cease. Unskilled labour was rapidly being replaced where it was uneconomical by mechanical methods, and there was no doubt that this would, before long, reach a position in which the balance could be adjusted should the demands of unskilled work rise unduly.

Ventilation.—The last part of the labour paragraph was replied to under the third heading, ventilation. The outcrop mines did not provide artificial ventilation, but there was no doubt many of them would have been much better off had they done so. The deeper mines, which were now going in for fan installations wholeheartedly, did this with the knowledge and assurance that they would be more than repaid for the cost of the outlay in the saving by improved conditions underground resulting in a greater and legitimately expected amount of willing labour from underground employes. Those who knew the class of work and the cost of labour in the non-ventilated mines would not hesitate to say that mechanical ventilation was no drawback to economical mining.

Mine Drainage.—Most of the water was caught in the upper workings, and, in view of this continuing to be, many mines would have to introduce water to their lower workings for purposes connected with mining.

Hoisting Rock to Surface.—As the workings got deeper, hoisting required increased cost in ropes and increased power per unit hoisted, but it must be pointed out that the increase in the power charges resulted in the middle of the shaft, where the greatest speed was attained and the most economical hoisting was done, the starting and stopping costs remaining constant.

Vertical shafts had now probably reached the limit of depth to which they would be designed or sunk, and the subsidiary inclines which would feed these in the lower levels were charges against the development which they opened out, and would be run at no greater expense than in the mines nearer the outcrop. Indeed, they would probably be on more economical lines, as their work would be more systematised than it had been in the past.

Underground transport along the levels would go over greater distances, but mechanical haulages now being designed and installed should show a greater saving in expense over the handtramming system than any other item underground.

Power and hoisting had already been dealt with as being over-rated.

Less air would be required for drilling the same footage of holes in the rock, as that to be supplied in the future would be of greater pressure and would give a much greater efficiency per unit of air used than could be indicated by terms of simple proportion.

The theory of yield shown in item 7, Table II., on p. 467, the speaker has already disposed of. As regards the other item calling for attention, No. 2, mining costs, the author had not justified his estimate of the increase from evidence given. Important reductions in various items of underground mining should be looked for in the future, where it was well known to-day there was room for better organisation of labour and more economical application of the power employed.

The author, at the bottom of p. 467, realised the temptations that enticed those responsible for underground work, when they were judged on the basis of the ton milled, and it was gratifying that this was readily admitted. Until those responsible for mining were judged by the relative profit they had obtained from the extent of the company's property they had depleted, this danger must always remain alive.

The epithet of "probable loss" on the top of p. 468, would read better as "undoubted absolute loss."

As to that portion of the paper which was legitimately included under the title, the speaker preferred to leave criticism to those who had themselves been doing recent work on these lines. He would only point out, as a general principle, that whatever economy could be effected in removing waste or barren rock after it had been delivered into the headgear at the surface, this amount could be increased to a greater extent if much of the rock so arranged to be sorted out were either left standing underground where nature had placed it, or where broken, removed and restored underground as an operation separate from the recovery of profitable ore.

One point on p. 475 should have attention drawn to it, as showing the limitations of present-day discussions. The writer stated that the tonnage milled did not readily express the work done anywhere but in the stamp mill. This should be taken to heart, and remedied.

Another point on p. 478 should not pass without remark, although in this he (the speaker) was speaking without definite knowledge, and only on information afforded by others. He understood that there were plants working on gold, considerably more difficult of extraction than that of the Rand, where the introduction of tube mills had been attended by a previous thickening of the pulp and the addition of free cyanide at the head of the tube mill. It had been demonstrated satisfactorily to those responsible for this system that the gold in the ore passed into solution so readily during its normal course through the tube mill that there was not enough left undissolved to make it worth while to continue any subsequent treatment. This, he believed, had not yet been demonstrated on a practical scale on the Rand, but, if it should prove only partially correct, and it were found possible to effect satisfactorily the rearrangement of solution circuits, it would eventually lead, not only to a modification of our great cyanide plants, but probably to a tandem tube mill, and the doing away with cyanide plants altogether.

The author concluded his paper, on p. 491, by again stating that the great problem was the reduction of working costs in order to increase the lives of the mines of the Witwatersrand. There was no value in length of life when it was obtained by reduction of profit. He (the speaker) would again say rather that the problem was to supply to the reduction plants only that ore which it was undoubtedly profitable to treat, and to reduce the cost of treating that ore as far as possible so that more ore standing below would be available for treatment. But the main point should be to ensure that to-day the profit that the mine was legitimately capable of producing was obtained.

Mr. H. S. Denny said that the author was to be congratulated on his exhaustive and painstaking contribution to the literature dealing with the Rand gold industry. His statements covered a lot of ground and contained much carefully compiled information that must prove valuable to those interested, and it was refreshing to note that he had not hesitated to launch out into a discussion of some of the more delicate and controversial problems that were bound up in the subject which he had chosen, although he might find, as others had done in the past, that there was little honour or profit to be derived from such an attitude.

In the speaker's opinion the value of the paper lay rather in its statistical than its discursive features. In the compilation of many tabulated statements reflecting the essential characteristics of single-stage and double-stage crushing as it actually was to day, the author had done some most excellent work and his figures should give an impetus to the movement in favour of multiple-stage reduction.

Unfortunately, however, the author has allowed himself to wander away from his excellent marshalling and discussion of facts into fields of speculative fancy, where he not only did himself an injustice, but incidentally aimed a blow at the absolute foundations of the Rand industry as was shown in his remarks (*a*) on working costs: their reduction in recent years and their future; and (*b*) falling off of grade in the deep level mines. The speaker proposed to confine himself to those two sections of the paper.

The author believed that the splendid reduction in the average operating costs witnessed in the past five years had been brought about by improvements under the head of mining. Unfortunately he did not specify in detail the bases of this assumption, and the speaker was rather inclined to believe that he would eventually find that the real source of the improvement lay in other directions. It did not seem possible to lay a finger on any one point and credit it with the whole improvement, because there had undoubtedly been a general as well as a particular improvement, but it was probable that the increased duty of crushing plants must find a very important place in this consideration.

The author further believed that working costs were more likely to increase than to decrease as the workings become deeper, and naturally this would be so if all the conditions of working remained unaltered; but judging by the experience of the past, the difficulties in the one respect were more than counterbalanced by improvements in the other. In fact, it was extraordinary, but none the less true, that working costs, by some mysterious process, seemed to level down to meet the demands of production. Every mining district, apparently, had to begin with high grade ore and high costs, and yet they nearly all in their final stages reached the opposite condition. If only the cost factor could have been dealt with in the earlier stages of the history of most mining camps, what wonderfully enhanced profits would have resulted!

It was difficult to account for the fact that the decrease in cost came only when there was no alternative, and when it must be either that or a "shut down." It seemed impossible to find any other factor, either in the shape of an incentive or a spur, to reach the same result earlier, and it was only when every official and employe, from the managing director down to the unskilled labourer, were all striving in unison for the one object, that these really big reductions were ever achieved. Apparently it was only when the greatest danger was threatened that this unity of purpose could be unearthed. It seemed a pity, but it also seemed inevitable, that this good work must wait till its real brilliance was obscured in the downfall of the grade of the ore. And here they came to the author's second point, namely, the grade.

Few people would be ready to accept the figures submitted in Table I., in regard to which the author stated: "It appears reasonable to estimate them at . . ." He (the speaker) had been too long away from the Rand to take the responsibility of replying to this assumption, but he would suggest that it was rather too serious a statement to pass unchallenged, and accordingly he would be as interested, as doubtless the author would be, in reading the reply. There were some aspects of the case that had a direct bearing upon this question of grade to which he proposed to refer later. The future of the Rand was every year more and more confined to the deep levels, and as the author estimated that the average from which rock was at present being hoisted was 2,400 ft., he was tacitly stating that the industry was on the second leg and down grade of its existence.

To hoist 17s. rock from a depth of 5000 ft. was not an alluring prospect, even though the working costs were by almost superhuman effort reduced appreciably below their present standard. Every reduction became in turn more difficult. seeing that the margin for such reduction must of necessity be a limited and decreasing one. Whilst it was safe to say that the average grade of the ore developed on the Rand has shown a decrease as between the outcrop and the lowest existing working depth, the degree of that decrease had been exaggerated by the process of high grade ore selection so generally adopted in the first decade of the Rand's history; that was to say, that an unfairly large percentage of the higher grade ore having been milled in the period referred to, there was left a correspondingly abnormal proportion of the lower grade ore to be included in current statements of ore reserves, and the inclusion of these blocks from the older areas of development had naturally reduced the standard of the total development.

Furthermore, the demand for larger daily tonnage outputs and the increased facilities that had grown for the handling of this larger tonnage, had brought into the category of payable ore large blocks that under previous standards of operating costs and earlier conditions of working were regarded as quite unpayable. And that this had been a steady process was well illustrated in the following example, the figures of which were extracted from the published reports of one of the leading companies:—

Ore Reserve.

Year.	Tons.	Value.
1905	437,074	8 dwt. or over.
1906	627,880	6 ,, ,,
1907	910,200	5 ,, ,,
1908	1,087,760	4·5 ,, ,,
1909	1,419,430	4 ,, ,,
1910	1,590,711	4 ,, ,,

It is at once evident that accumulations of what was considered unpayable ore in 1905 had, in the natural order of things and by virtue of the large decrease in working costs, at last come inside the boundaries of payability, and the ore reserve statement of to-day was gaining a large tonnage, the development cost of which was met several years back. The development redemption account was naturally reduced from a large figure per ton obtaining under the restricted statement of former years to a mere cipher under the new conditions; but, and this was an important point, it must not be forgotten that the inclusion of these large blocks of comparatively low grade ore, which formed the preponderating proportion of the present statement, must have a diluent effect on the statement of average values of the past three years' actual development. This dilution might even go so far as to create the impression that a serious falling off of grade had occurred in a mine where in reality the current grade had been quite equal to or even considerably higher than in the previous year.

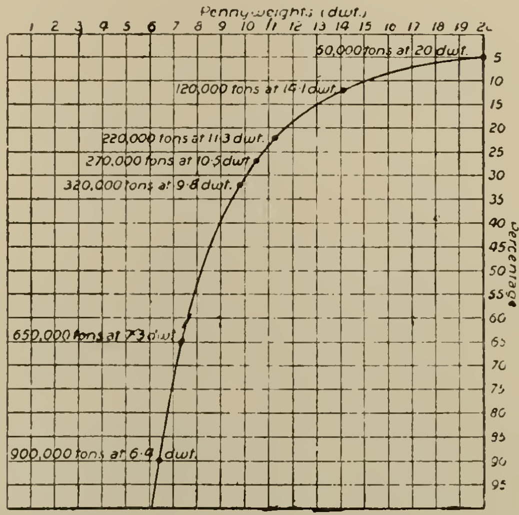


Fig. 1.—This diagram represents one of the blocks of 1,000,000 tons, and illustrates the value of any given percentage, and *vice versa*. It illustrates clearly at every stage the effect of the inclusion of large blocks of low grade ore.

For instance, take the hypothetical case shown in the attached table and in the accompanying diagram, Fig. 1.

Explanatory Notes to attached Table.

1. There are four blocks representing the first, second, third and fourth rows of deep levels.
2. Each block is fully developed in turn in a period.
3. Each block has the same ore content.

	Tons.	Dwt.	
Block A 1st row deep levels.	50,000 at 70,000 ,, 100,000 ,, 50,000 ,, 50,000 ,, 330,000 ,, 250,000 ,, 100,000 less than	20 10 8 7 6 5 4 4	FIRST PERIOD. 1,000,000 tons developed. 120,000 ,, crushed } 50,000 at 20 dwt. 70,000 ,, 10 ,, Average milling value, 14·1 dwt. 100,000 tons of ore to reserve, consisting of 100,000 tons at 8 dwt.
Block B 2nd row deep levels.	Do.		SECOND PERIOD. 1,000,000 tons developed. 120,000 ,, crushed } 50,000 tons at 20 dwt. 70,000 ,, 10 ,, Average milling value 14·1 dwt. Added to ore reserve 100,000 tons consisting of :— 100,000 tons of 8 dwt. Brought forward .. 100,000 ,, TOTAL ... 200,000 ,,
Block C 3rd row deep levels.	Do.		THIRD PERIOD. 1,000,000 tons developed. 240,000 ,, crushed } 50,000 tons at 20 dwt. 70,000 ,, 10 ,, 120,000 from ore reserve at 7·25 dwt. Average milling grade 10·06 dwt. Ore reserve brought forward *Add* 200,000 tons of 8 dwt. ,, 100,000 ,, 8 ,, ,, 150,000 ,, 7 ,, ,, 150,000 ,, 6 ,, 600,000 ,, 7·25 dwt. *Less* 120,000 drawn. Leaving 480,000 tons of 7·25 dwt. *Add* 330,000 ,, 5 ,, from 1st period. ,, 330,000 ,, 5 ,, ,, 2nd ,, ,, 330,000 ,, 5 ,, ,, 3rd ,, TOTAL 1,470,000 average 5·8 dwt.
Block D 4th row deep levels.	Do.		FOURTH PERIOD. 1,000,000 tons developed. 480,000 tons crushed } 50,000 tons at 20 dwt. 70,000 ,, 10 ,, 100,000 ,, 8 ,, 50,000 ,, 7 ,, 50,000 ,, 6 ,, 160,000 ,, 5·8 ,, from ore reserves. Average milling grade 8·4 dwt. Ore reserve brought forward 1,470,000 tons of 5·8 dwt. *Less* 160,000 1,310,000 at 5·8 dwt. *Add* 330,000 ,, 5 ,, 1,000,000 ,, 4 ,, TOTAL ... 2,640,000 ,, 5 ,, Tons milled up to end of 4th period : 120,000 120,000 240,000 480,000 960,000 tons milled. Ore reserves ... 2,640,000 at 5 dwt. TOTAL ... 3,600,000 tons, or 90% of the total tonnage [developed. Average value of the 3,600,000 tons = 6·4 dwt.

4. In the first two periods the milling basis, values, ore to reserve, remain the same.

5. In the third period the milling capacity is doubled and the standard of ore reserve is lowered from 6 dwt. to 5 dwt.

6. In the fourth period everything at 4 dwt. and over is taken into the ore reserve.

7. The rich ore in each block (20 and 10 dwt.) is assumed to be entirely exhausted in each period.

8. At the end of the fourth period the mine is left with 2,640,000 tons of 5 dwt. ore, all developed and paid for, equivalent to a further 5½ period life.

9. If the policy followed in the first year had been maintained the mine at the end of the fourth period would have crushed altogether 480,000 tons, and the grade would have remained constant throughout.

10. It is assumed that by increasing the milling capacity the working costs have been reduced to a point where the inclusion of the poorer rock in the ore reserve is justified.

11. As between the first policy and the final policy respectively, the tonnage crushed and the gold recovered would be as follows :—

(*a*) 480,000 tons at 14·6 dwt.
400,000 ,, 8 ,, (ore reserve)
———————
880,000 tons yielding 500,000 oz. fine gold
(*b*) 3,600,000 ,, ,, 1,060,000 ,, ,,
(This tonnage includes all the ore down to and including the 4 dwt. grade).

It could be seen at a glance that a very extraordinary falling off in grade was shown both in the average screen assay and the ore reserve value after the second period, although as a matter of fact the grade of ore developed in each period was identically the same, and, in the speaker's opinion, it was quite certain that on the Rand it was to this feature that one must look for the explanation of a large percentage of the drop in the value of the milling ore and the ore reserves. The real drop in grade—and it was generally conceded that there was such a drop—was nothing like so extensive as would appear from the statement of ore reserves. It was due, on the contrary, to the automatic inclusion of tonnage that had become payable as the standard of working costs had been lowered. Naturally, in the earlier stages, the mine was being depleted of too big a percentage of its higher values all the time, but in the illustration before them there would be no drop in the grade if the same policy of selection were followed with each successive block and in each successive period. It was only because the lower grade ore left in the first block had been carried forward to the second, and the first and second to the third, and so on, that the drop occurred, but then it must not be forgotten that the tonnage of the mine, and therefore its life, had been increased enormously by this policy. There were many other questions that arose in considering this illustration, such as the total gold recovered from the four blocks under the comparative schemes of milling only the high grade ore or gradually taking in everything down to 4 dwt., which it would be interesting to pursue, as also the question of total profit to be gained by the alternative policies, but the speaker feared that it would be going too far afield to take up such questions at this juncture.

To what degree the adoption of the principle of the inclusion of the low grade ore had affected the statement of average grade on the Rand as between the outcrop and the fourth row of deep levels he was not prepared to even guess, but he thought it was quite clear that the difference was much exaggerated if one attempted to assess it merely on comparison of present day statements, both in regard to milling returns and ore reserves, with statements of preceding years; hence the falling off was not to be decried in the spirit adopted by some too pessimistic friends or malicious critics. If, for the reasons of economy or other equally potent reasons, it were found expedient to handle lower-grade rock to-day than was thought possible some years ago, that fact must not in itself be interpreted as a sinister indication that the grade was falling utterly away. In other words, had it not been for a change of policy in the illustrations instanced, the mine could have been worked from outcrop to its lowest boundary at a constant grade, but for those very reasons it had been found more profitable to take in gradually a lower grade of ore, offsetting the reduction in grade to a decrease in working costs, and thus ensuring to the mine a larger profit over a longer period than would have been consistent with the continuance of the first policy.

Naturally, he did not claim that this happy result would always be realised. There were included in the proper working of such a policy a multitude of nice calculations, upon the accuracy of which would depend their intrinsic merit. It was not unreasonable to suppose that those of a too optimistic tendency would include in the statement of ore reserves large tonnages that would never really come within the limits of profitable treatment, and it was possible (though not probable) that there would be others of a too pessimistic tendency who might err on the side of caution. He knew of many such cases in his own experience.

Again, it must not be forgotten that in the illustration taken the grade of ore *in situ* was just the same in the fourth row as in the first, and this assumption would in almost every case prove to be too favourable a basis. There were some mines notoriously rich down to 1,000 ft., which in the next thousand feet were very much poorer, and in the third thousand feet descended to a dangerously low average. Almost every mine had its own peculiarities in this respect, and must be judged accordingly.

Mr. S. J. Truscott said that in this paper the author had allowed himself to speak a good deal about the grade of the ore, and on the first page he gave a statement from which one would gather that the grade was a factor of depth. It was true that he somewhat qualified this in a footnote, but since, however, it was repeated again several times in the body of the paper, the ordinary person after reading it, would certainly point to this paper as evidence of the unqualified decrease of value of the banket in depth. The author, in spite of the above-mentioned footnote, regarded depth and value as being so interdependent, that he was able on pp. 465 and 486 to find the average depth of the workings at a particular date by ascertaining the grade of ore at that date.

As a starting point for his deductions he referred to a paper written by Ross E. Browne in 1905, at which date 1,000,000 tons of ore, having a grade of about 35s., were being crushed monthly, whereas now the figures were about 1,800,000 tons at 28s. He implied that because, since that earlier date, the average depth of workings had become deeper, the value consequently was lower. But since that date the expansion had not only been in depth, but had also taken place to the east and to the west. To the west the Bantjes Consolidated had started, yielding about 25s. per ton; the West Rand Consolidated, yielding about the same; the Roodepoort United had increased its plant and was now crushing ore yielding about 21s.; the Randfontein Central had also recently added greatly to the previous figures from that group of mines, and the average value was under 30s. These were all outcrop companies which had started since 1905, and the average yield of the whole lot was only about 25s. Within the same period there had been several deep levels started, such, for instance, as the Geduld, Knight Central, Simmer Deep, Jupiter, City Deep, Main Reef West, and the average yield of these was higher than that of the above-mentioned outcrops.

These actual cases showed that expansion was a factor of economy and not of depth, and if reference were made to the expressed opinions of authorities upon this same subject it would be found that they were in accordance with these actual cases. Thus Ross E. Browne, in the paper referred to so often by the author, said:—

"The fact must be recognised that whereas the Witwatersrand contains, in a comparatively small territory, a greater quantity of gold than ever known to exist elsewhere in the same area, the average grade of the ore is quite low. In the first the higher-grade ores were particularly sought and the yield per ton is naturally and inevitably diminishing . . . In the average operating mine on the Rand the masses of ore yielding less than 22s. or 23s. must now be left behind."

Hennen Jennings, as spokesman for a committee of engineers, said in the statement put before Mr. Chamberlain and Lord Milner in 1902:—

"It may be confidently stated that there are no general unfavourable indications regarding future yield at present in sight; still it must be remembered that, as working costs are reduced and the scale of operations extended, a lower yield per ton must naturally be expected, for it is seen that there is an almost indefinite amount of conglomerate ore, and as working conditions become more favourable so will the working of more extensive amounts of low-grade ore be justified."

Due weight must be given to such statements as these.

And then, if comfort be sought from theoretical considerations, these were also at hand. In a recent work upon ore deposits written by Beyschlag, Krusch and Vogt, the opinion, to which the speaker also subscribed, was given that the gold was in the banket beds before they were raised. In such a case the primary deposition of the gold in this banket could have no relation to depth, but after the tilting of the beds to their present inclined position there was room and occasion for the working of atmospheric agency to produce secondary enrichment in the first few hundred feet; beyond that, however, there did not appear to be justification for the statement that there was depreciation of the value in depth. There were, indeed, cases of pronounced opposite tendency, as, for instance, the Wit Deep below the Witwatersrand and the Ginsberg, the Main Reef West below the Unified and Aurora. In one and the same mine also there were cases of better value being found in the deeper workings, as, for instance, in the Village Deep, where the values struck in the Turf Mine's shaft at a depth of approximately 4,000 ft. were better than those in the upper levels.

It was in one way especially fortunate that Mr. Marriott had started the discussion, because he belonged to a school and a House which was not pessimistically inclined, but which recently showed its confidence by taking over from the

Consolidated Goldfields of South Africa the large area known as the Rand Mines Deep, which was situated to the dip of the Simmer Deep and the Jupiter: these two latter mines showing, at present, ore of a low value.

Leaving the question of value, the author turned to that of cost, and said that mining costs must increase in depth, giving in support of this statement a long list of contributory causes for this increase. But from the beginning many of these causes had been operative, and yet in spite of them the costs had regularly decreased to this date, and one was therefore justified in believing that there would be further reductions.

To say, for instance, that the cost of hoisting would increase was decidedly wrong; it would be right to say that the consumption of power in hoisting from greater depths would be greater, but as power is daily becoming cheaper, this greater consumption might well be accompanied by a smaller cost, and this, he believed, would be the case. It must be remembered that power would shortly be purchasable from a large supply corporation which could retail it at such a figure that it was estimated that it would make a difference saved of about 9d. per ton crushed.

Following the author, a reference might be made to the unit employed. The author referred to the unit of the "ton milled," and said that it was not a good one; but it must be remembered that this was the unit which was almost universally adopted in gold mining, and, by using it on the Rand, the mines there were ranging themselves in line with those in other fields. A very good reason for the adoption of this unit was that at the battery screens there was the first chance of getting a reliable sample of the ore which was being passed through the reduction works, and that was an important matter. In many ways the "ton hoisted" would make a better unit, and he (the speaker) thought that for the purposes of the logical handling of the questions raised in the body of the paper, this unit was best employed.

It must not be forgotton, however, that there were very few gold mines where sorting was so largely employed as on the Rand; in fact, few gold mines found it possible or expedient to sort at all. In such cases the "ton hoisted" was the same as the "ton milled," and if on the Rand these two had become separated, the fact that the "ton milled" carried with it a properly determined original value to the ore, had made it expedient to adopt that unit. If it were said that from the value of the ore passing through the battery, the value of the ton hoisted could be deduced by considerations of the amount and value of the waste sorted out, that would hardly be the case, because the determination of the value of the waste was most difficult to arrive at by the ordinary taking of samples. In fact, the only proper way to do this would be to put a large parcel of it through the battery. When such had been done it had always been found that the value so determined was considerably higher than the result determined from the usual regular samples.

Coming now to the principal portion of the paper, which dealt with the future possible economies in reduction plants, it would be seen that the author's suggestions depended chiefly upon two things, viz., sorting and screening. With regard to sorting, the great difficulty which would arise would be the value of the waste thrown out. With an average of 16% discarded at the present moment, there was an appreciable amount of gold also discarded. Should this proportion be increased to 30% it was more than likely that the gold discarded would be more than proportionately increased. But taking the 16% now sorted, it was probable that this had an average value of about 1·25 dwt. At this figure, and at 80% extraction, the recoverable amount of gold would be 4s. per ton. In the figures given before the Industrial Commission in 1907, it was stated that to redeem all the money which had been spent upon capital account for reduction works, surface and underground equipment, etc., would require about 4s. 3d. per ton milled. Suppose, then, that this waste instead of being dumped were crushed, the redemption of the necessary increase in the reduction works might be set down as about 1s. per ton. For the cost of reduction, seeing that the newest plants were working for 2s. 6d. per ton, it might be said that the extra running costs for this waste would be about 2s., making, with the redemption cost, a total of 3s. With a yield of 4s., stated above, there would therefore be room to crush this waste at a profit. He did not quote those figures necessarily as an advocate of non-sorting, but only to show that even at 16% the question of sorting was hardly closed, and to indicate that at 30% sorting there would be real danger of losing gold and profit. The case required even more careful consideration when it was remembered that in crushing the waste the cost of sorting would be obviated, and the cost of sorting down to ⅜ in., to bring the percentage up to 30%, would be no small sum.

With reference to the use of screening to separate out sizes which should pass direct to the crushing stage suited to them, and to leave the coarser stages free to deal with a greater quantity of coarser material, it was better, in order to appreciate the present position, to review the evolution of the past few years.

In the year 1905, ordinary battery screening was in use, and the stamp had no auxiliary. With the advance of research, however, it was found that a good deal of the material passing these screens was too coarse to allow a satisfactory cyanide extraction, and finer grinding was advocated in a campaign in which Mr. Denny took a prominent part. For this finer crushing tube mills were introduced, not at first to increase the capacity but to better the extraction. Then with the enormous quantities handled, the stamp mill of ordinary weight was found to be a small unit and heavier stamps were advocated. It was found with these that there was no greater mechanical efficiency obtained, but an economy in arrangement and in accessories ; but when they were associated with coarser screening there was a distinct advance also in mechanical efficiency. The discharge then, however, being so coarse, secondary crushing had to be adopted and double-stage reduction then became prevalent, this being realised by the use of more tube mills. But here again, according to the author's figures, there was no increase in mechanical efficiency of the double-stage over the single-stage ; the advantage was in the lower capital cost of the double-stage equipment and in the cheaper costs of its running.

This question of the mechanical efficiency was very interesting. On p. 486 the author gave the comparison between the two systems, and showed that in 1905 each ordinary stamp crushed 2·7 tons of material fine enough to give a satisfactory cyanide extraction, whereas in 1910, with double-stage reduction, the equivalent of an ordinary stamp crushed 2·8 tons of material to that fineness, practically an identical result. Now, in 1910, the stamps themselves with coarser mesh had an undoubtedly greater efficiency, that was to say, with coarser screens they crushed an amount of fine material not less than before and in addition, on account of the higher total duties, they prepared a large tonnage for the tube mill ; and therein was their greater efficiency. If, then, it were found that the mechanical efficiency of the whole double-stage equipment was not increased, this was equivalent to stating that the increased efficiency of the stamps had been offset by low efficiency in the tube mills, and this might well be the case because a larger proportion of work was now put upon the tube mill which was, mechanically speaking, a less efficient machine than the stamp.

The claims, however, of the author's suggestions for consideration depended on questions of economy in capital and working cost, and the gains in these factors the author worked out very logically with the help of a series of interesting diagrams, afterwards putting them in tabular form for greater convenience in comparison.

Leaving out of consideration for the present the suggestion of closer sorting, the Tables IVa and VIIIa provided a *resumé* of the monetary advantages to be expected. It would appear, however, that in calculating the working costs in Table IVa, an error had crept into the calculation, that was, in working out the total cost of sorting, the figure of cost for the waste discarded, 1s. per ton, had been multiplied by the tons of material which had been subjected to sorting.

Making correction for this, the saving in working cost by differentiation in sizes would average just under 5d. per ton. This was not an example of great promise, nor did reference to Table VIIIa show that the saving in capital cost would be very great. In any case, whatever the saving might be, it had to be discounted somewhat by the fact that the present practice of sampling the ore at the battery screens would have to be modified, and it was not clear where an equivalent opportunity would be obtained.

Mr. C. E. Hutton pointed out that the author, in the course of a somewhat general survey of some of the points dealing with the future of the Rand, arrived at the conclusion that it was absolutely necessary that operating costs and capital expenditure be reduced if the lives of the mines were to be prolonged for any lengthy period, and the general tenor of his paper was to show that working costs would be higher and capital expenditure greater.

The author found, furthermore, that with the increasing depths of the mines there was a decrease in the value of the ore deposits, and he finally concluded that the limit of profitable mining would be reached on the Rand at a vertical depth of 5,000 ft.

The figures given by him clearly showed the advances made in the period 1905-1911, and it was interesting to learn that in this comparatively short time working costs had fallen no less than 7s. per ton.

This extraordinary drop in working costs the author believed to have been brought about by reducing the cost of mining, but the speaker took an entirely different view for the reason of this reduction and he would go as far as to say, from his own personal study of this question, that he was absolutely satisfied that the chief reason lay in the adoption and installation of the tube mill, which was first introduced to the Rand by the General Mining and Finance Corporation.

As general manager of a mine running 160 stamps for a period which covered the adoption of the tube mill and the large reduction in costs referred to by the author, he was in a position to make a close study of this subject, and in his opinion there is no question on the point.

By the addition of tube mills to the old plants the tonnage crushed had been enormously increased, with the consequent result that standing charges had been practically cut in two and other charges had indirectly benefited.

In order to understand thoroughly what effect the installation of tube mills had on departmental costs it was necessary to have some knowledge of mine accountancy, as the connection between the two was otherwise somewhat difficult to follow. The previous speakers, Mr. Marriott and Mr. Denny, had in their criticisms indicated in which direction those reductions lay, but to express this clearly would involve entering into a great amount of detail, and such a statement would, he thought, be more suitably expressed by means of a separate contribution.

Mr. E. H. Johnson had proved recently that, with a ratio of one large tube mill to every ten stamps of 1,634 lb. each, he could, after providing for screening off the $\frac{3}{8}$ in. product prior to entering the mortar box, secure a return of 261 tons of ore crushed in 24 hours, of which 99·5% was -90, and it would appear therefore that they might look for a still further appreciable increase in the crushing capacity of existing mills.

He did not propose to dwell very long on this particular point, as he thought that everyone would agree that the extraordinary increase that had been obained in the crushing capacity of mills by the addition of tube mills had been obtained at a comparatively low cost, and, therefore the effect must necessarily represent a quite appreciable reduction in working costs.

This, of course, was only one of the advantages that the system of stage crushing and grinding had brought, for no less important were the increased recoveries of the metal values which had been secured.

The author believed that the results now being obtained under the combined system of crushing and tube milling gave the death blow to all single stage crushing schemes, and in this the speaker was entirely at one with him.

The elimination of the fines before the mortar box was reached was not a new idea,* but it had never been carried into practice on the scale which Mr. Johnson was adopting.

The author, he thought, would find that the main reduction in costs was largely attributable to the tube-mill and did not lie so much in the underground work.

The author was apparently a strong believer in the possibilities of close sorting, and if his assumptions were practicable, he (the speaker) would be inclined to agree with him, but he had yet to see a sorting plant where it could be hoped to sort economically 28% of the $+\frac{3}{8}$ in. portion of a -2 in. product; even supposing that it could be done by having an enormously long and slow travelling belt manned by a large army of sorters, he could not believe that the cost of the operation would ever be justified.

He was astonished, too, to notice that the author allowed no value for the waste rock, in which connection he stated: "And as no protest is made regarding the value of the waste dump it may be presumed that only waste is discarded."

The smaller the percentage of sorting the lower the value of the waste rock, but even taking the best cases the speaker was certain that no engineer would like to tie himself to the statement that his waste rock was worth less than $\frac{1}{2}$ dwt. per ton, while in instances where sorting had been carried to a high percentage the value of the discarded rock would in most cases be higher.

The late L I. Seymour erected several sorting houses—one at the Jumpers Deep—where the coarse rock, after being sorted on one table, was passed through breakers and over a grizzly, and then again on to another table; but the practice had, he understood, been abandoned as unprofitable.

The author wrote, "It can safely be assumed that mining costs will go up rather than decline," and he submitted eight reasons in support of the statement.

He, the speaker, had noted the author's arguments in each case, but he was not at all satisfied that his judgment on the points was to be taken as in any sense final, and for the purpose of more closely presenting his own views under these heads, he proposed to deal with them in the same order.

1. *Labour.*—The constant scarcity of native labour would probably result in machine work being brought to such a pitch of efficiency, that the cost per ton broken by machines in the future would compare favourably with the hand drilling costs of to-day.

This, of course, was mere assumption, but the progress made in the use of machines in the last ten years must certainly be considered as lending encouragement to such a hope.

The author thought that increased temperature in the lower workings would badly affect the output of the labourers. He (the speaker) could not find much to justify such a statement in the evidence available to-day from mines working at a depth well below 4,000 ft., such, for instance, as the Cinderella Deep, where all theories in regard to increases in temperature had fallen down.

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* *South African Mines, Commerce and Industries*, p. 745, Oct. 21st, 1905.

Moreover, the latest practice in artificial ventilation at the East Rand Proprietary Mines showed that some 600,000 cub. ft. of air per minute could be circulated at a cost of less than 1d. per ton milled. Under such conditions mining could be made as comfortable at 5,000 ft. as it was at 1,000 ft.

2. *Explosives.*—Under this head there was not very much to be said, as the price of explosives might fluctuate either favourably or unfavourably.

3. *Ventilation.*—This subject had been already referred to under the heading of labour in dealing with the question of temperature. He thought the author had overlooked the fact that a great many mines even at the present day made a constant practice of securing ventilation by drawing air from the compressor lines.

It was a well-known fact that ventilation obtained in this way had the disadvantage of being abnormally expensive, and many of the mines that had followed this practice could have shown a reduction in cost by substituting for this practice a proper artificial air-blowing scheme.

In this connection it was interesting to refer to Mr. Speir's opinion; he stated that to supply 75 cub. ft. of air per person per minute, compressed at 73 lb. per sq. in., would require 13 h. p. per head, whereas for the same quantity of air supplied by ventilating fans only 0·113 h. p. was required.

It was doubtful whether an accurate estimate had ever been made on any mine of the cost of ventilation from air compression, because it was difficult to know exactly how much air was used for such a purpose; but he thought it might reasonably be argued from the figures quoted from Mr. Speir, and from the knowledge that was generally possessed of the excessive cost of such a practice, that there was every hope that, by proper provision for artificial ventilation by fans, a reduction under this head would be accomplished.

In any case, the author's figures were not supported by those quoted in respect to the East Rand Proprietary Mines.

4. *Mine Drainage.*—Relatively, it was safe to say that mine drainage would cost more at 5,000 ft. than at 2,000 ft., but it did not follow that, in the year 1920, mine drainage might not cost less at 5,000 ft. than it did in the year 1910 at 2,000 ft.

This was a cost that depended almost entirely on the question of power costs, and as shown elsewhere there was certain to be a considerable reduction in that direction every year.

5. *Hoisting Rock to Surface.*—Hoisting of rock in outcrop mines had been, from a strictly engineering point of view, inefficient and expensive. Tests made by the speaker showed that it took 1 lb. of coal to do 50,000* ft. lb. of useful work under the present system of isolated steam winding stations, and this figure approximately corresponded with the calculations made by Mr. Schweder,† of Goerz & Co. Compared with any ordinary statement of efficient work such figures were at once branded as unnecessarily wasteful, and it was not too much to say that, when working with purchased electric power and making allowance for the advantages to be gained under the heads of maintenance, etc., there was scope for a reduction of from 50% to 75% in this charge.

Underground Transport. — Under the old system of many hauling levels, short backs, and ill-conditioned tram lines with poor rolling stock, underground transport was in a great many cases expensive, and there was no doubt that, with fewer hauling levels properly equipped with motor power, under this head there was an excellent field for reduction of costs, despite the fact that the tram lines would be longer.

Power.—The cost of power on the average mine, generated at a number of points and including all the steam-heating devices inseparable from such plants, had figured on the average at something like £25 per h.p. per annum, but now that power could be purchased at a comparatively low rate and the consumption of power regulated to some fair approximation of the actual work to be done, there were coming and would continue to come enormous reductions under this head.

So much had already been written and so much proved of the actual advantages that lay in the purchase of electric power by mines, as against the generation by each mine of its own power, that he thought few would disagree with him in stating that there were great possibilities in this direction that made for the reduction of cost under this important head in the future.

Capital Charges.—One of the heaviest charges that handicapped the deep level proposition, was the cost of sinking and equipping deep level shafts. It was not improbable, however, that the deeper levels might be saved considerable expenditure in this direction, by amalgamation with the properties lying immediately above them.

In this connection Mr. G. A. Denny stated, in the *Mining Journal*, that:—

"The equipped mine should, after reaching the dividing line of the properties, proceed to develop the claims lying beyond its boundary, extracting and reducing the ores of the adjoining company . . . . having such an arrangement in view, its inclusive working costs might be relieved of anything up to 10s. per ton for interest and redemption on capital account and its condition changed from an unprofitable to a profitable one."

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* *Jnl. of S. A. A. Eng.*, vol. xi, p. 83.

† *Jnl. of S. A. A. Eng.*, vol. xi, p. 83.

This question of capital outlay became very serious as depth was reached, and it was necessary to have larger areas over which to spread the capital charges in order that the debit per ton might be brought to as low a figure as possible.

In conclusion, the speaker wished to say that whilst there was something to be said on the author's side there was also a great deal to be said on the other side.

The advances already made in metallurgical practice, coupled with the important strides being constantly made in almost every direction justified, in his opinion, the conclusion that continued progress would be made in coping with the disabilities of mining ore from very great depths.

He had only touched briefly on some of the more important points of the author's paper, but he hoped that what he had been able to state would to some extent counteract the pessimistic views put forward by the author.

Dr. A. L. Simon said that he listened with great interest to the sharp criticism to which the paper had been subjected; he welcomed such a practice, which was novel so far as meetings of the Institution were concerned. He thought that the author in his introductory remarks had mentioned a low grade ore for the deeper levels more to emphasise the necessity of a paper such as he had presented than to discredit the future possibilities of the Rand. The same remarks applied to the depth limit at which the mines could still be worked profitably. He thought these introductory statements were of minor importance; they simply expressed a pessimistic view in order to forestall any future eventualities.

The author advocated better sorting. Whether or not that were practicable was, however, still left an open question, unless the author could show that the sorting of the small sizes could be done effectually without unduly increasing the gold yield of the sorted rock. Mr. Claudet had already touched upon that point, and it would be interesting to hear the author's answers to Mr. Claudet's questions.

Mr. H. A. Tute: There appears to be very little to add to the criticisms on the author's introduction, but there still seems to be room for one or two remarks on the paper itself. The author endeavours to save about 8d. per ton (reduced to about half by Mr. Truscott's corrections) in his treatment costs. His method of doing so involves sorting $-2+\frac{3}{8}$ rock. Supposing this can be done, it would inevitably mean a large increase in the number of natives in the sorting floor (quite possibly amounting to 130%), and these natives would have to be of a better class than at present employed (see pp. 469 and 471, of the author's paper). Here he touches a big question, that of labour.

Unfortunately a large percentage of the labour supply of the Rand is not fitted for the hard work underground, and naturally this percentage is employed when it can be of some use, *i.e.*, on the sorting floors, etc. Further, the total supply is limited, so if one increases the surface hands to any large extent, one must do so at the expense of the mine underground; and if, in addition, one uses the better class demanded by the author, it would involve a further decrease in efficiency underground owing to such men having to be taken from natives who would otherwise be employed in breaking rock, etc., in the mine. From this it appears that any saving effected in reduction charges on these lines would be at the expense of the work in the mine and would prove a very costly economy.

Mr. H. Leupold: This excellent paper dealing with new proposals intended to effect remarkable economies in ore reduction, is sure to attract considerable attention, and will no doubt be much appreciated among mining engineers.

The diversion from the battery of all particles already fine enough before reaching it, has long been an ideal to many of us, but as long as 25 to 30 mesh screens had to be used the cost of preliminary fine screening was prohibitive. The advent of the tube mill, however, allowing of much wider mesh battery screens, has now rendered a practical solution possible, and the author is to be congratulated on the close and detailed study he has devoted to the subject. Especially to be commended is the new style of flow-sheet employed, which, instead of the mere well-known lines, gives, by the addition of a second transverse dimension, a graphical indication, easily grasped, of the proportional quantities involved.

A slight correction must, however, be made to the figures on p. 476. Ross E. Browne gives the cost of sorting and breaking per ton milled (in 1905) as 0·543s. or 6·52d., but assuming with the author that this has now been reduced to 3·75d. per ton milled, the cost of sorting becomes 1·875d. per ton milled. On the basis of 16% waste rock removed this equals 1·575d. (and not 2·24d.) per ton hoisted, or 9·843d. per ton of waste rock removed (say 10d. instead of 14d.)

At this point a confusion appears further to have arisen between "per ton sorted," *i.e.*, per ton of waste removed, and "per ton sorted," *i.e.*, per ton of mixed rock submitted to sorting.

For while here and on p. 477 the cost of sorting is given as 1s. per ton *of waste discarded*, in Table IV. this rate is used as per ton of rock *going to the sorting belts.* The total cost of sorting is thus greatly exaggerated in Table IV.

To estimate it correctly I will assume the author's figures of 1·875d. per ton milled for sorting and screening combined, and of 0·5d. per ton screened, *i.e.*, delivered to the screen, though the latter figure may appear low where trommels and abundant water are required.

In Table IV., "Existing Plant," col. 1, the 100 tons hoisted are all screened, and 70 tons (above 2 in. size) are submitted to sorting, and the total cost at 1·875d. per ton milled (84 tons) should be for the combined process 84 × 1·875 = 157·5d. = 13·125s.

If we now deduct 4·16s. as the given cost of screening, the cost of purely sorting 70 tons stands at 8·965s., or 0·128s. per ton sorted (that is, *submitted to sorting*), and as 16 tons of waste are removed, the cost per ton of such is 0·56s. or say 6·7d. Further, as 0·5d. per ton delivered to screen (of which there are 100 tons) equals 3·3d. for each of the 16 tons waste, the sum—6·7 + 3·3 = 10d. is the same as found previously per ton of waste removed.

If now the rate of 0·128s. per ton to be sorted is substituted in Table IV., line 5, the items of cost in all the seven columns of line 5 are considerably reduced, *i.e.*, by 0·872s. per ton in each case, giving a far more favourable aspect to those systems which involve much sorting.

In fact the costs now stand :—

			Fig. 2.	Fig. 5.	Fig. 8.	Fig. 6.	Fig. 9.	Fig. 7.	Fig. 10.
			1	2	3	4	5	6	7
8	Total Cost	shillings	363·04	274·12	277·14	263·62	268·93	266·31	271·08
10	Per ton hoisted.	Saving shillings	—	0·89	0·86	0·99	0·94	0·97	0·92
11		Saving pence	—	10·68	10·32	11·88	11·28	11·64	11·04
12		Saving per cent.	—	24·5	23·7	27·3	25·9	26·7	25·3

The difference between the various new proposals is slight, and an approximate figure for their working costs is 2s. 9d. per ton hoisted as against 3s. 7½d. for existing plant.

These figures are eloquent in favour of the author's proposals and especially of Fig. 7, which is further recommended by its great simplicity.

The corrections indicated also affect Table XI. and Fig. 11, the total saving being now from 11·61d. to 13·35d. per ton hoisted, or an average of 14·9d. per ton milled, improving the nett profit of Fig. 11 by 2·5d. per ton.

Part II. of the paper discusses the interesting results of single *versus* double-stage crushing trials from the point of view of their relative crushing and consequent extraction efficiencies, and will, I trust, give the final quietus to those, if any still survive, who in past years persistently opposed the introduction of tube mills on the Rand.

THE AMALGAMATION OF GOLD IN BANKET ORE.

(*Read at March Meeting, 1911.*)

By W. R. Dowling, M.I.M.M. (Vice-President).

DISCUSSION.

Mr. J. R. Williams (*Past-President*) : I am not going to attempt to criticise this paper, but I notice one part of it where a little explanation is required. It is with regard to the object of putting a shake on the plate of the first tube mill that was originally started at the Glen Deep. At that time I had a great deal of discussion on this question with Mr. G. E. Webber, the General Manager, who doubted the necessity for shaking plates following the tube mill. The object of the tube mill then was totally different from what it is to-day. We were then endeavouring to regrind the spitzlutte concentrate which consisted of about 15% by weight only of the mill product, carrying roughly about 24% of pyrite. Without a shaking plate it was quite impossible to obtain any amalgamation, because the large amount of concentrate present banked up on the plate leaving only a small channel for the pulp. I understand instead of 15% of the mill product is separated, the author's contention is that where about 60% the ratio of classification and concentration is entirely different now from what it was then.

The Chairman: I am sure we are very much obliged to Mr. Williams for his historical explanation.

Dr. W. A. Caldecott (*Past-President*): Mr. H. A. White's proposal that "all the gold possible shall be won by amalgamation even though some slight loss of profit is involved in securing it" is hardly likely to be generally accepted, since the ideal of ore treatment should be the maximum profit per ton of ore treated. To suggest that this fundamental principle should be departed from for share-market considerations, is to admit the influence of considerations upon metallurgical practice which are foreign to its scope. In any case, however, Mr. White has indicated a method apart from a gold reserve which sacrifices neither profits nor uniformity, in that if the cyanide clean-up is completed well before the end of the month, the amount of the amalgamation recovery is in accordance with the date of the later mill clean-up.

As regards the possible loss by non-amalgamation of black sand, upon which Mr. White lays stress, the actual results on various plants quoted in the paper and discussion constitute the practical reply. From the theoretical standpoint and defining black sand as particles of pyrite attached to particles of metallic gold so that the surfaces of each constituent of such 'mixed grains' are partly exposed, this is precisely the condition in which most gold exists in ordinary sand, except that in the latter case the gold particles are mainly attached to particles of silica which encase them in part, in which condition they are dissolved by cyanide solution. But whilst the gold extracted from black sand contained in ordinary sand charges involves virtually no addition to the ordinary cost of sand treatment, it must be remembered that the concentration of black sand on amalgamated plates necessitates a heavy subsequent cost per ton of such material for skilled and trust-worthy labour, transport by hand, grinding in barrels and a special tube mill, mercury, and finally a special plant for air-lift cyanide treatment. In addition it is possible that some particles of black sand may have their exposed gold surface amalgamated on the plates without being retained thereon, but pass into sand vats. In such case the gold is protected against dissolution by cyanide until the mercury coating has been dissolved, with the result that the discharged sand residue may contain undissolved gold, which would not be present but for the detrimental effect of prior partial amalgamation. In addition to this loss, there is the amalgam and gold-bearing mercury eroded from amalgamated plates taking the overflow from the tube mill cones. The amalgam in launders to the cyanide plant, and the residue after cyanide treatment of "spitzlutte concentrate" before the advent of the tube mill and the efficient amalgam trap which its circuit constitutes, furnish very strong evidence as to the danger of amalgamated plates beyond the tube mill circuit to deal with the pulp overflowing the tube mill cones. Amalgam is liable to be abraded from these plates and is then subject to losses such as the above, which were unavoidably experienced in a less developed state of gold recovery practice. The use of amalgamated plates under modern conditions, other than in the tube mill circuit, hence appears to offer very doubtful advantages to compensate for the additional capital expenditure, increased working cost, and locked-up capital which their use certainly involves.

Mr. S. J. Truscott's reference during the discussion upon this paper to the effect of lime in mill water upon settlement and amalgamation in the tube mill circuit appears to be based upon a misconception as to the developments of Rand metallurgical practice in recent years. The Rand has been and is fortunate in drawing skilled metallurgical workers from all parts of the world, and in being able to profit by their knowledge gained under varied conditions, and has in consequence gradually developed a system of ore treatment which is still advancing, and whose actual progress is perhaps not less rapid, and certainly more profitable to shareholders, in that the vast interests involved cause those who control its operations to accept as final no claim for improvement by radical change, until demonstrated by continuous operation under regular working conditions.[*] The result is that the earlier conditions, when the same individual could deal authoritatively with mining, metallurgical and mechanical problems, have passed away and been succeeded, as in all highly developed industries, by an era of specialization.

In regard to the effect of lime a simple experiment shows that only the lime actually *dissolved* in mill water produces its characteristic effect of causing agglomeration and settlement,[†] and it is quite immaterial from this stand-point whether the desired percentage of alkalinity of mill pulp is due to alkaline mill service water returned from the slime plant to the mill, or to lime added to the ore[‡] and dissolved in the mortar-box or mill launders. In like manner the neutralization by lime of free acid or acid salts in the ore, otherwise causing amalgamated plates to tarnish, is equally well effected in either case. As a matter of fact, however, in the latest practice, with efficient cone classification of sand and slime in tailing pulp, lime is usually added to the ore entering the mill bins, as settlement of layers of slime with sand does not occur as in hose-filling of sand collecting vats, the cost of installing and operating a special ball mill for supplying finely ground lime to the pulp entering the slime collectors is avoided, and less lime is required per ton of ore owing to the thorough dissolution of the calcium oxide in the commercial lime.

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[*] See Dr. F. H. Hatch, The "James Forrest" Lecture, Institution of Civil Engineers, 1911.

[†] Proceedings of this *Society* Vol. II., 1898, page 797.

[‡] This *Journal* Vol. X. August, 1909, p. 52.

As regards amalgamated plates in the tube-mill circuit these were installed as a matter of course by Mr. J. R. Williams with the pioneer tube-mill on the Rand at the Glen Deep mill, to recover the metallic gold liberated by crushing coarse particles of ore in the tube-mill, and the author, so far from proposing to do away with these plates, has devoted much space and cited practical results on a large working scale to show that with good classification and cyaniding no other plates than these are required.

We are fortunate in having to-night a contribution from one of our oldest members Mr. J. S. Curtis, who was likewise one of the first trained mining engineers to come to the Rand after its discovery. Mr. Curtis' name is very closely associated with the early development of these fields, and in particular with the putting down on the Village Main Reef of the first borehole,* which definitely proved the value of the deep levels, until then an unrealised though all-important factor in the history of the Rand. The following are the remarks by :—

Mr. J. S. Curtis (*Member*): As regards the earlier statistics of amalgamation mentioned in my paper before this Society in 1894, Mr. Dowling quotes me as saying that the recovery was over 60% where a 900 mesh screen was used.

The gold produced in the early days, say from 1887 to 1892, was mostly from oxidised ore. In the latter part of 1887 I started the Langlaagte and Robinson mills of ten stamps each at about the same time. We were getting about 3 oz. to the ton from the Robinson, and nearly 2 oz. from the Langlaagte. The weight of the ore crushed was determined by the number of trucks, and the value of the ore was computed from the gold recovered, plus the gold in the tailing and in the slime, the relative weight of these being determined by separating a sample of the sand and slime and drying them. If I remember correctly, 15% was deducted from the weight of the ore for moisture and other losses Of course this was only approximate, but all calculations to-day are more or less approximate, as the weight of the ore going to the mill is an unknown quantity. However, there was little I.G.B. in those days, and 60% given as about the amount caught on the plates is probably correct. Although inside amalgamation was used in some mills on the Rand in the early days, I never made use of it in any of the mines I superintended. In wet crushing (silver) mills on the Pacific Coast we used 1,600 screen, but in gold mills the screen was not so fine.

In 1889 and 1890 I was manager of Du Preez. The new 20-stamp mill, built by Sandycroft on Western models, was doing very good work. The ore (which by-the-bye was not too easy to get) was running from 14 dwt. to 15 dwt. and gave 8 dwt. to 9 dwt. over the plates, which was about 60%. I only had a few assays made of tailing and slime while at the Langlaagte and Robinson, but when I was at Du Preez, from 1889 to 1891, I had more made, as then there were several laboratories in Johannesburg.

The 20-stamp mill at the Village Main Reef, of which company I was managing director, was running in 1892, and I had as assayer Mr. H. Ewer Jones, now of Rhodesia, on my staff. I remember that among other things I gave him to do was the making of an assay plan of the reef development. If I am not very much mistaken this was the first ever made on these fields.

On the Village we had stamps and Scoular concentrators. We sampled and assayed the ore, concentrate, tailing and slime; but the data in regard to the milling of the pyritic ore of those days, which generally made its appearance at a depth of 130 ft. to 150 ft. in the Witwatersrand mines, is not of very much interest, as concentration cum chlorination were trying to beat the cyanide process, which latter eventually proved an easy winner.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Assay of Tin.—"Considering the present high price of tin, the methods of assay generally used in technical work for valuing tin ores and products are most unsatisfactory. The methods usually described in text books are volumetric assays, depending upon the oxidation of stannous chloride to stannic, by means of various standard solutions. These assays are little more or less than approximations: the more experience the assayer has, the nearer the truth his results are. With very great care the practical limit of accuracy does not exceed 0·5% on 70% material. Volumetric assays of tin always tend to give low results. Stannous chloride is so easily oxidised to stannic chloride that, unless the flask the assay is made in, is filled with carbonic acid gas, the act of shaking the flask will cause the stannous chloride to oxidise to a certain extent and give a low result. A serious objection to the iodine assay, is that the iodine solution loses strength. The separation of tin from its solution by means of zinc, is most unsatisfactory; it often takes hours to completely separate the tin. The chemical properties of tin makes its accurate assay very difficult. On the one hand, the stannous salts are easily oxidised, on the other hand, the stannic chloride is volatile to a great extent unless precautions are taken. The oxide of tin obtained by digesting alloys or ashes with nitric acid, although very impure, is quite insoluble in nitric acid, and native tin stone is insoluble in acids to such an extent that it is possible to clean the ore of the bulk of other metals, other than antimony, associated with it.

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* This borehole was begun on 1st Dec. 1889, and the South Reef was cut on the 18th Feb. 1890 at a depth of 517 ft. Six inches of the core were assayed and found to contain 9 oz 12 dwt. fine gold per ton. The reef was not within the shaft at a depth of 486 ft. in April 1891, and the value of the reef was about the same as that found in the borehole. (*The Star*, 25th March, 1910).

These properties make it possible to combine the electrolytic assay and the gravimetric separation, depending upon the solubility of stannic sulphide in oxalic acid and obtain far more reliable results than is possible with the volumetric methods. Having reduced the tin to metal in a current of coal-gas or hydrogen (that is, the oxides obtained by digesting alloys or ashes with nitric acid, or the ore cleaned by acid in the usual way), it is dissolved in hydrochloric acid, and oxidised to the stannic chloride by heating a few minutes with a few drops of nitric acid. It must not be boiled, or stannic chloride may volatilise. The solution is diluted with water, filtered, made up to a standard bulk, an aliquot part containing about 0·4 gramme tin is measured into a beaker, dilute ammonia is cautiously added until a decided precipitate forms, this is re-dissolved by adding a hot solution of 3 grammes ammonium binoxalate and 7 grammes oxalic acid in water. The solution, which should be about 100 cc., is heated to boiling, and a current of H_2S is passed through for one hour, the solution is filtered, and the precipitate well washed with as little warm H_2S water as possible. The solution containing the tin is boiled on the hot plate about two hours to remove H_2S, then cooled and electrolysed. I use four pint Daniel cells. The small amounts of lead, copper, antimony and arsenic which contaminate the impure tin oxide are removed by H_2S in oxalic acid solution. The only metal likely to be present—and that only in very small amount—is iron. The deposited metal may be dissolved, and the iron estimated and allowed for. When the conditions of rapid electro-analysis with rotating electrodes are better known, there is not much doubt that the electrolytic assay for tin will replace the uncertain volumetric methods in technical laboratories. As regards convenience, it is not much more difficult to work than the volumetric assay. The operations up to the reduction in coal gas being the same, I have not given the full details of the process, as they are evident to any assayer; the method is not claimed as new."—ERNEST A. LEWIS, *London Mining Journal*, June 10, 1911, p. 606. (A.R.)

THE CYANIDE INDUSTRY.—"The world's output of gold in 1910 was, approximately, 23,000,000 fine ounces, about 25% of which was recovered by cyanide. The indirect value of the process may be measured from the fact that the cyanide bullion produced on the Rand in 1910 was equal to the profit earned by the mines, the figures being: Yield from cyaniding, £11,552,743; total working profit, £11,567,099. Conversely, without cyanide the Transvaal gold mining industry would not be a profitable one, and this would also apply to the principal mines with cyanide annexes in Australasia and the American continent. The gold curve since 1891—the year in which the cyanide process was introduced—pays tribute to the inventors of the process, to the able band of British and American metallurgists and engineers who developed it and to their successors who year by year bring the most complex ores to the cyanide vat and excavate for retreatment abandoned residues.

Fall of Cyanide Prices.—In this development the industry has been supported by engineering firms who specialise in mining plant and machinery. It has also received the support of cyanide manufacturers, and indeed the progress made in the manufacture of cyanide during the last 20 years is coördinate with the progress made in its application. Research chemists and chemical engineers in this field have solved many problems—and have failed to solve as many more. The expenditure on research and experimental work, on factories successful and unsuccessful for the manufacture of cyanide, cannot be less than $8,000,000. This covers Britain, America, and the Continent, in the period since 1891. In that year the price of cyanide was about 48c. per lb. on a 100% basis; to-day the price is 14c. f.o.b. European ports. The fall was gradual and reflected the advent of new processes and reduced manufacturing costs. The following are f.o.b. London prices and indicate the tendency: 1894, 50c.: 1895, 30c.; 1896, 24c.; 1897 to 1901, 20 to 22c.; 1902, 19c. per lb. Toward the end of 1903 the market found its level at 14c., several producers going under.

Concurrent with the reduction in price the standard of quality increased. Calling to mind the commercial cyanide of 18 years ago, a dirty gray in colour with 20% of insoluble matter and 5% of cyanates, manufacturers who have lived through the period are more inclined to regret the errors of the past than to glory in present attainment. It will serve no useful purpose, however, to consider methods of manufacture which are long since dead and buried, or processes which have had an untimely end, or ideas which have not yet fructified. The mining community is concerned with present supply. The commercial cyanides of to-day have sodium as a base and are supplied in three grades containing cyanide equal to: (1) 98 to 100% KCN; (2) 120% KCN; (3) 128 to 130% KCN. The demand for the first is gradually diminishing, consumers recognising the economy involved in the transit of the higher grades. The first and third are produced in Britain, in America, and on the Continent, and are fused. The second (120%) is a product of the Continent alone, and being obtained by a wet process is finished by compression in the form of small briquets. The principal producers of fused cyanides work on lines laid down in Castner's patents; using metallic sodium as the base and ammonia as the source of nitrogen. An important quantity of high-grade fused cyanide is also produced, chiefly in Britain, through the medium of ferrocyanide and metallic sodium. The briquetted cyanide is a by-product of the beet-sugar refinery. The residue (*schlempe*) of the sugar, when heated, gives off a gas rich in ammonia and hydrocyanic gas. The former is collected as sulphate and the latter through caustic soda as cyanide of sodium.

Production Now Exceeds Demand.—While the producers of cyanide from *schlempe* and ferrocyanide are dependent for their supply of cyanogen upon the sugar refineries and the gas works, the synthetic makers have at their command unlimited quantities of raw material. At no time in the history of the industry has there been any shortage in the supply of cyanide; local scarcity excepted. The production during the last five years has far exceeded the consumption, and the chief producers to-day are running on decreased outputs. This is due on the one hand to the great development which has taken place in the treatment of the beet-sugar residues on the Continent, and on the other hand to a world-wide economy in the consumption of cyanide, fine grinding of the ore being largely responsible for this economy. The tonnage milled on the Rand in 1910 was almost double that of 1905, while the cyanide consumed showed an increase of barely 50%. Apart from its application to the treatment of gold and silver ores the market for cyanide is unimportant. A few hundred tons will cover the requirements of the electroplaters, and of the fruit growers who use it as an insecticide. These consumers prefer 98% potassium cyanide (single salt) despite its higher price. This product is still manufactured on the Continent, and to a lesser extent in Britain."—William Neill. *Engineering and Mining Journal*, May 6, 1911. (A. K.)

Detection of Nickel in Presence of Cobalt.—"Neutral solutions of nickel and cobalt salts produce with potassium chromate solution precipitates of hydrated basic chromates, $2NiO, NiCrO_4$ and $CoO, CoCrO_4$, respectively. The nickel compound, chocolate brown in colour, is formed slowly in the cold, even in concentrated solutions. To obtain a rapid precipitation it is necessary to heat to boiling. This reaction will show 0·000028 gram of nickel. The cobalt precipitate differs from the one just described in being reddish brown in colour, in having a strong tendency to adhere to glass vessels, and in being formed rapidly in the cold in solutions containing more than two grams of cobalt per litre. From more dilute solutions the precipitation occurs on heating to boiling, and in this way 0·000032 gram of cobalt has been detected. Both compounds are formed only in neutral solutions; they are soluble in weak acids and in ammonia.

The recognition of nickel and cobalt in solutions containing both metals in about equal amounts is simple (*Bull.* Soc. Chim. 1911, 9, 20-22.) The solution is treated in the cold with a 10% solution of potassium chromate. The cobalt precipitate, formed in the cold, is filtered out and the filtrate heated to boiling to precipitate the nickel. If the solution of the metals contains little nickel relatively to the cobalt, the filtrate from the cobalt chromate is, as before, heated to boiling, when a precipitate appears containing nickel with a little cobalt. This precipitate is washed free from potassium chromate solution, dissolved in ammonia in the cold, and evaporated by heating in a small basin. If nickel is present the basic chromate separates after a time with its characteristic colour, while the cobalt compound becomes dirty green. It is possible in this way to detect one part of nickel in presence of 100 parts of cobalt. If, in the solution of the metals, nickel preponderates greatly, *e.g.*, in the proportion of 50:1, the cobalt chromate does not separate in the cold. In this case the liquid is heated to boiling, and the basic chromate precipitate then formed is filtered, washed, dissolved in ammonia and evaporated as described above. The formation of a dirty green deposit shows the presence of cobalt."—*Engineering and Mining Journal*, May 13, 1911, p. 956. (M. T. M.)

METALLURGY.

Dressing of Ores by the Dallemagne Process.—"An interesting and comparatively new departure in connection with the dressing of composite ores is presented by the Dallemagne process, of which we have just received particulars, the main features of which may be summarised as: (1) A double hydraulic classification obtained by batteries of classifiers coupled in parallel, by which a perfect sizing of sand and slime is effected previous to feeding the pulp to the tables. (2) The peculiar semi-circular shake, adjustable for each kind of pulp, which imparts to all particles retained on the riffles a motion and a classification the effect of which is similar to the action obtained by the plunger jig. (3) The number of shakes which effects a successful separation on the slime: these shakes vary from 600 to 1,800 per minute as required in each case. As will be gathered from the report of the meeting of the owning company, which appears in another column, a large demonstrating works has been installed at Clichy, near Paris, where people interested may witness the actual operations. The advantages claimed for the process may be enumerated as: (1) Separation of the most complex ores. (2) Suppression of middlings. (3) Economy of power and upkeep."—*London Mining Journal*, May 27, 1911, p. 552. (A. R.)

Review of Metallurgy in 1910.—"Under this heading Mr. R. Stuart Browne writes an interesting article in your issue of January, 1911.

I cannot agree with him, however, when he gives the South African metallurgists the credit of establishing the value of coarse feed to tube mills.

As far back as May, 1907, I crushed through four-mesh screens on stampers, followed by dewatering previous to tube-milling, and that practice has been in vogue in our battery ever since.

Regarding tube-mill liners, we are using a patent liner of our own, which is also being used by other large mines in New Zealand, and judging by results it would seem that other liners are not to be compared with it. It is a bar-type liner and has been used by us for years."—S. D. McMiken.—*The Pacific Miner*, April, 1911, p. 117. (H. A. W.)

The American Metallurgical Society.—"The American Metallurgical Society was incorporated under the laws of the State of California, March 14, 1911. On April 12th, in accordance with the law, the members met in San Francisco and adopted a code of bye-laws and then elected a board of directors to serve for the years 1911 and 1912.

The mission of the Society is explained in Article 2 of its bye-laws, from which I take the following:

The objects of this Society shall be two-fold: First, it shall work for the advancement of the arts of mining and metallurgy, employing for that purpose the various means at its command, and more particularly by encouraging the exchanging of ideas on mining and metallurgical topics through the medium of professional papers and discussions which shall be given publicity in the transactions of the Society; and second, it shall work for the welfare of its members and shall encourage in them a healthful spirit of mutual co-operation."—C. P. Griffin.—*The Pacific Miner*, April, 1911, p. 132. (H. A. W.)

(We offer the new Society a very hearty welcome and our sympathy with their aims. We wish them every possible success.—Editorial Committee.)

MINING.

The Marathon Mill.—"The substitution of heavy iron or steel rods for the use of pebbles in tube mills is the essential principle upon which the Marathon mill, the invention of Ernest C. Johnson, is conceived. The actual grinding surface of the rods at a given moment is manifestly greater than the grinding surface of the pebbles, proportionately considered, and it is claimed for the invention that the rod method has been found more efficient, even to the point of dispensing with preliminary grinding. A possible reduction to 200-mesh fineness, if desired, is said by the makers to be accomplished by this one machine upon any size fed up to 2 in., thus doing away with the incidental appliances of stage reduction. The mill has adjustable tilt, which governs the progress of the charge through the cylinder, and thus determines the amount of grinding which it receives.

The grinder consists of a cylindrical drum having a specially designed lining of very hard chilled iron or steel, constructed so that the parts of the lining are keyed together when in place, thus obviating the need of clamps or bolts. Lengthwise they have a wedge shape which permits of easy insertion and removal, the whole lining falling to pieces when a single wedge is driven out. The interior surface of the lining is corrugated, to prevent the slipping of the rods in a mass. The rods range from 3 in. to 4 in. in diameter, and are approximately as long as the drum, which may be built up to 12 ft. in length, according to present specifications. The diameter of the cylinder is from 2 to 4 ft. The rods are given a rolling motion during the revolutions and they are said to be in constant agitation.

Revolutions to the number of about 35 per minute are recommended, the slow speed, it is said, having the effect of causing the greater part of the pulp to be ground by attrition on itself, thus saving the wear of the rods and lining. The pulp is confined within the spaces between the rods, and receives the continual action of them. The greater weight of the rods as compared with pebbles, is emphasised by the inventor. The heads and ends of the cylinder are built the same, so that it may be reversed or the feed removed from one end to the other.

It is claimed that the Johnson mill gives equally good results as a dry and wet pulveriser. In the course of dry pulverising the ground pulp drops through the peripheral discharge channels in the head. The discharge in wet work is affected by reversing the head, the side here used being flat and smooth, and thus procuring a tight joint. The discharge in this case is arranged by removing the plug in the centre of the head and inserting a pipe flange and pipe. The mill is suitable for grinding in cyanide solution.

The mill is designed to be applicable to the direct amalgamation of gold, an interior annular channel being left between the end of the lining and the inner surface of the discharge head, which in this case is reversed, as already described for wet grinding, and therefore tightly jointed to the cylinder. The quicksilver is placed in this channel, where it is well below the central discharge and also protected from the agitation of the grinding rods.

The inventor especially recommends the mill for the grinding of coal to dust, to be used in shot flame burning where a long flame is desirable. This method of coal burning is in use at Portland cement plants and elsewhere."—*The W.A. Mining Building and Engineering Journal*, April 29, 1911. (A. K.)

FROZEN DYNAMITE.—"Frozen dynamite is the subject of grave suspicion among most engineers, says the *Engineering Record*, yet experts hold that it is far less sensitive than unfrozen dynamite. In order to prove this, Dr. Walter G. Hudson, and E. J. Riedered, the latter superintendent of the Du Pont works at Lake Hopatcong, conducted some experiments on Feb. 24 that are decidedly interesting. They were made with a particularly sensitive grade of 60% gelatine dynamite and also with some straight dynamite. A number of these sticks were used as targets for bullets from a Krag-Jörgenson government rifle, loaded for a velocity of 2,150 ft. and fired at a distance of 50 ft. from the dynamite. The bullets discharged in this way failed in every case to explode the sticks of frozen, straight and gelatine dynamite. Other frozen sticks were then thawed and used as targets in precisely the same way. In every case the bullets detonated them, thus showing the decreased sensitiveness due to freezing. The same thing was shown in another way by using thawed sticks which were broken in two. If a cap with 30 grs. of fulminate of mercury was attached to half of a thawed stick and the other half was placed 8 in. from it, the latter would be exploded when the former was fired. If, however, a whole stick of frozen dynamite was placed 8 in. from a half stick of thawed dynamite and the latter was fired the frozen stick remained unaffected. In fact it was not until the frozen material was placed within 1 in. of the piece fired by the cap that it could be exploded. When sticks of frozen and of thawed dynamite were blown into the air with black powder it was found that the former took about three times as long to burn as the latter. Tests of this character seem to prove that frozen dynamite is not sensitive, yet even when it has been benumbed by the cold it should be handled with great care. It is best not to become familiar and careless with anything of such a violent temperament."—*Mining World*, April 1, 1911, p. 692. (A. R.)

UNDERGROUND METHODS IN THE WAIHI GOLD MINE.—"The ore is won from the mine by driving and widening out of levels, ordinary flat-back or overhand stoping and filling, overhand stoping with square sets and filling, shrinkage stoping and filling, taking out of arches, and surface cuttings. In driving and widening out of levels preparatory to ore-winning, a careful scrutiny and comparison is made of the plans affecting the locality, the position of the ore values is noted, the character of the lode and the enclosing walls considered, and the system of stoping determined. In most cases the developing level is widened out to the full width of the payable ore and to a height of 10 ft. with machine drills.

Winzes are sunk through the block of ore to be worked on, from the level above, at 80 ft. apart, either during the progress of development or before stoping is begun. The first or leading stope, is started at each winze on the top of the level sets, and the filling that has been previously packed in, and the full width of the lode or section of the lode is broken out each way horizontally for a height of 8 ft. and a length of 40 ft., the broken ore being trucked out to the shafts. Under this system stoping widths of 50 ft. are common on the Martha lode, and on the Edward a width of 80 ft. has been taken out without any support to the roof. In stopes of these dimensions, however, it is necessary to have two parallel drives in the level to facilitate the handling of the quantity of ore. After timbering up, filling material is then dumped into the winzes at the level above, and trimmed along the stope from the bottom of each winze 40 ft. each way to within 2 ft. of the roof of the stope. Slabs are then laid on the top of the new material, and the block is ready for the second stope. Stope after stope is broken out and filled in this way up to what is considered a safe thickness of arch for the temporary support of the level overhead. The thickness of this block varies with the width of the stope and the nature of the lode and surrounding walls, and may run from 7 ft. to 24 ft. In this method of stoping no timber is used, as a rule, to support the roof. The roof is kept well arched, and any loose rock barred or worked down. Machine drills are not used in this system, except, perhaps, in the leading stope, owing to the soft nature of the filling and the necessity of keeping this material dry.

Stoping by the method described is let on contract, the number of tons let in one contract varying from 5,000 to 50,000; and a system of weighing, tallying, and checking, determines the quantities put out. Overhand stoping with square sets and filling is in use where the roof will not stand without support, and also where a band of loose rubbly quartz or old filling exists on the side of the stoping block. As each stope proceeds the timbers are put in close up to the work, and wedged up to the roof, and, as work advances, the portion taken out is filled up close with filling from the surface. The following stope is worked off the filling, the timbers being placed on top of the sets in the lower stope, and this system is carried up to the floor of the level above, the whole of the ore being taken out. In certain cases, where a large portion of the lode consists of loose rubbly ore, the winzes, originally sunk in the reef, have fretted away to large dimensions, and eventually have had to be filled with waste. Shrinkage stoping and filling has been adopted during the last four years in places where the reef walls are sound and unlikely to fall away. Seven or eight blocks are now constantly at work by this method, and an endeavour is made to regulate the procedure so that two blocks will have completed ore-breaking, and be in process of emptying, whilst the others proceed with breaking. The lengths of blocks taken out by this method vary from 150 ft. to 300 ft., and widths up to 50 ft. have been taken. The ore is broken down with machine drills, spalled to small size, and a proportion drawn off from the shoots as work proceeds, leaving sufficient in the stope to enable men to work without staging or timber of any kind. No shovelling of the ore is required, and the material is broken out for the full width of the lode, or stoping section, up to a safe arch for the level above. This stoping is let by contract at a price per ton for ore delivered at the shaft, and a price per ton of ore broken in the stope.

When emptying the stope the ore is drawn off from one end first, and filling from the surface is dumped into the space down winzes sunk through the level arch. As the ore is drawn away in front, waste filling is tipped in till the whole block is filled up tight to the arch above. It is sometimes found convenient in blocks of small length, to draw off all the ore first before filling begins, and then quickly fill up the whole space. Usually in drawing off, some of the country flakes away from the hangingwall, and this is left in the stope for filling. Under favourable conditions, this system of stoping has been found to be the most economical method yet practised, there being no cost for shovelling ore, no timber used, no cribbing up of passes, and less trimming of filling material. In taking out the arches of ore usually left to support the levels, a start is made by placing strong stringers or booms horizontally along the bottom of the level on each side of the track close up against the sets. These are from 15 ft. to 20 ft. long, and are so arranged as to keep the legs of the sets up. The arch is then ready for breaking out, and the ore is removed down to the old stope filling. After the ground has been taken out for three or four sets the open space is immediately filled in with hard rock, if procurable, and the booms are moved on another length. In cases where the walls are not good, sill pieces are laid down on the top of the old filling in the stope below, and square sets built on these, the level timbers and booms being taken upon the square sets. When the old level timbers are crushed or decayed, a footwall gangway is driven in the country rock about 6 ft. from the lode and parallel with it, but about 10 ft. or 15 ft. below the floor of the original level. A quartz pass is constructed from the gangway to the level below, and the arch is attacked from the gangway and timbered with square sets. This method is the safest in treacherous ground, and recovers all the ore standing.

At the conclusion of the paper, Captain Richards, manager of the Mount Morgan mine, said the system of mining was an excellent one. He had been much interested in the subject, because trouble and loss of life had been caused at Mount Morgan by the nature of the country and unsuitable mining methods, and as the result of a Commission every conceivable system of mining had been adopted. In the Waihi mine, which he had examined, every advantage had been taken of the nature of the ore in deciding upon the system adopted. Many difficulties had been experienced, especially with water, but they had been overcome, and had even been made use of in enabling the ore to be won at the lowest possible cost."—L. Gilmour and W. H. Johnston, *Australian Institute of Mining Engineers. — London Mining Journal*, April 8, 1911, p. 353. (A. R.)

MISCELLANEOUS.

Measuring the Flow of Compressed Air.—"One of the things at present most to be desired in compressed air practice is some practical and reliable system or apparatus for measuring the volume of air during transmission. We can ascertain both pressure and temperature with any desired accuracy, but the volume during transmission is more elusive, while upon it, in connection with the others, depends all accurate knowledge of capacies and efficiencies, and also of the rate of consumption and ultimate cost of operation in different lines of service. Contributions to our stock of knowledge in this field are accumulating and they are all to be welcomed.

We present a statement (says *The Compressed Air Magazine*) furnished us by Mr. W. F. Gillies, El Paso, Texas, of certain experiments made by Mr. Legrand, mechanical engineer for the Phelps-Dodge interests at the Copper Queen, Arizona (U.S.A.) mines.

Mr. Legrand has devoted several months to a series of interesting experiments in the measurement of the flow of air through an orifice. He has been testing Fliegner's formula as to its correctness for large volumes. He first with a small compressor proved it up to 1,100 cub. ft. of free air per minute, and has since tested it up to 4,000 cub. ft. with a larger compressor, and with a still larger compressor available he is planning to test up to 6,000 cub. ft.

His method is as follows: —

First—Knowing the clearances of the low pressure or intake cylinder, he calculates the actual piston displacement, or the free air capacity of the machine.

Second—He takes indicator cards of the low pressure cylinder, using a light spring, generally four pounds, and measures the actual volumetric efficiency as closely as possible.

Third—He tests the pipe line to the receiver and the receiver itself for leakage.

Fourth—Calculates the weight, or volume, of air passing through the orifice by means of his formula.

Fifth—Checks 4 by 1 and 2.

The formula used is Fliegner's, to be found in Peabody's 'Thermodynamics,' and is as follows:—

$$\text{Wt. of air per min.} = \cdot 53 \text{ area of orifice} \sqrt{\frac{P}{T}}$$

P = Abs. pressure.
T = Abs. temperature.

The orifice, made of brass and polished, was rounded on the side on which the air entered, the radius being equal to the radius of the opening and the length being twice the diameter. The first tests were made with a ¾-in. orifice, but in the last test a large one, 1·05-in. diameter, was used to show whether the constant ·53 held true for other diameters.

The tests of the smallest compressor up to 1,100 cub. ft. checked to within 1% or 1½%. The following figures will show how near the results obtained on the larger volumes worked out:—

No. of Test.	Size. of Orifice.	No. of Orifices.	Cub. ft. free air Displ'c'd.	Cub. ft. free air Deliv'r'd.	Volumetric efficiency by Card.	Volumetric efficiency by Orifice.
1	¾in.	1	1,401	1,251	94·7	89·3
2	¾in.	2	2,438	2,212	93·8	90·7
3	¾in.	3	3,513	3,217	93·7	91·5
4	1·05in.	1	3,555	3,256	93·7	91·5
5	2-¾in. and 1-1·05in.	3	4,328	4,017	94·5	92·8

The air pressure in the receiver was held at between 100 lb. and 101 lb. gauge.

It is thought, and the indicator cards show, that there was some leakage back of the discharge valves. This would account for the decrease of difference between the two volumetric efficiencies as the speed increased, and this matter is to be further investigated. The considering in this way the air discharged takes into account all losses due to leakage, the heating of the air entering the cylinder, etc., and the results seem to agree very closely."—*Australian Mining Standard*, Aug. 17, 1910. (K. L. G.)

VOIDS IN SAND AND BROKEN STONE.—"A simple and accurate method employed by the Aberthaw Construction Co., of Boston, Massachusetts, for determining the voids in sand and broken stone for concrete work is given in the *Cement Age*, as follows: Apparatus necessary, a galvanised iron ash-can and platform scales that will weigh up to at least 500 lb. Weigh can empty and indicate weight by W. Weigh can filled with water and indicate weight by W_2. Weigh can filled with stone and indicate weight by W_3. Weigh can filled with stone and water and indicate weight by W_4. Substract W from W_2 and divide by 62·4 to find the number of cubic feet in the can, indicate by C. Substract W_3 from W_4 to find the number of pounds of water required to fill the voids in the stone or sand; divide by 62·4 to reduce to cubic feet and indicate by V. Then C divided by V will give the percentage of voids in the stone or sand. This method is used extensively by the Aberthaw company, and is accurate, convenient, and quick."—*Mining and Scientific Press*, Oct. 29, 1910. (K. L. G.)

THE VALUE OF CHROME ORE.—"The price of chrome ore varies from $10 to $20 per ton, according to grade of ore and conditions of supply. Imported ore from New Caledonia carrying 50% chromic oxide is quoted in New York at $14 to $16 per long ton, exclusive of cost of transportation. If the chromic oxide exceeds 50%, the value of the ore rises in proportion; if the chromic oxide is less than 50%, the value of the ore decreases at a more rapid rate. The price of the California ore is governed almost entirely by local conditions, as there is little or no competition with foreign ores."—*Mining Science*, May 4, 1911, p. 463. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

GUIDE TO TECHNICAL WRITING, by T. A. RICKARD, editor of the *Mining Magazine*. Published by Mining and Scientific Press, San Francisco, Cal.

"The book contains 168 8mo pages embodying the criticisms, pleas, and suggestions of an earnest man for better mining and metallurgical writing. Several years ago Mr. Rickard sent the writer a pamphlet accompanied by a letter in which he suggested that uniformity in expression and style be adopted. The United States Printing Office, the United States Geological Survey, The International Textbook Co., and others, have adopted 'style sheets' that contain rules for editors, writers, proof readers and printers, but this is not what Mr. Rickard had in mind, for an author could conform to the rules yet the subject matter be written poorly. A plea for 'Greater Simplicity in the Language of Science' was another paper by Mr. Rickard and was read before the American Association for the Advancement of Science. By this time Mr. Rickard had come to the conclusion that to disseminate mining and metallurgical information where it would do the most good, the literature must be stripped of pedantry. He commenced by telling the pedants, in the parlance of Wall Street, that while they were long on orthography and etymology, they were short on syntax. While Mr. Rickards criticicisms pointed out the idiosyncrasy in others, they were not always received in the right spirit, for which reason he made a few enemies and a host of friends. Being in sympathy with Mr. Rickard, it is difficult to criticise his 'Guide to Technical Writing,' for when following the argument interestedly, carelessness like that on page 24, where *then* is used for *thus*; or '12 divided by 3 making 4 ft.; or 'Thus 8 by 12 in., not 8 × 12 inches' is apt to be overlooked. Again, some might differ with him on page 25 and write ten is accurate, 10 is exactly, 10·0 is accurate to tenths only. Probably no one has done more to eliminate superfluous words in technical writing than Mr. Rickard, nevertheless on page 62 he writes 'So far so funny, but . . .' When writing on the use of titles, he has this statement 'In America, it is chaos; the titles Professor and Doctor are employed so loosely that they are well-nigh meaningless.' The criticism is that 'if it be chaos' his sentence is just as fulsome without that phrase. Usually when an author trips in one paragraph, the critic finds two other stumbles, for example 'Surgeons, veterinaries and dentists are denied the privilege,' could have been written 'The privilege is denied to Surgeons, etc.' without making use of a false passive. Mr. Rickard's paper on 'Standardization of English in Technical Literature' fell like a bombshell among the Englishmen, who were using dreadful words in their writings. It made them gasp and exclaim 'Whatever manner of man is he.' Read his 'Guide to Technical Writing'; hit him in the back when he is looking at you and you will ascertain as the *Canadian Engineer* remarked, that' His pen has a sting as sharp as a serpent's tooth.' The intent of the author on Technical Writing was to benefit the mining and metallurgical professions, and this he has accomplished to a greater extent than he is aware. Mr. Rickard's writing is not above criticism; he does not profess to be a hierophant; he is always fair and open to conviction; and does not use his time in useless fault finding."—*Mines and Minerals*, June 1, 1911, p. 680. (H. A. W.)

Barlow, C. W. C., Magnetism and Electricity, (Vol. 1. Mathematical Physics). *University Tutorial Press.*

Berry, C. W. The Temperature-Entropy Diagram. 3rd edition. Revised and Enlarged. Cr. 8vo. *Chapman & Hall.* Net 10s. 6d.

Bleloch, W. E., The Witwatersrand System. With Map Mounted on Linen. Royal 8vo. limp. *Marlbrough.* Net 42s.

Bousfield, W. R. & W. E. The Specific Heat of Water and the Mechanical Equivalent of the Calorie at Temperatures of 0°C to 80°C. 4to. sd. *Dulau.* Net. 3s.

Burnside, W., Theory of Groups of Finite Order. 2nd ed. 8vo. pp. 536. *Cambridge University Press.* Net. 15s.

Fay, I. W. The Chemistry of Coal Tar Dyes. 8vo. *Constable.* Net 16s.

Friend, J. N. The Corrosion of Iron and Steel. Cr. 8vo., pp. 314. *Longmans.* Net 6s.

Fritsch, J. The Manufacture of Chemical Manures. 8vo., pp. 320. *Scott, Greenwood.* Net 10s. 6d.

Hooton, W. M., and Mathias A., A Preliminary Course of Mechanics and Physics. *University Tutorial Press.*

North, S. H. Oil Fuel: Its Supply, Composition and Application. 2nd edition. Cr. 8vo., pp. 250. *C. Griffin.* Net 6s.

Paterson, Marcus. Auto-Inoculation in Pulmonary Tuberculosis. 4to. *Nisbet.* Net 21s.

Read, E., Steel Bar and Plate Tables. Royal 8vo. sd. *Spon.* Net 1s.

Redmayne, R. A. S. Modern Practice in Mining. Vol. 4. The Ventilation of Mines. 8vo., pp. 246. *Longmans.* Net 6s. 6d.

Simmons, W. H., and Mitchell, C. A. Edible Fats and Oils, their Composition, Manufacture and Analysis. 8vo. pp. 158. *Scott & Greenwood.* Net 7s. 6d.

Stewart, R. Wallace, and Satterly, John. A Text-Book of Heat, Theoretical and Practical. (The University Tutorial Series). Cr. 8vo, pp. 480. *Clive.* Net 4s. 6d.

Students' Lyell, The Principles and Methods of Geology. etc. Edited by John W. Judd, 2nd edition; revised and enlarged. Crown 8vo. pp. 646. *J. Murray.* Net 7s. 6d.

Thorpe, Sir Edward. Essays in Historical Chemistry. 8vo. pp. 614. *Macmillan.* Net 12s.

Warden, E. C. Nitrocellulose Industry: a Compendium of History, Chemistry, Manufacture, &c. 2 Vols., 8vo. *Constable.* Net 42s.

Abstracts of Patent Applications.

(C.) 227/11. Arthur Francis Berry. Improvements in or relating to electrical transmission and distribution systems. 21.4.11.

This is an arrangement which aims at protecting a transformer from the effects of lightning or electrical surging by automatically placing a coil of considerable impedance or a supplementary transformer in series with the transformer to be protected and providing a path to earth by means of spark gaps or equivalent electrical discharging devices. The automatic action is effected by electro-magnetically operated switches.

(P.) 568/10. Henry Arthur Goddard. An improved mode of building in concrete and apparatus therefore. 21.11.10.

This invention relates to an improved method of building hollow walls of concrete.

(*a*) An internal and external panel or frame maintained in position by a saddle clip, yoke, or straps, and

(*b*) An intermediate box or collapsible core, for coring a hollow, or space between the two faces of the wall the formation of the core box in two parts, the adjacent and co-operating edges of which parts are inclined, whilst when both parts are in position, the whole forms a parallelepiped. A space or interstice, either taper or with parallel sides is left between the adjacent inclined edges of the two parts of the core box, and this space is maintained notwithstanding the pressure of the concrete about the box, by the insertion of wedges or by other means as detailed in the specification accompanying this application. By adopting this system, modified to suit various requirements, it is possible to easily withdraw the core box, after the concrete has set.

(C.) 241/11. Edgar Arthur Ashcroft. Improvements in or relating to sulphide ores and products. 28.4.11.

This application refers to improvements in the treatment of mixed sulphide ores, particularly those of zinc and lead. The hot mixture is leached with fused zinc chloride and sodium chloride, lead chloride being formed and zinc sulphide left behind in the residue. Following this partial separation, methods are employed to obtain the free metals, principally with the aid of chlorine followed by electrolysis.

(P.) 365/10. Allan Thomas Cocking. Improvements in torches for fuse lighting and analogous purposes. 28.7.10.

This application is for a torch suitable for fuse-lighting consisting of a core of gun-cotton gelatinised by nitro-glycerine in such proportion as to form a hard elastic compound similar to vulcanite. The outer covering is of muslin slightly impregnated with potassium nitrate to assist in its combustion in front of the flame, such covering being glued to the core and afterwards rubbed with an inert substance such as French chalk to render it superficially incombustible.

(C.) 553/10. Christian Eliza Liebenberg. Improvements applicable to headgear crushing plants for collecting the rock-dust produced therein. 12.11.10.

This application is for apparatus for the collection of dust produced during rock-breaking operation and consists of a suction-fan operating within a conduit the open ends of which are in proximity to the discharge from the rock-breaker.

(C.) 239/11. Wilfred Watkins Pitchford. Apparatus and the method of using the same, for titrating solutions of arsenical and other active substances. 27.4.11.

This application relates to apparatus and method of using same, for the titration of solutions.

The apparatus consists of two graduated tubes, closed at one end. The first called the "Isometer" has an indrawn mouth so as to be conveniently closed by the thumb, and the second called "drop tube" is fashioned to hold a ground in, channelled, drop-bottle stopper. The advantages claimed are that having both vessels graduated check readings are obtainable: and also that an ordinary bottle can be used instead of the "drop tube"; further that the apparatus is portable and dispenses with burette stands, beakers, etc.

(C.) 242/11. Anders Ponten. Centrifugal sluicing apparatus. 28.4.11.

This invention relates to an apparatus for separating and recovering fine metallic particles, such as gold from sand or gangue.

The inventor claims in a centrifugal sluicing machine, a hollow rotary circulator composed of a plurality of conical sections arranged to form inwardly projected circumferentially disposed riffles, an annular ring on the uppermost section forming an inwardly projected circumferential riffle, each conical section being outwardly flared adjacent its upper edge to provide a retaining seat. A bowl within which said circulator is suspended, a series of radial blades or wings situated within the annular passage way between the bowl and the circulator; a helical feeder at the lower end of the bottom section of the circulator, a bottom detachably secured to the bowl, a casing secured to the bowl, a trough surrounding the same, a cover for the casing, means for delivering material to be treated to the helical feeder already mentioned, and mechanism for imparting rotation to the circulator.

The object of the invention is to provide for the rapid and excessive wear to which the collecting surfaces of the circulator is subjected by the cutting action of the material working there against, to simplify the construction of the circulator and its associated parts, to reduce the cost incident to said circulator, to facilitate the repair of worn parts, and in general to improve the working condition of the sluicing apparatus by prolonging the usefulness and life thereof.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 331/11. Hugh McMenamin. Improvements in apparatus for crushing rock and the like. 13.6.11.

(P.) 332/11. Hans Glück. Improvements in and relating to tube mills and the like. 13.6.11.

(C.) 335/11. Samuel Archibald Vasey. A soluble protein or albumen and a process for the production of the same from meat, fibre or other insoluble protein matter. 16.6.11.

(C.) 337/11. Nils Testrup. Improvements in and relating to the treatment of oils, fats and the like. 16.6.11.

(C.) 338/11. Lyndall Forse (1), John Alderson Mayo (2), Frederick Stanley Spiller (3). Improvements in and relating to wheels for vehicles. 16.6.11.

(P.) 339/11. Andre Leon Felicien (1), Levy Strauss (2). Improvements in torches suitable for igniting fuses. 16.6.11.

(C.) 340/11. Martin Deacon (1), William Gore (2), Palatine Engineering Co., Ltd. (3). Improvements in or relating to filtering apparatus. 17.6.11.

(P.) 341/11. James Gitsham. Improved process for the recovery of gold and silver from antimonial, arsenical, and other gold bearing ores. 17.6.11.

(C.) 342/11. Patrick Edward Cunningham (1), Reginald Reid (2), Valentine Muir Fitz-Hugh (3). Improved spark arrestor for locomotives. 17.6.11.

(P.) 343/11. Thomas Dawson. Improvements in fencing standards, pegs, posts and the like. 17.6.11.

(C.) 344/11. Harry Pauling. Improvements in the absorption of nitrous gases. 17.6.11.

(P.) 345/11. James Wilshir. Valve device for checking the flow of fluid. 17.6.11.

(C.) 346/11. William James Dunne. An improved liner or foundation for abrasive or corrosive treatment appliances. 19.6.11.

(P.) 347/11. James Edward Vaughan. Improvements in rock-drilling machines or apparatus. 20.6.11.

(P.) 348/11. F. W. Smith. Means for allowing battery stamps to continue crushing whilst battery plates are being dressed. 21.6.11.

(P.) 349/11. William Robert Nichols. Improvements in means for checking the speed of runaway skips and the like. 23.6.11

(P.) 350/11. William McVitty. Improvements in cam shaft bearings. 23.6.11.

(C.) 351/11. Harry Pauling. Process for completely absorbing dilute nitrous gases. 24/6/11.

(P.) 352/11. William Charles Stephens. Improvements in rock drills. 24.6.11.

(P.) 353/11. William Charles Stevens. Improvements in cocks and valves. 24/6/11.

(C.) 354/11. John Thomas. Improvements in smoke consuming and fuel economising apparatus for furnaces and the like. 24.6.11.

(C.) 357/11. Peter Burd Jagger. Improvements in means for attaching rails to concrete sleepers. 26/6/11.

(C.) 358/11. Peter Burd Jagger. Improvements in collapsible cores for making hollow concrete posts and the like. 26.6.11.

(C.) 359/11. Peter Burd Jagger. Method of manufacturing pipes from concrete and means therefor. 26.6.11.

(P.) 361/11. Richard Evelyn Parker (1), John Thomas Cook (2). Improved coal burning assay furnace. 28.6.11

(C.) 362/11. William Campbell Paterson. Improved agitation apparatus especially applicable for the treatment of pulp containing precious metals. 30.6.11.

(C.) 363/11. Walter Mackesy. Improvements in cams for operating the stamps of gravity stamp mill. 30.6.11.

(P.) 364/11. Samuel Piffard (1), Dan Nicholas (2). Improvements in means for heating water applicable for use with domestic stoves. 30.6.11.

(C.) 365/11. Chaimsonovitz Prosper Elieson. Improvements in clocks, tell-tales, alarms and the like. 30.6.11.

(C.) 366/11. William Ernest Laycock. Improved apparatus for testing vacuum brakes. 30.6.11.

(C.) 367/11. Kurt William Otto Schweder. Improvements in means for dividing and distributing a stream of pulp or other mixtures of solids and liquid. 1.7.11.

(C.) 368/11. Thomas Fisher. Improvements in hot water geysers and the like. 3.7.11.

(P.) 371/11. William Thomas Bell (1), Arthur John Wood Graham (2). Improved construction of apparatus for automatically controlling power driven air compressors. 5.7.11.

(P.) 372/11. Edwin Gabriel Harris. Composition for the destruction of noxious fumes and prevention of gassing. 5.7.11.

(P.) 373/11. Augustine Patrick Farnham (1), Jacobus Adriaan Maude (2). Rock drill dust allayer. 6.7.11.

(P.) 374/11. Edmund Paul Millington Jerrard (1), Paul Millington Jerrard (2). Improvements in internal combustion engines, turbines, pumps and the like and apparatus therefore. 7.7.11.

(P.) 375/11. Edmund Paul Millington Jerrard (1), Paul Millington Jerrard (2). Improvements in motor road vehicles, ships, tramcars, and other machines and apparatus therefore. 7.7.11.

(P.) 376/11. Alfred James Johnson. A machine for wrapping dynamite and other substances such as the machine will wrap. 7.7.11.

(P.) 377/11. Francis White Smith. Machine for sharpening edges of plate scrapers. 7.7.11.

(P.) 378/11. Francis White Smith. Means for changing mortor box screens without hanging stamps up. 7.7.11.

(P.) 379/11. Richard Middleton Simpson. Improvements in centrifugal separating machines. 8.7.11.

(C.) 381/11. Albert Ribbs Taylor. Improvements in rotation devices for fluid pressure operated hammer tools. 8.7.11.

(C.) 382/11. Albert Ribbs Taylor. Improvements in rotation devices for fluid pressure operated hammer tools. 8.7.11.

(C.) 383/11. Milon James Trumble. Process and apparatus for the evaporation of liquid. 8.7·11

(C.) 384/11. Graflich Schaffgotschsche Werke Gesellschaft mit beschraenkter Haftung (1), Valentin Stasch (2). An improved detent for checking runaway trolleys and the like. 8.7.11.

(C.) 385/11. Alfred Andrew Lockwood. Improvements relating to the extraction of the precious metals from ores by the cyanide process. 8.7.11.

(C.) 386/11. William Arthur Hone (1), James William Wilson (2), Cyril Douglas McCourt (3). Apparatus for steam generating, feed water heating and the heating of liquids generally. 8.7.11.

(P.) 388/11. William Arthur Caldecott. Improvements in tube mill linings. 10.7.11.

(C.) 391/11. John Joseph Charley. Improvements in or relating to wheeled vehicles and carriers. 12.7.11.

(P.) 392/11. Gabriel Christian Erasmus Keet. Improved gearing machine by which power can be gained for any purpose. 12.7.11.

(P.) 394/11. James Wilshir. Improvements in hose unions. 12.7.11.

(C.) 395/11. Robert Okell. Improvements in combined gates and nozzle holders for classifying cones and the like. 13.7.11.

(C.) 396/11. Sydney George Sweet. Improvements relating to screw and nut gearing for converting rotary motion into reciprocating motion and *vice versa*, 13.7.11.

(P.) 398/11. Henry Gilbert Hubbart (1), Charles Craig (2). Improvements in apparatus for washing and separating or concentrating metallic ores, minerals and the like. 13.7.11.

(P.) 399/11. Thomas Passe. Improvements in and appertaining to the valve devices of rock drilling machines and the like. 14.7.11.

(P.) 400/11. Hugh Campbell (1), James Grant Gibson (2). Durita flooring. 14.7.11.

(P.) 401/11. Andre Leon Felicien Strauss. Improvements in torches suitable for igniting fuses. 15.7.11.

Changes of Address.

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BROOM, W., *l/o* Johannesburg; Pickstone Mine, Hartley, Rhodesia.

BYERLEY, B. G., *l/o* Bulawayo; P. O. Box 416, Salisbury, Rhodesia.

CARTER, G., *l/o* Denver; Keyton, Penare Road, Penzance, Cornwall.

COCKBURN, P. A., *l/o* Francistown; Bwana M'Kubwa Copper Mining Co., Ltd., N.W. Rhodesia.

CHEW, B., *to* Moodies Fortuna G. M. Co., P. O. Box 124, Barberton.

COMRIE, A., *l/o* Scotland; Broomassie Mine, Abroomassie, Gold Coast Colony, via Sekondi, W. Africa.

DYASON, E. C., *to* "Warrain," Sandown Street, Middle Brighton, Victoria, Australia.

FREEMAN, C. C., *to* Lake View and Star G. M. Co., Fimiston, W. Australia.

HOWE, BEN, *l/o* Sumatra; 30, Bramham Gardens, South Kensington, London.

JOHNS, J. H., *to* Elmfield; Westbury-on-Trym, Near Bristol, England.

KING, A., *l/o* Germiston; P. O. Box 19, East Rand.

NIXON, G. H., *to* Post Office, Pilgrims Rest.

MARRIOTT, H. F., *to* The Central Mining and Investment Corporation, Ltd., 1, London Wall Building, London E.C.

POTHECARY, G., *l/o* Eureka; Nelspruit, Transvaal.

REDFERN, A., *l/o* Brugspruit; Vaalbank Colliery, P. O. Box 67, Middelburg, Transvaal.

RICHARDSON, W. P., *l/o* Swaziland; Horo Mine, Louws Creek, near Barberton.

ROBERTSON. J. R. H., *l/o* Scotland; c/o The Mysore G. M Co., Marikuppam, Mysore State, S. India.

STOKES, R., *l/o* Johannesburg; c/o 5, Summerland Mansions, Muswell Hill, London.

THOMAS, W., *l/o* India; "Menzar," Barncoose, Redruth, Cornwall, England.

VAUGHAN, J. E., *l/o* Johannesburg; Inspector of Mines Office, Maritzburg, Natal.

WISEMAN, E. J., *l/o* Johannesburg; Brakpan Mines Ltd., P. O. Box 3, Brakpan.

WILSON, G. C., *l/o* India; 298 Byars Road, Hillhead, Glasgow, Scotland.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

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Vol. XII. AUGUST, 1911. No. 2.

Proceedings
AT
Ordinary General Meeting, August 19, 1911.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, August 19th, Mr. W. R. Dowling (Vice-President) in the chair. There were also present :—

51 Members: Messrs. H. A. White, E. J. Laschinger, Tom Johnson, Prof. G. H. Stanley, Prof. J. A. Wilkinson, F. W. Watson, A. F. Crosse, W. A. Caldecott, J. R Williams, W. Cullen, A. McA. Johnston (Members of Council), W. J. Abel, B. V. Blundun, T. H. Clesham, L L. Ellis, R. Gascoyne, B. C Gullachsen, J. M. L. Henry, A. B. Inglis, W. W. Lawrie, J. Lea, A. Lilley, G. Lawson, J. E. Metcalf, J. T. Milligan, T. G. Martyn, M. T. Murray, W. J. R. North, S. Newton, E. A. Osterloh, F. S. Pearce, H. W. Pidsley, J. F. Pyles, J. B. Polglase, W. A. Quince, E. Roberts, O. D. Ross, G. A. Robertson, F. L. Simmons, A. M. Stuart, K. K. H. Sartorius, B. C. T. Solly, S. H. Steel, I. Tom, C. Toombs, J. T. Triggs, J. M. Thorburn, W. A. C. Tayler, A. Wilkinson, and John Watson.

21 Associates and Students: Dr. J. L. Aymard, J. P. Beardwood, G. Bulman, W. E. Burrows, J. Cronin, J. I. Coppin, I. B. Dicks, C. A. Damant, F. E. Doble, W. J. N. Dunnachie, C. L. Dewar, P. A. Grumitt, A. King, L. T. Leyson, G. H. Little, H. Lomberg, G. F. Matthews, D. Nicholas, H. Stadler, S. A. Woolf, and W. Waters.

11 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as published in the June *Journal*, were confirmed.

NEW MEMBERS.

Messrs. A. F. Crosse and W. Cullen were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been duly elected as follows :—

CANDY, HENRY STAWELL, P. O. Box 2216, Johannesburg. Mill Manager.

CRAKE, RALPH VANDELEUR, B.S.A. Explosives Co., Ltd., Dynamite Factory, Modderfontein. Chemist.

DAMANT, GEORGE CUMMING, Glynn's Lydenburg, Ltd., P. O. Box 24, Sabie. Mine Manager.

MANNERS, WILLIAM GEORGE, Lourho Buildings, Salisbury, Rhodesia. Mechanical Engineer.

PEARCE, ERNEST VIVIAN, Transvaal Consolidated Land and Exploration Co., Ltd., P. O. Box 4303, Johannesburg. Metallurgist.

REID, WILLIAM, Main Reef West, Ltd., P. O. Box 2, Maraisburg. Surveyor.

The Secretary notified that since the last meeting of the Society the following Associates and Students had been admitted by the Council :—

As Associates—

ASCHMANN, JAMES JOHN, Waterfall Syndicate, Penhalonga, Rhodesia. Cyanide Manager.

BECKER, EDWARD HUGH, Eldorado Mine, via Salisbury, Rhodesia. Amalgamator.

CADDY, JAMES PASCOE, Lake View and Star Gold Mine, Kalgoorlie, Western Australia. Mining Engineer.

COWLES, EUGENE POMEROY, Princess Estate and G. M Co., Ltd., P. O. Box 114, Roodepoort. Assistant Surveyor. (*Transfer from Students' Roll.*)

EDWARDES, ROY DES ANGES, P. O. Box 4873, Johannesburg. Cyanider. (*Transfer from Students' Roll.*)

IMPEY, ROBERT LANCELOT, Consolidated Gold Fields Laboratory, P. O. Box 108, Germiston. Metallurgist. (*Transfer from Students' Roll.*)

MACARTHUR, J. H., Globe and Phoenix G. M. Co., Ltd., Queque, S. Rhodesia. Mine Captain.

MAINE, PERCY CECIL DE, Simmer Deep, Ltd., P. O. Box 178, Germiston. Assistant Reduction Works Sampler.

MALY, FRANK, Marysville, Montana, United States America. Cyanider.

MCLEAN, CALVIN STOWE, Lonely Reef G. M. Co., Private Bag, Inyati, S. Rhodesia. Mine Captain. (*Transfer from Students' Roll.*)

STEPHENS, ALEXANDER HERBERT, Van Ryn Deep, Ltd., P. O. Box 225, Benoni. Shift Boss.

TAYLOR, NORMAN SARGANT, French Bobs Mines, Ltd., Louws Creek, Transvaal. Mine Pupil. (*Transfer from Students' Roll.*)

As Students.—

*Beardwood, Joseph P., Messrs. Littlejohn & Whitby, P. O. Box 2028, Johannesburg. Assay Learner.

*Doble, Frank Edward, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

*Nellmapius, Ernest Harold, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

*Rowe, John Lofflin, Sub-Nigel, Ltd., Nigel. Assay Learner.

*Spence, Eric Hudson, S.I.M.M., City Deep, Ltd., P. O. Box 1411, Johannesburg.

* Renewals.

General Business.

THE INSTITUTE OF CHEMISTRY.

The Secretary read the following letter :—

Mr. J. McCrae (*Member*): As I have recently been consulted in a number of cases respecting the examinations of the Institute of Chemistry of Great Britain and Ireland, and as I am led to the conclusion that an idea prevails that candidates can only be examined in Britain, I solicit the opportunity of bringing, through the medium of the Chemical, Metallurgical and Mining Society of South Africa, to the notice of prospective candidates the fact that arrangements can be made for the examinations to be held locally.

Before accepting candidates for examination the Institute of Chemistry requires that a prescribed course of study shall have been followed. The Council of the Institute is, however, prepared to consider special applications, and if the merits of the case justify the action the Council is prepared to allow the candidate to proceed to examination.

For the Associateship of the Institute three examinations require to be passed : (a) preliminary, (b) intermediate and (c) final. Certain approved preliminary examinations (University and other educational bodies) are accepted in lieu of the preliminary examination. For the intermediate examination it is required that a candidate shall have regularly attended systematic courses at a recognised Institution in Chemistry, Physics, Mathematics and an optional subject. Candidates who have obtained certain degrees from specified Universities are exempt from the intermediate examination. The examination consists of (a) an examination in General Theoretical Chemistry and (b) an examination in Practical Chemistry.

The final examination is conducted in the following branches :

Branch A.—Mineral Chemistry.
,, B.—Metallurgical Chemistry.
,, C.—Physical Chemistry.
,, D.—Organic Chemistry.
Branch E.—The Chemistry of Food and Drugs, and of Water.
,, F.—Biological Chemistry.

The Fellowship is granted to Associates of three years' standing provided certain conditions have been fulfilled.

The Institute also holds examinations in Chemical Technology. These examinations are open only to Fellows, and certificates are granted to successful candidates.

Information concerning the Institute and its examinations can be obtained from the Honorary Local Secretary, P. O. Box 1080, Johannesburg.

The Chairman : We have five Past-Presidents attending our meeting to-night, and amongst them is Mr. Cullen. You will remember that he left us some months ago, owing to ill health, but he is back again now, and is looking very much better for the change. I am sure that we are all pleased to see him, and I offer him a welcome on your behalf. Mr. Cullen has samples of zinc and lead ribbon which he has offered to show us and make a few remarks upon.

METAL RIBBON.

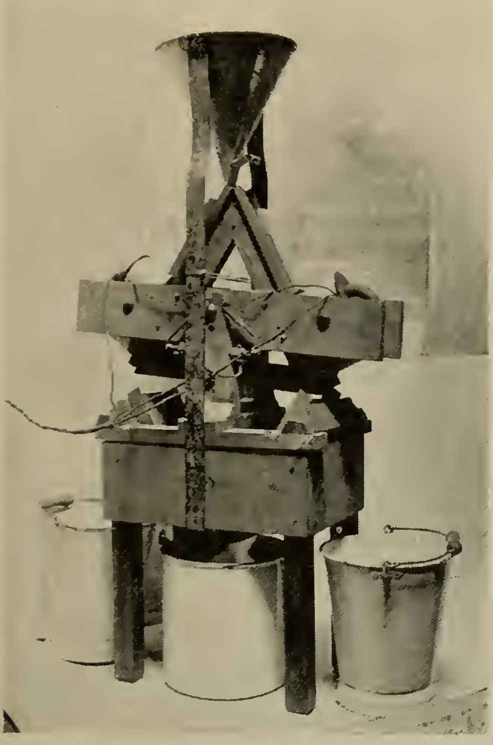
Mr. W. Cullen (*Past-President*): In the January number of the *Journal* Mr. Crosse referred to the ribbon form of zinc and described how this product was made. I did not happen to see this number of the *Journal* until some months afterwards, and in the interval had got interested in the same matter. The sample which I exhibit was made in my presence by an exceedingly simple plant, which Mr. Crosse described, but I understand that unless the ribbon can be made in a less brittle form it is of very little use for precipitation purposes. The other specimen which I show is that of ribbon lead, which is made in exactly the same way, only it can be used for box lining. The main application, however, for the lead in the ribbon form is for jointing water pipes. Hitherto this has always been done by the molten metal, but it is claimed for the ribbon lead that by merely hammering it in without tow, a very much better joint is formed. This point may be of interest to our engineering friends.

AN ELECTRO-MAGNETIC SEPARATOR.

The Chairman : I will now ask Mr. Crosse to show us the working of the model of the electro-magnetic separator which he has brought here to-night.

Mr. Andrew F. Crosse (*Past-President*), then exhibited and described the model in operation as follows :—

The machine comprises a triangular-shaped rocker, upon the sloping sides of which are attached trays or shoots to receive the ore from a feeder hopper above. Over each tray is fixed an electro-magnet, so that as the material falls down the tray the magnetic portion is, of course, immediately arrested. The problem is to discharge this magnetic portion, and to effect this, it is contrived that the frame supporting the trays shall be slowly rocked up and down. The current is automatically cut off from one of the magnets as a result of this alternating motion, and the collected material falls back upon the tray, from which it was originally attracted. At about the same moment this tray is withdrawn from below the feed hopper discharge and its lower edge moves over a special shoot, into which the concentrate is delivered. Upon the other side of the rocker the magnet is in action, and the reverse process is being carried on. The only power required, apart from the current used, is that necessary to effect the rocking of the apparatus, and since this is practically balanced, the effort demanded is very small.



NEW GRADING MACHINE.

Mr. B. V. Blundun (*Member*), after giving a practical demonstration of the working of the machine said:—In this machine two complete samples are graded at once, one set of sieves being clamped in the cage on each arm. Two distinct motions are given, a lateral motion to the arms and a rotary motion to the cages carrying the sieves, by which means the sand is kept constantly moving across the face of the sieve. There is also a distinct bump at the beginning and end of the stroke. The best results appear to be obtained by a speed of about 200 shakes a minute, 4 minutes being sufficient in most cases to get a good grading. The results obtained, tested against careful hand grading, have been remarkably accurate, and as many as 46 gradings have been done in a day by one learner.

SINKING OPERATIONS AT THE NORMA SHAFT, KATTOWITZ, UPPER SILESIA.

By B. C. Gullachsen, M.I.M.E., (*Member*).

Object of Shaft.—The Kattowitzer Aktiengesellschaft decided to sink a new shaft, to be known as the Norma, to be used for the purpose of an upcast fan shaft for the ventilation of the easterly extension of the Ferdinand Colliery workings, for sand filling pipes and also for the purpose of lowering timber to the workings in that portion of the mine. The shaft was to have a finished inside diameter of 3 metres (9 ft. 10 in.).

Preliminary Work.—The chief feature which decided the selection of the shaft site was its position with regard to the workings below, but it was also desired to have it conveniently near a railway siding at the surface.

The site having been decided upon, preparations were made for the putting down of a bore hole in order to ascertain the character of the ground to be passed through, which bore hole would also be used for the unwatering of the shaft during the sinking operations, the water to pass down the bore hole into the workings below, and ultimately to be pumped to the surface from another shaft, thus avoiding the necessity of pumping gear in the small shaft.

Bore Hole.—The bore hole was put down by the use of a simple free falling cutter bore machine, the hole being commenced with a diameter of 45 cm. (17·7 in.) and gradually reduced to 30 cm. (11·8 in.). It was lined throughout with iron pipes, and passed through the following:—

Section.	Metres.		Feet.
Subsoil and clay	2·00	=	6·56
Sand	5·00		16·40
Grey clay	1·00		3·28
Damp sand	12·00		39·36
Quicksand and bands of clay	25·80		84·62
Soft sandstone	2·80		9·18
Compact sandstone ...	15·61		51·20

Section.	Metres.	Feet.
Strong sandstone with shale bands	6·35	20·82
Sandy shale	9·20	30·17
Strong compact sandstone ...	46·90	153·83
Shale	3·00	9·84
Coal (Hoffnungs seam) ...	1·70	5·57
Shale	6·20	20·33
Sandstone with shale bands ...	6·85	22·46
Coal (Blucher seam) ...	1·50	4·92
Shale and sandstone ...	29·50	96·76
Shale	10·29	33·78
Coal (Fanny seam) ...	7·50	24·62
Total	193·20	633·70

Sinking (First Attempt).—The bore hole not having indicated the presence of any large quantity of water in the quicksand, it was hoped that the shaft could be sunk through the sand by the use of timber only, until the sandstone was reached, where a good walling bed could be obtained and from which the brickwork could be built up, and with this end in view the shaft was commenced on February 1st, 1909.

No difficulty was experienced for the first 15 metres (49·21 ft.), the timber held fast and showed no signs of breaking. The timbering of the shaft was made hexagonal, being 4·2 metres (13·8 ft.) between opposite faces of the hexagon. Round timber was used and placed horizontally skin to skin, and at the angles of the hexagon vertical supports were placed, being held in position by cross beams. Three iron girders, 42 cm. (16·4 in.) wide by 42 cm. (16·4 in.) high, and of 16 mm. (0·63 in.) metal were placed across the sides of the shaft, 1 metre (3·28 ft.) from the surface, with the object of carrying the weight of the timbering. At intervals of 1 metre (3·28 ft.) iron rails as stringers were hung from the girders, which were at frequent intervals bolted to the timbering. By these means a depth of 20 metres (65·62 ft.) was reached, but during the sinking of the last 5 metres (16·4 ft.) the timbering showed signs of movement, and broke in several places, this was renewed and strengthened, and sinking continued. The bottom of the shaft was now in sand containing sufficient water to make the work very difficult, the sand constantly running into the shaft from the sides. Progress was very slow, and the timbering required constant attention, the iron rails also bent under the pressure, portions of the timbering gave way every day, making the work in the shaft bottom of a dangerous nature, as well as causing constant interruptions, in order that the men might assist in the repairing of the timbering. The iron girders were next noticed to have bent considerably, and further breakage of the timbering to have taken place. On November 20th, 1909, the shaft having reached a depth of about 25 metres (82 ft.) it was decided to cease work for the time being. The entire abandonment of the shaft was out of the question, as during the time the 25 metres (82 ft.) had been sunk from the surface, work had also been carried on from below, with the result that at this period of the work 120 metres (393·72 ft.) had been completed, and ultimately 132 metres (433·1 ft.) were completed below ground, thus leaving 61·20 metres (200·8 ft.) to be sunk from the surface, a depth of 193·20 (633·9 ft.) in all being the depth to the Fanny seam.

Sinking (continued) Second Attempt.—Early in the year 1910 it was decided by the company that the best method of sinking through the sand was by means of iron tubbing to be forced down by hydraulic jacks, and arrangements were accordingly made.

The surface around the shaft mouth had sunk and cracked in many places. and the old wooden headgear was tilted to one side; this was removed and the ground around the shaft mouth filled up with ashes and stones, 60 truck loads being used for this purpose. Brick foundations were then built and a new iron headgear erected. The work of clearing the old shaft and retimbering it was then commenced. As the old timbering was withdrawn it was replaced by new, but on this occasion the shaft was made somewhat wider, namely, 5 metres (16·4 ft.) from side to side, and in the form of an octagon. The timbering for the first 12·6 metres (41·34 ft.) was built up on an altered system, the horizontal timber (wall plates) being 40 cm. (15·75 in.) apart, supported in the middle and each end by small props (studdles). Behind the horizontal timber planks (lagging) were placed 3 cm (1·2 in.) thick. At each angle of the octagon round timber was placed and the whole held in position by cross pieces, passing from one side of the shaft to the other and firmly wedged. At a depth of 12·6 metres (41·34 ft.) a bed was made for the timber to rest upon, below this depth the shaft was continued with the same diameter as in the old shaft, and also the timbering was now placed skin to skin, and thus another 7·7 metres (25·26 ft.) were successfully timbered. At this point it was decided to proceed no further with this system, but commence with the installation of the tubbing. The renewed timbering now reached a depth of 20·3 metres (66·6 ft.), thus leaving 5 metres (16·4 ft.) of the old timbering below; this remaining 5 metres (16·4 ft.) was then filled up, 4 metres (13·1 ft.) with ashes and small stones, on the top of this a cement floor was made 50 cm. (19·68 in.) thick, on the top of this a timber platform of 11 cm. (4·33 in.) thickness was laid, an opening being left in the centre of all three for the passage of the bore hole pipes by which means the shaft was kept free from water, and from this platform the tubbing was built up.

Before proceeding to place the cutting shoe and tubbing in position something was necessary to ensure that the tubbing would sink vertically; for this purpose old shaft guides were employed, 14 cm. × 18 cm. (5·5 in. × 7·6 in.) thick and over 8 metres (26·25 ft.) long. Six of these guides were bolted to the shaft timbering, care being taken to place them exactly vertical, so that when the tubbing was built up only about 1 cm. (0·394 in.) play was left. Seven bolts 32 mm. (1·26 in.) diameter were used for each guide, their heads being counter-sunk in the guides.

Cutting Shoe.—The cutting shoe (Fig. I.) was then built up on the wooden platform, and consisted of three cast iron segments, each 70 cm. (27·56 in.) high and 16 cm. (6·3 in.) wide, the metal being 45 mm. (1·77 in.) thick. The toe of the cutting shoe was provided with a steel cutter 20 mm. (0·79 in.) thick, as shown (Fig. I.). Each segment was bolted to the other by two bolts 36 mm. (1·41 in.) diameter, six in all, each ring was bolted to the next ring above by 36 bolts, 32 mm. (1·26 in.) diameter. The lower halves of the cutting shoe were also secured to each other by iron plates and six bolts 32 mm. (1·26 in.) diameter.

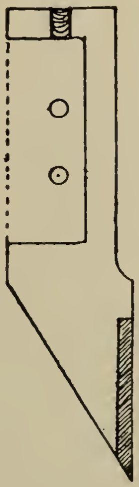


Fig. I.

Tubbing.—Above the cutting shoe the first ring of tubbing was placed, each complete ring consisting of six segments, each weighing 18 cwt. Each segment (Fig. II.) was 1 metre (3·28 ft.) high and 160 mm. (6·3 in.) wide, the thickness of the metal being 45 mm. (1·77 in.). The segments were bolted together by four bolts 29 mm. (1·14 in.) diameter, and to the next ring of tubbing above by 6 bolts 29 mm. (1·14 in.) diameter, or 36 bolts for the complete ring. Between each joint sheet lead was placed. Several rings of tubbing were built up in this manner until the wider portion of the shaft was reached. At this point the brickwork to carry the anchor ring and pressure ring was commenced. The brickwork was commenced with a thickness of 80 cm. (31·5 in.), and the top portions of the wooden guides were built into it, bolts 40 cm. (15·75 in.) long passed through the guides, the bolts having iron plates at the end which were built securely into the brickwork. In order to act as an additional foundation to the brickwork, 16 old rails from the railway were employed (each rail was from 4·5 metres (14·76 ft.) to 5 metres (16·4 ft.) long), and they were driven into the sides of the shaft, at equal intervals around the shaft for a distance of 3·7 metres (12·14 ft.) to 4·2 metres (13·78 ft.), thus leaving 80 cm. (31·5 in.) protruding over the brickwork which was then built up round them. This first length of brickwork was 1 metre (3·28 ft.) high, and upon it rested the anchor ring.

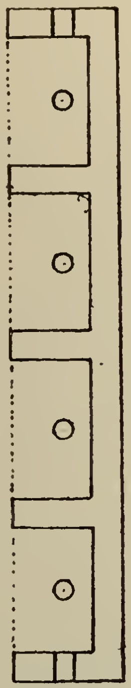


Fig. II.

Anchor Ring.—The anchor ring (Fig. III.) consisted of four segments (each weighing 1½ tons) bolted together by eight bolts, each bolt 48 mm. (1·89 in.) diameter. In each segment were four vertical holes 70 mm. (2·756 in.) diameter, equal distances apart, and through these holes passed the anchor bolts. The segments were 80 cm. (31·5 in.) wide and 30 cm. (11·91 in.) high, and after having been placed in position and bolted together they were filled with cement.

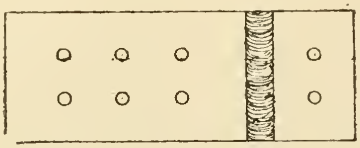


FIG. III.

Anchor Bolts.—The anchor bolts, which were made of wrought iron 65 mm. (2·66 in.) diameter, were each in two parts; the upper and lower, being joined together above the pressure ring by a screwed coupling 20 cm. (7·87 in.) long and 12 cm. (3·72 in.) outside diameter. The total length of each bolt was 11·16 metres (36·6 ft.), the lower portion 5·66 metres (18·57 ft.), and the upper 5·5 metres (18·04 ft.). The lower portions of the bolts were passed through the holes in the anchor ring, underneath which openings had been left in the brickwork, in order that the iron cotters or holding pieces could be placed in position through the cotter holes formed in the bolts, this having been done the openings were bricked up. The brickwork was then continued, being built around the anchor ring to a thickness of about 1 metre (3·28 ft.). Above the anchor ring the thickness of the brickwork was gradually reduced to 80 cm. (31·5 in.). The anchor bolts during the time they were being built round were held in their correct vertical position by a wooden template placed where the pressure ring would come. The brickwork was continued for a height of 4·4 metres (14·44 ft.), the last 50 cm. (19·68 in.) of it being gradually widened out. The surface of the brickwork was then carefully levelled with cement and prepared to receive the pressure ring.

Pressure Ring.—The pressure ring was built up of four cast iron segments, each 80 cm. (31·5 in.) wide and 40 cm. (15·75 in.) high, the weight of each segment being nearly 2 tons The segments were bolted together by ten bolts, 48 mm. (1·89 in.) diameter. Through each segment were four vertical holes, 70 mm. (2·75 in.) diameter for the passage of the anchor bolts (Fig. IVA.).

Pressure Shoe.—The pressure shoe was made up of four cast iron segments (Fig. IVB.) and was bolted to the pressure ring as shown, four bolts, 36 mm. (1·41 in.) diameter for each segment or 16 in all. The segments of the pressure shoe were also bolted to each other by two bolts, 36 mm. (1·41 in.) diameter, or 8 in all. The pressure ring was first laid in position on the brickwork and bolted together, the pressure shoe was next bolted to it, after this was completed the upper portions of the anchor bolts were connected to the lower portions, which projected about 10 cm. (3·93 in.) above the pressure ring. The connection was made by a screwed coupling, which was screwed down fast on to the pressure ring, and then the upper portion of the anchor bolts screwed into the upper half of the coupling.

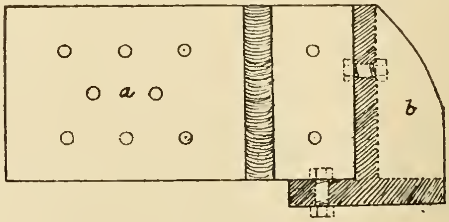


FIG. IV.

Behind the pressure ring the shaft was widened out, and the brickwork made 1·8 metres (5·9 ft.) wide, or 1 metre (3·28 ft.) behind the pressure ring, the brickwork was continued upwards in a wedge shape, and gradually reduced in thickness until it was 80 cm. (31·5 in.) thick, at which thickness it was continued until within 2·5 metres (8·2 ft.) of the surface, from where it was gradually widened until at the surface it was 2·5 metres (8·2 ft.) wide. The anchor bolts were built into the brickwork, large iron washers were placed over the anchor bolts and bolted tightly down on to the brickwork. The washers were 25 cm. (9·84 in.) diameter and 8 mm. (0·31 in.) thick. The preparations for sinking through the sand were now all complete, as shown in Fig. V., and on May 17th, 1910, sinking was resumed.

The first portion of the sinking consisted of removing the timber platform on which the cutting shoe rested, and the cement floor below, the tubbing, due to its own weight, sinking as the cement was broken from under it. The sinking through the 4 metres (13·12 ft.) of ash, with which the bottom of the old shaft had been filled in was next proceeded with, the tubbing gradually sinking owing to its own weight, but to assist it to do so evenly one hydraulic jack was occasionally used, when one side of the tubbing showed signs of sticking, the hydraulic jacks being placed between the top ring of tubbing and the pressure shoe. When only 1 metre (3·28 ft.) of ash remained to be removed, more hydraulic jacks were used in order to force the tubbing in advance, and about this time water and sand commenced to flow into the shaft from below. Sinking now became difficult and slow, as although more jacks were applied with the object of forcing the cutting toe as far as possible in advance, it was found to advance very little, this being due to the fact that the old timber and iron rail which had been used to support the old shaft had got forced inwards below the cutting toe and thus very much hindered its progress. To sink as deep as the toe of the cutting shoe was now nearly impossible on account of the continued inflow of sand, but in order to do so, and thus remove the obstruction, a strong timber platform was built and placed on the bottom of the shaft, the platform being circular and just fitting the interior of the tubbing. In order to hold the platform down strong timber was placed across the shaft and fixed in the tubbing, from these cross pieces, vertical timber descended to the platform and were wedged tightly, the platform thus preventing the further inflow of sand. The platform was made in several sections, and sinking now proceeded by removing one section at a time, and sinking in the small opening thus made, the section was then fixed a few centimetres lower down, and the next section treated in a similar manner. By this slow means sinking was continued until many of the obstructions to the progress of the cutting toe had been removed, and the jacks were then able to overcome the resistance of the others. After this the work proceeded more rapidly, as the tubbing could now be forced well in advance of the sinking, and the timber platform removed. The sand did not contain so much water as had been anticipated, and was of a clayey nature, which prevented the rapid flow of the sand into the shaft. No winding of the material was necessary as all of it was run through the bore hole in the centre of the shaft, and used as filling material for the old workings below.

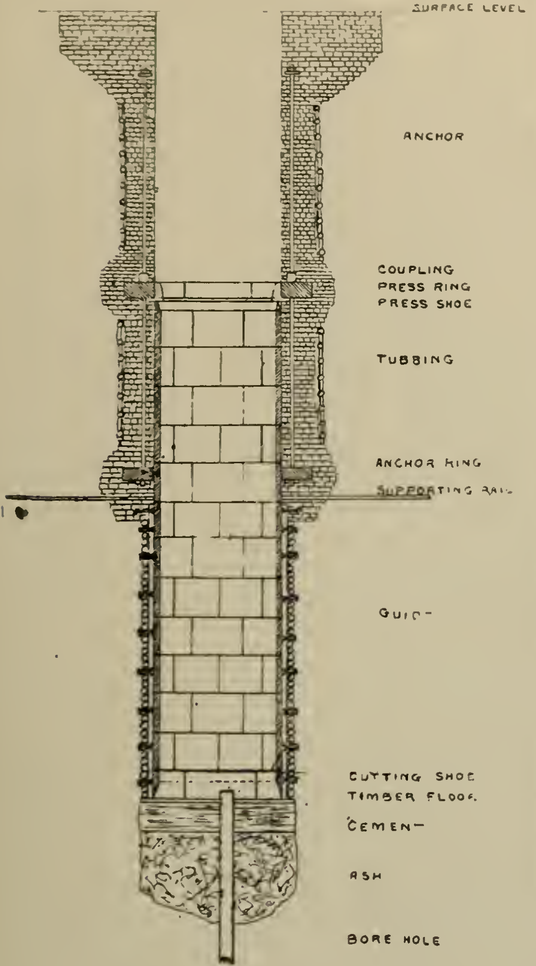


Fig. V.

As sinking proceeded the sand became more and more of a clayey nature, and contained less and less water. The forcing of the tubbing through this material became a matter of difficulty, but so little water was now coming into the shaft that it was found possible to sink as deep as the cutting toe, and by removing the material from under it its advance was continued, but after advancing a short distance in this manner it was found impossible to force the tubbing any further down, due to the friction of the clayey material on the outside of the tubbing. Six hydraulic jacks were used, but the only result was to crack four segments of the tubbing. Under these circumstances it was decided to advance without the tubbing, which now reached to a depth of 38·7 metres (126·9 ft.), and after sinking had been continued another 2·3 metres (8·54 ft.) the clay was of such a strong nature that preparations were made for brickwork to be built up from it. This brickwork was 1 metre (3·28 ft.) thick, and built up until it came under the cutting shoe, when it was 0·75 metres (29·5 in.) thick. Between the brickwork and the cutting toe a ring of iron 15 cm. (5·9 in.) wide and 10 mm. (0·39 in.) thick was placed, and it was found possible to force the tubbing tightly down upon it.

The work was now practically completed and the remaining 20 metres (65·62 ft.) were sunk in the usual manner until the connection was made with the finished portion of the shaft below, this remaining portion of the shaft being brick walled throughout and the shaft completed.

Cost.—The following is the total cost of the material and labour used for sinking the upper portion of the shaft down to the walling below the cutting shoe, a depth of 41 metres (134·5 ft.) :—

	Marks.		£	s.	d.
Labour, wages ...	44,158·17	=	2,207	18	2
Smithy work ...	7,219·30		360	19	3
Cost of Tubbing ...	30,508·85		1,525	8	10
Cost of materials, etc	15,081·85		754	1	11
Timbering ...	9,395·35		469	15	4
Total	106,363·52		5,318	3	6

The total cost per metre was thus 2,594·23 marks or £129 14s. 3d., being a cost of about £39 10s. per foot.

Mr. Tom Johnson (*Member of Council*): In proposing a vote of thanks to the author for his paper, I should like to say that the sinkers of some shafts here on the Rand would save a lot of trouble and expense if when they came across soft dykes, etc., they did not try to carry the shaft down full size at once, but took it down in sections, as was the case with the shaft referred to in this evening's paper. I hope our mining men will come along and criticise this paper, though we do not have to do this work to-day: still we might have to. I have much pleasure in proposing a vote of thanks to Mr. Gullachsen.

Mr. K. Sartorius (*Member*): I should say that in taking the water and sand away from underneath they produced a detrimental effect on the sides of the shaft, as the soft ground continued flowing from behind timber, etc., affecting the above lying strata. Certainly the difficulty was overcome finally, but I should have thought that straight sinking with tubbing, and in sections, would have given a much reduced figure for the work, if the flow of sand and water had been discontinued from underneath. I have much pleasure in seconding the vote of thanks.

Mr. E. J. Laschinger (*Vice-President*): It seems a pity that the persons responsible for the sinking of this shaft did not take the advice of experienced engineers first. They started a difficult shaft in the old-fashioned way, and I think they would have been wise if they had benefited by past experience in all parts of the world. They would then have seen the futility of trying to sink a shaft in sand and clay with ordinary timbering methods. There was a similar shaft sunk in South Africa before the war. They had three trials at it, and finally came to the cutting shoe with cast iron tubbing in much the same manner as described by the author in his paper to-night. I quite appreciate that the author deserves the thanks of the Society for giving us this paper.

SOME ASPECTS OF MINE VENTILATION AND TEMPERATURE ON THE WITWATERSRAND.

(*Read at May Meeting, 1911.*)

By J. Whitehouse, B.Sc., M.I.M.E. (*Associate*) and W. L. Wotherspoon

DISCUSSION.

Dr. J. L. Aymard (*Associate*): On a previous occasion I congratulated the authors on their valuable paper, and it is indeed a matter of sincere regret that so little apparent interest has been taken in the subject of ventilation. Whether it comes from indifference or ignorance or both, or whether it is due to some other reason I do not know, but in any case it is extremely regrettable considering the public position this subject has taken.

I want to ask Mr. Whitehouse whether it is not a fact that within the past three months he made a personal visit of inspection to the dust prevention system on the Nourse Mines, and whether he does not agree with the Government officials and others that the system installed there is working excellently and giving splendid results? I want Mr. Whitehouse to answer a straightforward question. What is the reason that other mines, including his own, have not adopted the Nourse Mines system in its entirety. Is it for the reason given some time back, that good men might leave?

AIR LIFT AGITATION OF SLIME PULP.

(*Read at March Meeting, 1911.*)

By Robert Allen, M.A., B.Sc., M.I.M.M.
(Member of Council).

REPLY TO DISCUSSION.

Mr. Robert Allen (*Member of Council*): Mr. C. B. Kingston has contributed some interesting and useful experimental data, which illustrate the rapidity of treatment by air lift agitation of a "slimed" free-milling ore. Although a pulp of ratio 1 to 1 may be treated experimentally, I think that on a large scale it would be impracticable to produce a thicker pulp than 1¼ to 1. I shall read with interest Mr. Kingston's promised description of the Lonely Reef plant.

Mr. Toombs is to be complimented on his lucid description of the oxidation of cyanicides in accumulated slime pulp. His data on experimental cyanide treatment of ordinary pulp and accumulated slime at the East Rand Proprietary Mines are very interesting. It is to be hoped that data of the relative efficiencies of the air lift agitation and decantation plants at the Brakpan mines will be contributed in the near future.

Regarding Mr. Laschinger's criticisms, I am rather at a loss to divine what his objections are. He has said much, but most of it is beside the mark, and when it is reduced to its lowest terms I find that it consists chiefly of the following objections:—

1. I have not proved my case. My paper was not intended to prove anything.

2. I have missed the point in failing "to differentiate between agitation as the simple aim in keeping ore particles in suspension in a fluid and agitation as a means of ultimately bringing into contact ore, chemical reagents and air so as to thoroughly facilitate certain chemical reactions." It seems to me that Mr. Laschinger has missed the point, and if I could make out what it was he felt a difficulty in understanding I should be glad to explain it to him. My paper was not a treatise on the physical and chemical aspects of the subject but, as correctly described by my critic, "a compilation of the history of and present practice in air agitation of pulp," and it appears to me that Mr. Laschinger has cut the ground from under his feet by saying I have not "even attempted" to differentiate. Where there is no attempt there is no failure!

3. My figures are not reliable. The quoted power consumptions, which called forth Mr. Laschinger's scathing remark that "*even* the author's figures are not reliable," are those of F. C. Brown, an engineer of repute, and given as such: and they are, except the fourth which is obviously a slip, quite near enough for practical purposes. Elsewhere amongst his remarks Mr. Laschinger estimates the efficiency peculiar to an air lift vat to range from 70% to 80%, yet here, in order to make the figures appear wrong he makes the efficiency of the air lift vat depend on that of the air compressor with which it happens to be connected—a variable efficiency indeed! It seems just as logical to make the efficiency of a compressor depend on that of the boiler plant with which it is connected.

4. "It requires a certain amount of power to compress air." I think we may consider this criticism as negligible together with the textbook matter which follows.

5. I spoke of the mechanical stirrers of large diameter agitation vats as being generally of the clumsiest construction, and of the necessity of sometimes digging out the paddles. I would point out to Mr. Laschinger that the mechanical agitation gear of a large diameter vat such as I referred to must necessarily be clumsy, especially when it is required to agitate an "all slimed" ore, and even then it is rarely practicable to agitate in such a vat a pulp of less dilution ratio than 3 to 1. In every case of such agitation practically all the coarsest sand, which usually requires most treatment, remains at the bottom of the vat, and it is this that causes troubles by embedding the paddles, immediately the power happens to be cut off, thus necessitating the digging out of the paddles after a short stoppage. Apparently Mr. Laschinger's experience in this direction is limited. It is interesting to note that in the large diameter agitation vats of the City Deep plant mechanical stirrers are not used, nor even the low lift centrifugal pump with its 50% efficiency, but air lifts with an excess of power!

6. I said "the higher the vat compared with the diameter the less the power required." Mr. Laschinger complains that I omitted to mention that this was true only for equal weights of pulp handled. It is evident that I rated the intelligence of my audience higher than did Mr. Laschinger, since I left that obvious fact to be inferred. Amongst his general remarks Mr. Laschinger gives what he calls "a good practical rule of thumb" for estimating the pressure of air required:—"to take the figure representing the total depth of the vat in feet and divide this by two," etc. It might be pointed out that the density of the pulp must be taken into consideration, and as it is evident that Mr. Laschinger is here considering the agitation of water only and not of pulp, which has a variable specific gravity, always higher than that of water, the "rule" is utterly wrong. It is extremely interesting to hear from Mr. Laschinger that "in Rand practice alkaline solutions are always used, quicklime being added for the purpose." This takes me back to the days of my youth!

FUTURE ECONOMIES ON RAND PRODUCTION PLANTS.

(*Read at April Meeting, 1911.*)

By C. O. Schmitt (Member).

DISCUSSION.

Mr. C. E. Meyer (*Member*): As so many members have come forward to discuss the author's valuable paper, and their chief complaint being the lack of information with regard to the value of the waste sample, I have appended here a table taken from the late Mr. Wager Bradford's paper entitled "Limitations of Ore Sorting"* as a reference for the elucidation of the point.

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* Proceedings S.A. Association of Engineers, Vol. ix., Sept. 30, 1903, pp. 61-86.

Mine.	Average Value of Waste Sample from Sorting Station.	Average Width and Value of Partings Sampled in the Stopes.		Highest Value.	Lowest Value.	No. of Samples Taken
	Dwt.	inch	Dwt.	Dwt.		
Langlaagte Deep ...	1·143	32	1·385	27·00	Trace	518
Geldenhuis Deep ...	0·861	not given	1·049	8·50	„	435
Jumpers Deep ...	1·282	„	2·852	25·00	„	105
Rose Deep	0·609	13	0·85	6·00	„	70
Crown Deep ...	1·246	34·5	1·307	27·00	„	200
Ferreira G. M. Co. ...	1·007	17·8	3·065	106·00	„	574

Samples represent for:

Langlaagte Deep
Geldenhuis Deep
Jumpers Deep
Rose Deep
Crown Deep
} the average for 9 months during 1899

Samples for the Ferreira G. M. Co. represent the average for 9 months during 1896 In all cases the partings are the averages for 1898.

As Mr. Wager Bradford pointed out in his valuable paper, the formation of reef from the Roodepoort section to the Central Rand varies much, and the value of the partings is very high in some cases. In the former the gold is contained in a very narrow leader, often not more than 2 in. wide, making it necessary to carry the stope to break a high percentage of barren rock. In this case a high percentage of sorting would be both practicable and profitable.

On the mines of the Central Rand, however, the reef is split into many stringers, the intermediate partings being more or less auriferous. The partings having been formed and mineralised at the same time as the reef, these partings would carry value depending upon the grade of the ore.

It seems to me that the author has these facts to contend with in his proposals :—

1. Inefficiency of labour.

2. Insufficiency of labour. Labour for sorting might be more profitably employed elsewhere especially in breaking rock underground.

3. The value of waste; inefficiency of the labour employed might lead to valuable ore being discarded.

4. The probable employment of artificial light for sorting, owing to the possible running of night shifts with the increased sorting proposed.

The tendency to-day being towards the crushing of very large tonnages, bringing within the pale of profits low-grade material to be crushed, makes this proposal of Mr. Schmitt's quite contrary to the principles adopted to-day, as the life of the mine would suffer.

The paper is an exceedingly lengthy one and must have involved a great deal of work, yet I think it lacks consistency as some very important points which have direct bearing on the subject have been neglected.

THE TESTING OF TRANSVAAL COALS.

(*Read at June Meeting, 1911.*)

By M. Thornton Murray, M.Sc. (*Member*).

DISCUSSION

Prof. J. A. Wilkinson (*Member of Council*): Within recent years the subject of coal testing and analysis has become of great importance, largely due to the fact, that the conservation of the natural resources of a country has occupied a place in modern thought. In consequence of this engineers and others have begun to study the complete utilisation of the minerals yielded by a provident earth in abundance, the stores of which are diminishing year by year. Of these none appeals to the engineer more than coal, because it is from this mineral that he draws his motive power. Nature has been so lavish in this respect, that fuel economy was of no concern, but with the increasing activities in the world it is not a matter of wonderment that prophecies have from time to time been made, setting a short period as the limit of our resources in this respect. Although for the present we may reasonably overlook these scaremongers, yet we ought on the other hand to be careful to see that we utilise to the fullest extent possible the material won from the earth. The paper before us deals with this subject and requires no apology, since it coincides opportunely with the recent pronouncement of the Government Mining Engineer as to the marvellous extent of the coalfields of this country.

In connection with the paper itself the title would lead one to expect perhaps more than the author has presented. The determination of the calorific value by means of the Berthelot bomb forms the bulk of the contribution and other points of great importance receive but a brief treatment by way of introduction. Thus the author lays himself open to criticism in this regard, having ignored, for example, the question of the proximate analysis of coal and its teachings entirely, although, as he is fully aware, this is even yet, in spite of its lack of scientific basis, the principal method of examination used for passing judgment on a given coal.

The question of sampling calls for little comment, except in so far that it may be stated that this is so intimately connected with the testing of coals, as to make even the best and most careful analysis worthless unless a sample has been rendered to the analyst, which may without doubt be considered to be truly representative of the mass. If a true judgment of the economic value of a fuel is to be obtained, this operation requires to be performed with as much care as the analysis itself. The subject has received little consideration in this Society, but its intrinsic importance is so great and so general in character, that it is to be hoped that those who have had practical experience of this work will come forward and give us some of their results. With the author's remarks most will certainly agree, but emphasis should be laid on the statement made, namely, that the greatest accuracy should be aimed at in order to make the small quantity taken for analysis as representative of the several tons as it possibly can be. In view of the subject matter of the paper it may be stated that calorific value is one of the quantities chiefly affected by sampling. When the sample has been obtained it is then subjected to various tests, the nature of which depends entirely on the use, to which the coal is to be put, whether steam raising, gas manufacture, coking, etc. The author has tacitly assumed in this respect that the coal for which his tests are destined is to be used for the purpose of raising steam, and hence deals entirely with this aspect of the question. The recent introduction of the Berthelot bomb calorimeter in place of the Lewis Thompson instrument into commercial practice on the Rand is sufficient justification for this, since the determination of the calorific value may be considered the most important datum required in this connection. This when translated into what is known as the evaporative factor is a quantity readily appreciated, even by those whose knowledge of calorimetry is insignificant and who cannot be expected to have much understanding of the interpretation of the results of chemical analysis.

With regard to the methods of the estimation of moisture, investigation is, as stated, necessary and I trust that the author's hopes may soon be fulfilled. The sulphur present in coal may be present either as inorganic or organic in kind. The former variety has been generally assumed to be in pyrite, which is undoubtedly often present in large extent, even when not visible to the naked eye. But sulphides of other metals may also be present, as also sulphate of calcium. In one sample of coal, which I examined some years ago, the latter was present in such large quantity that in one place crystals were visible to the naked eye. Although this was in my opinion a rarity, yet it must not be assumed that inorganic sulphur is present exclusively in the form of sulphides, which on combustion would contribute their share to the heat value. In the case of organic sulphur practically nothing is at present known, but from the thermal point of view it may be assumed without serious error that this will burn away completely to the oxide under the circumstances of bomb combustion.

As far as the actual experimental work is concerned the adjustment and regulation of the stirring is necessary, if the results of two observers are to agree. The ideal in this case is to provide efficient stirring without any appreciable heat effect being produced. In any case the determination of the latter should be made and it would add to the value of his paper if the author would give the results of trials in this direction, in his reply, at the rate of stirring given. In this connection it should be mentioned, that the stirring of the water in the jacket should not be neglected; but in this case also the heat effect must be carefully watched. It is pertinent here to remark that the methods employed by Rumford, namely, of keeping the water in the jacket approximately at the mean temperature of those attained during the experiment in the calorimeter would be difficult of attainment in commercial practice and this would counterbalance any of the doubtful advantages which might accrue. Adiabatic calorimetry* also may be ruled out of court in determinations such as these as it would be impracticable under ordinary conditions to produce the same rise in the temperature in the jacket as in the calorimeter at the same time. Under the heading "Insulation from Air Currents," sufficient stress has not been laid upon the necessity of a thermal equilibrium between the instrument and its environment. The greatest possible care should be exercised in protecting it during an experiment from any external sources of heat loss or gain, a fundamental precaution in all thermal measurements of this nature. If possible a small room should be set aside for this purpose, the temperature of whose walls should not be subject to great variations. Sufficient care has not been exercised locally in this respect, which would account to some extent for variable results with the same coal. The formulae given for calculating the calorific value of coal from the amount of the main combustible elements present will not have, as indeed the author states, any commercial significance chiefly because the ultimate analysis of a coal is an operation outside the range of ordinary analyst's work. It therefore behoves commercial analysts to take as much care as possible in the direct experimental determination of the heat value, more particularly with respect to the elimination of or correction for all errors, should their elimination be impossible. With regard to the fusibility of the ash and the formation of slag, this is a point which is worthy of extended study owing to the large ash percentage in our coals. It should be remembered, however, that although the temperature of combustion is in the neighbourhood of 1,500° C., yet this temperature is only existent for an indefinitely short space of time, and unless the acidic and basic constituents of the ash are present in approximately the right proportions to form a low melting point slag, it would be most improbable that the thermal effect from this cause would be sufficiently great to make any appreciable difference to the result. The author is, however, quite right to point out the effects that might arise from this source in such coals as those of the Transvaal, and at the end of each experiment the cup should be carefully examined to ascertain if complete combustion has occurred, and secondly if slag formation has taken place to any extent in which case it will be necessary to make due allowances ascertained by separate experiments on the ash. *Two recent papers entitled "Recent Progress in Calorimetry" emanating from the Geophysical Laboratory of the United States Geological Survey, by W. P. White, are worthy of attention by all those interested in this subject. In conclusion, I think you will agree that our thanks are due to Mr. Murray for his very opportune paper, and I trust that his object of promoting full discussion on this subject will be duly fulfilled.

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* *Cf.* Transactions of the Chemical Society, London. Faraday Lecture by Prof. T. W. Richards, June, 1911, Vols. 99 and 100, p. 1213.

Metallurgical and Chemical Engineering, Vol. ix., No. 4 p. 202, April, 1911; also Vol. ix., No. 6, p. 290, June, 1911.

EXPERIMENTS IN THE TREATMENT OF ACCUMULATED ORE SLIME BY AIR LIFT AGITATION.

(*Read at July Meeting, 1911.*)

By J. E. R. Adendorff, F.C.S. (Member).

DISCUSSION.

Mr. Andrew F. Crosse (*Past-President*): The author's paper on the experiments which he sometime ago carried out on accumulated ore slime, interested me very much, as in the early part of last year I had carried out some tests on what I feel certain must be the same slime. Before describing my results I should like to draw attention to an interesting fact, which is, that when a solution of cyanide of potassium is mixed with a neutral solution of sulphate of lime, a certain amount of alkaline cyanide is dissociated and free hydrocyanic acid is given off, wherefore it is necessary to have an excess of protective alkali. Sulphate of magnesium has a still more powerful dissociating effect on the cyanide of potassium; in fact, to a certain extent the following reaction takes place:

$$2KCy + MgSO_4 + 2H_2O = 2HCy + MgH_2O_2 + K_2SO_4.$$

Sulphate of magnesium is very common in many gold ores and accounts for a large proportion of cyanide decomposed, though these ores have a neutral reaction. After neutralising the free acid in accumulated slime there is a considerable amount of sulphate of lime present; I have even come across saturated solutions of calcium sulphate containing about or slightly over 0·2% calcium sulphate.

I made several tests on what I believe to be the same slime referred to by Mr. Adendorff. I took care to oxidise the slime after adding an excess of lime. Precipitated ferrous hydrate is soon converted into ferric hydrate during the process of air agitation. After five and a quarter hours treatment I obtained by my method—that is washing during agitation—a residue containing 0·21 dwt. per ton fine gold, the original slime assaying 2·4 dwt. fine gold per ton. There is no doubt in my mind that in many instances a quicker solution and higher extraction are obtained by displacing the solution during air agitation.

In Messrs. Julian and Smart's work on "Cyaniding Gold and Silver Ores" they go fully into the question of fresh *versus* used solutions in chapter ten. In treating highly ferruginous slime the benefit obtained by washing during agitation is very marked, and I believe the same benefit is shown in treating slimed pyritic concentrate.

THE ASSAY OF USED PLUMBAGO CRUCIBLES FOR GOLD.

(*Read at May Meeting.* 1911.)

By John Watson, F.I.C (Member).

DISCUSSION.

Mr. L. J. Wilmoth (*Member*): Assayers, I feel sure, will be grateful to Mr. Watson for his contribution, which enlarges our knowledge of the most interesting side of a mine assayer's work—the assay of bye-products. While in one sense Mr. Watson has done assaying a service, I think in another way he has done rather the opposite in recommending so complicated a method for so simple a problem. Taking the paper in detail, the first point most noticeable is the brief reference to Mr. Crosse's paper in September, 1897, and the total ignoring of Dr. Loevy's paper in January, 1898, and a contribution in February, 1899, of Mr. R. Dures to a paper by Mr. Lofts. I refer to these, as Mr. Crosse's paper seems to be the only one that has influenced Mr. Watson's work. Mr. Crosse describes a method whereby he takes a given quantity of graphite and mixes it with manganese dioxide, places the two in a crucible, calcines and then without withdrawing the calcined mixture adds flux (no mixing is mentioned, and obviously not intended, as the mass was often partly fused) and finishes off the fusion. However, in replying to the discussion, Mr. Crosse withdraws the method as unreliable, and supports the scorification. I give these details as they have a distinct bearing on Mr. Watson's method. Undoubtedly the weak points in Mr. Crosse's method were the calcination in the crucible and the adding of flux without the grinding and thorough mixing of the calcined charge with the added flux. Mr. Watson modifies these by first conducting his calcination in a roasting dish, and then by grinding the residue carefully and mixing it thoroughly with his flux.

To the mine assayer the paper offers no simplification of work but rather the opposite, besides the method used is based on an older method withdrawn by its originator as being unreliable. Most assayers on the Rand use a direct fusion method, and one that can be fitted in easily with the usual routine. This method can be attributed to Mr. R. Dures, who gave in February, 1899, a flux for direct fusion as used at the Assay Office of The Rand Central Ore Reduction Company. The charge is as follows: Plumbago Crucible 5 gm., Soda 25, Borax 15, Litharge 85, Nitre 7 and Silica 10 gm. Various modifications of this flux will be found in use, some assayers using a little more nitre, some eliminating it and using red lead as mentioned by Mr. Whitby. Personally, I consider the elimination of nitre by the use of red lead, and the careful washing with a mixture of charcoal and litharge, to be the only material improvement to date on Mr. Dures' method. The use of a wash is not peculiar to any assay, but should be general in the assay of any rich material. In the past we evidently had a division of opinion as to the relative merits of the methods by scorification or by crucible. Mr. Crosse's method was an attempt to uphold the crucible assay. However, though the crucible assay holds the day, a complicated method such as Mr. Watson's cannot be its mainstay, and improvement rather lies in the further development of Mr. Dures' charge.

Mr. Whitby mentions metallics. These are usually eliminated before assay, for otherwise an assay of this material is almost impossible. A further remark of Mr. Whitby's in regard to the assay value of plumbago crucibles after scraping never assaying over 4·0 oz. is interesting. This has not been my experience, as for years I have noticed that in no case has a sample assayed under 9·2 oz. or over 9·5 oz. This fact has struck me as being so curious as to suggest some chemical explanation.

Before closing these remarks I would like to give a few figures comparing the results in assaying graphite crucibles by scorification and by a direct fusion with a charge similar to that of Mr. Dures. The average of six scorifications with 0·2 A T of crucible; gold = 7·23 oz.; silver = 12·30 oz. or a proportion of 1 of gold to 1·70 of silver. The average of six fusions with 0·2 A T of crucible, gold = 9·23 oz.; silver = 11·45 oz. or a proportion of 1 of gold to 1·23 silver. In view of my general experience with the assay of this material, I think for gold a direct fusion by Mr. Dures' method, and for silver a scorification with from 0·1 A T to 0·2 A T is most satisfactory. The author, I trust, will take this criticism in the spirit it is meant—an earnest desire to do my best to further our knowledge of assaying.

REPLY TO DISCUSSION.

Mr. John Watson, F.I.C. (*Member*): The discussion which has taken place on my paper proves, I think, that the subject was fully ripe for fresh debate. I sincerely thank all who have made contributions. It was a source of particular pleasure and satisfaction to me that Mr. Crosse was present and took part in the discussion, also that the assayers of the Simmer and Jack and the Robinson, two of our largest individual mines, sent their methods. I also heard by post from Dr. Garny, who had much experience in assaying this material in the days of the Rand Central Ore Reduction Co. He kindly draws my attention to the remarks of his then assistant, Mr. R. Dures, in the discussion of Mr. Lofts' paper on "The Assay of Bye-products."*

I was very pleased to peruse Mr. Wilmoth's contribution, which has just arrived. I plead guilty to having overlooked the short contributions of Dr. Loevy and Mr. Dures in Vol. II. I was present when Mr. Crosse read his paper in September, 1897, and not able to be present on the other two occasions. My journals were looted during the war, and it was not till after I had read my paper that I came across the two references mentioned above. It must be borne in mind that Vol. II. (our largest volume) has a very inadequate index, *e.g.*, no author's names are given. The method I described has been found fault with by three of my critics (Messrs. Whitby, Wilmoth and Alfred M. Thomas) on the ground that it is too complicated. They recommend direct fusion of portions of 0·1 or 0·2 assay ton. The first sample of this material that I ever had occasion to assay was tried by direct fusion: the results obtained were very variable, and the highest of them much too low. The mistake I made, as I found later, was in trying to fuse too large a quantity, viz.. portions of ·5 A.T. The method I evolved and described in my paper had in view two important facts:—

Firstly.—That the carbon of even finely pulverised plumbago is very difficult and tedious to oxidise. Any chemist who has made a proximate analysis of carbon and ash in a sample of plumbago will, I think, agree that you have to roast and stir several times before you obtain a clean ash of uniform colour throughout. Indeed it is on this property of plumbago that its use for stoves and crucibles depends.

Secondly.—Carbon and gold under certain conditions have a great attraction for each other. It has, of course, been known for several years that charcoal will precipitate gold from aurocyanide solutions, and it has been used in Western Australia as a precipitant. (See Francis L. Bosqui "The Cyanide Process," 2nd ed. 1901, p. 160). It was used also for several months in certain extractor boxes on the City and Suburban mine. The assay of such gold-bearing charcoal from extractor boxes I have found to be easy compared to the assay of these old plumbago pots. I am inclined to think that the gold and carbon may exist in some sort of chemical combination which has never yet to my knowledge been investigated so fully as it deserves. I had written the last paragraph before Mr. Wilmoth's remark arrived, and therefore am specially interested in the fact he mentions that for years he found samples of plumbago crucibles after scraping to vary only between 9·2 and 9·5 oz. No doubt some chemist in the future will experiment in this direction and tell us exactly in what chemical condition this gold exists. In addition to the two considerations mentioned I found the method I recommended to give accurate results. I freely admit that I put forward this method in quite a tentative spirit, and if so simple a process as direct fusion can be made to yield accurate results no one will be better pleased than myself.

To Mr. Crosse I must sincerely apologise for having used and criticised his older method of September, 1897, whilst I was unacquainted with the fact that in January, 1898, he withdrew that method in favour of scorification, as described on p. 207 of Vol. II. I am pleased to say I have tried the sample referred to in my paper by this scorification method which gave a good result on using 2 gm. charcoal as reducer, viz.:

From the lead button ...	127 oz. 5 dwt.	per ton
From the slag ...	0 ,, 17	,,
Total	128 ,, 2 dwt.	fine gold

Methods of Assaying.—As a result of this discussion the assayer has now the choice of eight different methods, or modifications of methods. I have tried all of these, working on an old sample of plumbago pots, which had not been used in a Tavener furnace, and which was pulverised to -100 mesh. I give a tabulated statement of results obtained. It was my intention to have tried all the methods in triplicate on two different samples; pressure of other work has prevented my doing this so fully as I wished. The results I now give are mostly the mean of two trials. In several cases one or more preliminary trials was made and the proportions of flux or reducer altered.

Our daily routine flux is made up, as follows, *by weight*:—

Litharge, pure as possible ...	58 lb.
Soda carbonate, exsiccated and granulated	32 ,,
Fused borax	10 ,,
	100 lb.

Mr. R. Dures (*loc. cit.*) made no mention of reducer. I tried the method in duplicate in a modern "Rival" furnace and the lead buttons obtained only weighed 1·5 gm. and 2·7 gm. However, on addition of 5 gm. mealie meal I obtained buttons weighing about 35 gm. each.

Of the three methods dependent on oxidation by nitre, I found that described by Mr. D. Nicholas to work best and give the most satisfactory results.

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* Proceedings of this Society, Vol. II., p. 577.

SAMPLE OF 11 TONS 323 LBS. OLD PLUMBAGO POTS, FROM CYANIDE DEPT., AS SOLD TO SMELTING WORKS.

Sample for Comparative Assays, pulverised to -100 mesh. The Buyer's Assayer made the value: 15 oz. 3 dwt. of Fine Gold per ton.

Method of Assay.	DIRECT FUSION.			FUSION WITH NITRE.			Scorification.	Roasting with MnO_2 and Fusion in crucible.
	With Litharge PbO.	With Red Lead Pb_3O_4.	With Dioxide PbO_2.					
Suggested by	Mr. A. M. Thomas.	Messrs. Whitby and Wilmoth.	Mr. Whitby.	Dr. J. Garny and Mr. R. Dures.	Mr. A. M. Thomas.	Mr. D. Nicholas.	Dr. J. Loevy and Mr. A. F. Crosse	Mr. A. F. Crosse modified by Mr. J. Watson.
Reference	*J. C. M. & M. Soc. S. A.* Vol. XI. p. 572.	Vol. XI. p. 464, and Vol. XII.	Vol. XI. p. 464.	Vol. II. p. 577.	Vol. XI. p. 572.	Vol. XII. p. 13.	Vol. II. p. 207.	Vol. XI. p. 463.
Weight of Lead Button.	35 gm.	33 gm.	22 gm.	about 35 gm.	about 30 gm.	about 45 gm.	Fair size not weighed.	31 gm.
Value from Button	15 oz. 0 dwt.	14 oz. 0 dwt.	14 oz. 17·5 dwt.	14 oz. 5 dwt.	13 oz 15 dwt.	14 oz. 18 dwt.	14 oz. 6 dwt.	15 oz. 15 dwt.
Value from Slag	0 oz. 3 dwt.	1 oz. 10 dwt.	trace.	0 oz. 15 dwt.	0 oz. 7 dwt.	0 oz. 5 dwt.	nil.	trace
Total Value Fine Gold per ton	15 oz. 3 dwt.	15 oz. 10 dwt.	14 oz. 17·5 dwt.	15 oz. 0 dwt.	14 oz. 2 dwt.	15 oz. 3 dwt.	14 oz. 6 dwt.	15 oz. 15 dwt.

Criticism.—One of our Members of Council indulges in would-be destructive criticism and lets himself go. I asked for free and open discussion; so I suppose I must not object to it when I get it. At the same time I have good reason to believe that this fault-finding style of criticism keeps away quite a lot of papers which the younger members might bring forward, and I know for a fact that we have lost at least one member from this Society who took offence at his treatment by certain members of the council and editorial committee. What we want is constructive more than destructive criticism.

I would like to quote the definition of criticism which was put forward by Matthew Arnold, who I need not say was one of the greatest critics of the nineteenth century:—"To know the best which has been thought and said in the world, and through this knowledge to turn a stream of fresh and free thought upon our stock notions and habits and thus create a current of true and fresh ideas. The business of criticism is to do this with inflexible honesty and due ability."*

I will now answer *seriatim* the points raised by Mr. Whitby. I have no justification (he says) for assuming that the assayer of 1911 rests satisfied with Mr. Crosse's method of 1897. I have already offered an apology to Mr. Crosse for having whipped a dead horse, as it were. Still, judging from the absence of papers and abstracts in our *Journal*, there has been nothing fresh, under this heading, nothing worthy of note, since the pre-war days. I have hunted through all the indices and I think I am right in saying the only abstract is one telling us how Salamander crucibles should be annealed before using. I may add that at least one Past-President of our Society has congratulated me on bringing up this matter for fresh discussion. Mr. Whitby's next point No. 1 is that I have only dealt with one sample. That is quite true. Owing to the fact that I wished to make a lot of comparisons of assays of this one sample I limited the premise to avoid confusion. In my tabulated statement this evening I have referred very fully to one other sample.

No. 2 point "incorrectly labelled," I disposed of at the time Mr. Whitby raised it.

No. 3.—I made no mention of metallics. When the sample was handed to me by our cyanide manager, Mr. Higham, I asked if there were no metallics. I was told there were none, the material being all of a very brittle nature had been included in the general sample.

Point No. 4 is that the sample only passed the 60 linear sieve. Before stating that the sample had been pulverised to − 60 mesh, I had put it through a mesh of that size. After Mr. Whitby raised this question of mesh, I tried the sample with 80 and 100 mesh and found that 6·0% by weight was + 80, 30% was + 100 and the balance 64% was − 100. The 36% by weight I then had pulverised to − 100 and kept a sharp lookout for metallics, with the help of a powerful magnifying glass. Later, in his contributed postscript Mr. Whitby says :—"No such material could be obtained through a 90 or 120 mesh without yielding metallics."

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* "The Principles of Criticism," W. Basil Worsfold, M.A. 1902, p. 137.

When an assayer or metallurgist refers to metallics, he means those particles of malleable metal which go flat on percussion or concussion and will not pass through the interstices of a sieve. This sample was *entirely free from such* and when you know the history of the sample you need not be surprised. The plumbago pots were from the cyanide department entirely, there was not a single pot which had been used for mill gold. These old pots had been ground up, as I described, and used as a lining under the bed of the Tavener furnace. When an old bed is taken out, the brickwork and lining are broken up into small pieces and carefully hand picked. Mr. Higham can treat the lead bullion in his Tavener furnaces, and he and his assistants take good care that very little lead bullion goes away in the broken pieces. Any finely divided lead is most likely to be in a brittle condition.

During the past 8½ years I have smelted, refined and poured into moulds, something over 350 bars of cyanide gold bullion. Out of that number, I do not remember one single bar, whose nature could be termed any other than brittle. Yet Mr. Whitby says :—"No such material could be obtained through a 90 mesh sieve without yielding metallics."

Point No. 5.—"That the analysis published of Salamander crucible is beside the question." As the intelligent assayer has to vary his flux and his method of working according to the composition of the gold-bearing material, I think he would consider such an analysis helpful and distinctly pertinent to the question. To the assayer who works by rule of thumb, no doubt the analysis would be labour in vain.

Point No. 6.—If direct fusion can be found to give a reliable result, I quite agree with Mr. Whitby on this point, that we should use and are likely to use the simplest method.

By way of testing Mr. Whitby's contention that no reducer was required, I fused a 0·2 A.T. portion of the sample with 2 A.T. by measure of routine flux without any reducer. The lead button obtained weighed barely 9 gm. and on cupellation and parting gave a somewhat low result, viz., 124 oz. 1 dwt. as against 128 oz. − 8 dwt. average by other methods.

I have disposed of the numbered points ; but Mr. Whitby further objects that I use so much flux to melt down 0·1 or 0·2 A.T. charges. My object was not merely to melt down the charge ; but to accurately determine its gold contents. The enormous excess of flux as he calls it, was used to keep the gold out of the slag ; and, after all, the amount of slag obtained is no more than one gets from a daily portion of 3 A.T. of a slime residue. Working on a 0·10 A.T. portion, of this rich sample, one still has about 16 times more gold present than is customary in a daily screen determination. If he had seen, as I saw, the slag obtained by the two methods I described in my paper, he might comprehend the reason why the fluxes were increased, and he would not have been so hasty to accuse me of "bad fluxing," as he does in his contributed post-script.

In the contributed post-script, to which I have just referred, Mr. Whitby says : "His error arises from inefficient crushing, imperfect mixing and bad fluxing."

My reply is, I had nothing to do with the crushing or mixing of the sample, which operations were carried out at the Central Co-operative Smelting Works, on the Robinson mine, in accordance with their usual custom and in presence of the City and Suburban Co.'s representative. The sample was mixed and divided into three equal parts, the assays of these portions when carried out by competent assayers, using modern methods, gave :—

	I.	II.	III.
	oz.	oz.	oz.
Fine gold per ton ...	127·20	128·04	130·00

I am quite willing to leave it to your judgment to say whether these figures prove good or bad mixing of material which, as Mr. Whitby truly says, is of such complex nature.

Had we got such assays in the first instance, our management would have been perfectly satisfied.

Unfortunately, the No. III. portion, sent at first to Umpire No. 1 (through sickness of the head of this business and absence in a distant part of the chief assistant) fell into the hands of a junior who "tried his prentice hand" and made it 85·85 oz. only.

In conclusion, I trust the paper, discussion and reply will prove useful to assayers. Personally, I feel that, thanks to the help of this active and virile Society, I now know a lot more on the subject that I did last March when I wrote my paper.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

CONTAMINATION OF LABORATORY SAMPLES BY IRON DERIVED FROM CRUSHING MACHINERY.—"Very few chemists realize to what extent iron is introduced into pulp by the abrasion of the faces of the crushing machinery. Even experienced mining men whom one would expect to be conversant with the fact that stamp batteries are heavy consumers of iron, frequently overlook the fact that iron is a necessary constituent of battery pulp, and when in panning down a sample they come across a heavy black substance in the concentrate, they at once jump to the conclusion that they have tungsten, tellurium, iridium, or some other rare metal present and at once call for an analysis to classify it. It might be said to be an almost daily experience in a large assay office to be called upon to make a determination of an unknown for some one who imagines he has discovered a rare compound in his ore, when in reality he has merely concentrated out some of the iron derived from his crushing machinery.

Carbon is a constituent of iron, and in crushing rock, the carbon is to a certain extent liberated and its presence is frequently made known by the formation of a scum on the surface of water in panning. It is this carbonaceous scum that is frequently mistaken for silver sulphide or some other dark coloured mineral notorious for its sliming propensities.

It is the purpose of this paper to draw attention to the fact that the amount of iron introduced into laboratory samples by fine grinding has an important bearing on the accuracy of the work, whether it be of an analytical nature or the determination of a method for treating the ore.

The extent to which abrasion takes place is determined principally by the character of the material crushed and the hardness of the grinding faces. Soft, talc-like material, as one would expect, causes very little wear on the iron, while tough, flinty quartz produces an abnormal loss. From carefully conducted experiments I have found that in reducing an ore to 80 mesh in a disc grinder, from 0·25 to 3% of iron is added to the sample. In stamp mills the consumption of iron averages from 1 to 2 lb. per ton of ore crushed. In regrinding in pans the wear amounts to from 3 to 8 lb. per ton, clearly showing the heavy increase in the consumption of iron when attempting to produce a fine product.

The introduction of iron into a sample is frequently productive of serious results. In smelter work, where the practice is to reduce the sample to 120 mesh or finer, I have frequently seen charges which were made up according to analysis of such samples give great trouble, and no one apparently seemed to be able to assign a reason for the failure of the charge to work. I recall a case where a chemist in charge of a smelter, weighed his charges out to the nearest pound, and had all of the laboratory pulps put through an excessively fine screen. Until his attention was called to the matter, he failed to understand the reason why his furnaces gave so much trouble. In smelter work, the error, it will be noticed, is two-fold; it increases the apparent contents in iron, and decreases the seeming amount of silica to be fluxed.

As a rule dry ores received by ore purchasers possess the greatest value and consequently bear the highest treatment charges. It has been noticed that these ores are the most abrasive ones, a point which possesses considerable importance when it is remembered that in order to get it through a 120 mesh sieve it is necessary to *salt* it with 2% iron. This fact is well known to smelters and when the seller insists on having the sample representing his shipment put through an extra fine screen he never meets with opposition. The profit which the smelter derives from samples innocently salted by fine grinding must be considerable in the aggregate.

In preparing large samples of commercial products, such as limestone, magnesite, clays and glass sands, for analysis, the difficulty of preventing contamination with iron, and the serious results which may follow should the iron find its way in, may be readily appreciated. The introduction of a very small fractional percentage of iron into a clay or glass sand analysis may result in the rejection of what might be actually an excellent grade of material. In carbonic-acid determinations, the effect of iron would be to generate hydrogen during the acid treatment, which would of course lead to erroneous conclusions.

In conclusion I would like to say that the introduction of iron into laboratory samples by the abrasion of grinding faces is a matter deserving of serious consideration on the part of chemists. The trouble might be overcome by substituting silex grinders for iron ones, but it is quite likely that an error of equal if not greater magnitude would be caused by their use because of a greater tendency to disintegrate. There is no data available on the subject, and it might be well to look into it. Aside from silex grinders, there seems to be little choice of any other material outside of iron or one of its alloys. It is quite possible that by using manganese-steel faces, the abrasion might be reduced, and it would be well to investigate the use of this material."—G. A. JAMES.—*Transactions American Metallurgical Society*, May, 1911, p. 5. (H. A. W.)

APPLICATION OF THE "GLOW REACTION" TO THE QUALITATIVE DETECTION OF THE PLATINUM METALS.—"For a long time it has been known that the platinum metals present the rather remarkable phenomenon of condensing relatively large volumes of gases upon their surfaces at room temperatures, and also, that this property is especially pronounced if the metals are in that finely divided state obtained by the reduction of their salts. From the early investigations of Dobereiner to the present day, this occlusion of gases by platinum has been the subject of much research, and among other things, it has been shown that hydrogen in the occluded state acts in the same manner as nascent hydrogen and that, in general, the presence of metallic platinum in the finely divided state greatly accelerates the oxidation of many oxidisable gases. It was thought that this property might be made use of and to that end an investigation was made to determine the reliability for qualitative purposes of the 'glow reaction' obtained by passing a mixture of air and illuminating gas over very finely divided platinum.

The substance to be tested is brought into solution by any of the common methods and about 0·2 cc. of this solution is absorbed in a piece of thin asbestos paper by alternately dipping the paper into the solution and heating until the required volume has been absorbed. Experience showed the above volume (0·2 cc.) to give the best results in these tests. The moist paper held by one end in a pair of tongs is now heated to redness in the Bunsen flame, then removed; after redness has ceased, but while the paper is still hot, it is brought into a stream of a mixture of illuminating gas and air from a Bunsen burner. If platinum is present, the asbestos paper will begin to glow. The glow must last for some time and can be brought back after it has once died out by again heating the paper and holding it in the stream of gas. The burner must be arranged to supply a fairly good mixture of illuminating gas and air and the pressure must not be too great.

It is also essential for the greatest sensitiveness of the test that the asbestos paper be very thin. In these experiments pieces of asbestos paper 0·5 × 3·0 cm. and 0·3 mm. thick were employed, only approximately half being used to absorb the liquid. The intensity of the glow can be somewhat increased by having the gas warm. This can be easily accomplished by slightly warming the upright tube of the burner before passing the gas through it. All the results given below, however, have been obtained with the gas at room temperatures.

Summary.—1. The oxidation of the constituents of illuminating gas, particularly of hydrogen, with finely divided platinum, palladium, rhodium, or iridium as catalytic agent is accompanied by a glow which is distinctly characteristic of these metals. Hence this glow may be used as a qualitative test for these metals.

2. This test is sensitive to extremely small quantities of the metals, the limits being: platinum, 0·002 mg.; palladium, 0·0005 mg.; rhodium, 0·0009 mg. and iridium, 0·005 mg.

3. The presence of other substances does not for practical purposes interfere with this test. In fact, the test may be applied directly to a solution without preliminary treatment for the separation of other substances.

4. In an analysis where all the platinum metals are to be considered, this test may serve to show the presence of the platinum group with the exception of osmium and ruthenium, and also to confirm the presence of platinum, palladium, rhodium and iridium, when the metals have been separated. In most practical work, the 'glow reaction' becomes distinctly a test for platinum, the most common metal of the group. For this latter purpose the glow test recommends itself strongly, because of its sensitiveness and the rapidity with which it gives results."—L. J. Curtman and P. Rothberg, *Journal American Chemical Society*, May, 1911.—*Transactions American Metallurgical Society*, May, 1911, p. 15. (H. A. W.)

Titrating for Protective Alkalinity.—"The authors, under the heading 'Errors in Testing Cyanide Solutions,' disagreed with Bede Collingridge as to the effect of potassium iodide in titrating cyanide solutions for protective alkalinity, and experimented themselves. They found that the presence of potassium iodide obscured the end point. In presence of silver cyanide and potassium iodide, phenol phthalein fails to give a sharp end point; the indistinctness is a double one, due, principally, to the constant return of the phenol phthalein color indicating alkalinity after its discharge by standard acid to an apparent neutral point, and, to a minor extent (owing to the employment of unnecessary amounts of potassium iodide and silver solution), due to the masking of faint alkaline colorations by the iodide of silver suspension.

When using a solution one-tenth normal, the end point of the reaction is sufficiently sharp to give accurate results, but to determine small amounts of protective alkali in small quantities of highly dilute cyanide solutions (such as may be derived from the treatment of slime by decantation) standard acid should be employed of a dilution commensurate with requirements, say ·04 normal. Under such conditions the error becomes noticeable, and is further accentuated by the use of silver nitrate in excess in the first (or cyanide) titration. If, however, no potassium iodide be used, the results with the weak acid are accurate and the reactions normal.

The phenomena are explained as follows: 'In the titration for cyanide with silver, all potassium cyanide (which would otherwise give its full alkali value on titration with acid) must first be converted into the double cyanide KCN, AgCN, a compound which is not decomposed by dilute acid in which the combined moiety of KCN does not therefore show alkalinity on titration with standard acid and an indicator.

The end point of the cyanide reaction is, however, necessarily marked by the precipitation of silver cyanide as a cloudy suspension. This precipitated silver cyanide now reacts with potassium iodide, forming yellow silver iodide, and free potassium cyanide, of course with its equivalent alkalinity.'

The authors conclude by re-stating Mr. Collingridge's proposition thus: 'If small amounts of protective alkali are required to be measured by dilute standard acid solutions, the use of potassium iodide as an indicator for the cyanide reaction point will give rise to serious error in regard to protective alkalinity present.

Further, the error will be greater in proportion to the overdoing of the cyanide end reaction with silver-nitrate solution, in presence of an equivalent addition of potassium iodide. This is due to the reaction between argentic cyanide and potassium iodide in producing free potassium cyanide.'"—H. L. Sulman and Frederick Reade. *Bulletin No. 67, Inst. Min. and Met.*—*The Pacific Miner*, March, 1911, p. 103. (A. R.)

Rapid Estimation of Sulphur in Coal Gas or Ammonium Sulphate.—"Use is made of Malfatti's reaction, namely, the formation of hexamethylene tetramine and liberation of acid by adding formaldehyde to the solution of an ammonium compound, thus: $4NH_4Cl + 6CH_2O = N_4(CH_2) + 6H_2O + 4HCl$ The liberated acid is titrated with standard alkali and phenolphthalein. For the determination of sulphur in coal gas the liquor obtained in the referee's test is made up to 500 ccs., and 50 ccs. are boiled for a few minutes to expel CO_2. To the hot solution 10 cc. of formaldehyde solution (about 30%) are added and the titration is at once made with decinormal potash, 1 cc. of which is equivalent to 0·0247 gr. of sulphur. The potash solution is best standardised by means of a standard solution of ammonium sulphate containing a little ammonium carbonate. The results are consistent but a little higher than those obtained by the $BaSO_4$ method. For estimating $(NH_4)_2SO_4$ the salt is dissolved in water, neutralised with KOH, boiled, and then formaldehyde is added and then boiled again for a few seconds and titrated. The results agree closely with those obtained by the $BaSO_4$ method."—H. Blair.—*Metallurgical and Chemical Engineering*, June, 1911, p. 321. (J. A. W.)

Modification of the Diphenylamine Test for Nitrates.—"After pointing out that the manner of carrying out the diphenylamine test is of considerable importance, the authors recommend the following mode of procedure, in which some of the features of modifications proposed previously are combined. The reagent is prepared by dissolving 700 mgm. of diphenylamine in a mixture of 60 cc. of concentrated sulphuric acid and 28·8 cc. of distilled water, cooling the solution, and then adding slowly 11·3 cc. of hydrochloric acid of sp. gr. 1·19. After standing overnight some of the diphenylamine should separate, showing that the solution is saturated. In making a test, 1 cc. of the liquid under examination is mixed thoroughly with 1 drop of the diphenylamine reagent, and then 2 cc of concentrated sulphuric acid are added from a pipette, so that two layers are formed. The whole is agitated gently so as to cause a slight mixing of the liquids at the plane of contact, and then kept at 40° C. for 15 or 20 minutes. It is stated that in this way 1 part of nitrate nitrogen in 25 millions or 1 part of nitrate nitrogen in 35 millions can be detected, whilst, by heating at 40° C. for 1 hour, the delicacy is increased to 1 part in 32 millions or 1 part in 44 millions respectively."—W. A. Withers and B. J. Ray, *Journal Amer. Chem. Soc.*, 1911, *33*, 708-711.—*Journal of the Society of Chemical Industry*. June 15, p. 708. (A. W.)

The Exact Determination of Sulphates.—"In a previous publication it was shown that barium sulphate precipitated from acidified solutions of alkali sulphate and chloride is, in general, contaminated by a number of impurities, the amount of which may be so great as to cause errors in sulphate determinations of 1% or more. Thus precipitates made from acidified solutions of sodium sulphate and chloride contain, before ignition: (*a*) chloride, almost certainly as barium chloride; (*b*) sodium sulphate; (*c*) sodium hydrogen sulphate; the amounts of these depend upon (1) the composition of the original solution, (2) the method of precipitation, (3) the length of the interval between precipitation and filtration. On igniting the precipitate, the barium chloride reacts with the sodium hydrogen sulphate as follows: $BaCl_2 + 4NaHSO_4 = BaSO_4 + Na_2SO_4 + 2HCl$; the hydrochloric acid gas escapes: and along with it is lost some free sulphuric acid by decomposition of the sodium hydrogen sulphate, if, as is usually the case, the amount of the latter is more than equivalent to the barium chloride originally present in the precipitate The present researches were carried out to study further the phenomena thus presented, and as a result it has been found that the occlusion by barium sulphate of other sulphates is a general phenomenon. The amount of this occlusion depends upon (*a*) the composition of the original solution (*b*) the fineness of the precipitate, which in turn is conditioned by the degree of solubility of barium sulphate in the particular medium, the rate of precipitation, and the time and manner of standing between precipitation and filtration. The phenomenon is therefore in all probability an adsorption at the surface of the grains of the precipitate; since it is affected by the factors just mentioned. On the basis of the knowledge gained in this way, attempts were made to find a direct method for the determination of sulphate which should be generally applicable, exact, and require only small and easily determined corrections. The authors suggest the following procedure: To the solution (300 cc. for a precipitate to weigh 2 grams) add 50 cc. of concentrated hydrochloric acid, heat to boiling, and precipitate, stirring constantly, with a 10% solution of barium chloride, this should be added at such a rate that about four minutes is required in running in the 22 cc. necessary, the rate is best regulated by attaching a suitable capillary tip to the burette containing the barium chloride solution. Evaporate the whole to dryness on the steam-bath (this may be done immediately after precipitation), take up with hot water, filter Through paper, wash until the washings are free from chloride, ignite very carefully (so as to obviate reduction) and heat to constant weight over a Bunsen burner. The necessary correction is determined by a concurrent calibration of the method: that is, by dissolving an equivalent weighed amount of pure dry sodium (or potassium) sulphate in a medium such that the resulting solution is as nearly as may be of the same composition as the solution to be analysed; the sulphate in this comparison solution is then determined precisely as above. The difference between the calculated amount of barium sulphate and that actually found is the correction to be applied to the weight of the precipitate obtained in the actual analysis."—John Johnston and L. H. Adams.—*Journal American Chemical Society*, xxxiii., 6, 829, June, 1911. (J. A. W.)

Separation of Chromium, Iron and Aluminium by means of Ammonium Persulphate.—"For the separation of chromium, aluminium, and iron, treatment of the mixed hydroxides of the metals with sodium peroxide, which converts the chromium into chromate and the aluminium into aluminate, in presence of water, is now largely employed. The author has found that the sodium peroxide may be replaced by ammonium persulphate, which is easier to handle and cheaper than sodium peroxide. The hydroxides are mixed with water and warmed with a small quantity of the persulphate until solution is complete, the chromium hydroxide being converted into a compound of chromic anhydride. On the addition of alkali, ferric hydroxide is precipitated; aluminium and chromium can be detected in the solution in the usual way."—R. C. Cowley, *Chem. and Drug.*, April 29, 1911, 133.—*Journal of the Society of Chemical Industry*, May 31, p. 651. (A. W.)

Separation of Lime and Magnesia.—"The author finds that trustworthy results, in the separation of lime from magnesia, may be obtained if the slightly acid solution containing the calcium and magnesium salts be treated with ammonium chloride and oxalic acid and then rendered nearly neutral to methyl orange by the addition of either aniline, pyridine, or quinoline. The calcium oxalate is precipitated completely in the cold within 10 minutes. This process has the advantage over the ammonia-ammonium oxalate method that correct results may be obtained with concentrated solutions of magnesium salts containing relatively small quantities of calcium salts. Precipitation of the lime from acetic acid solution with the addition of mercuric chloride, and the sodium sulphate method, yielded unsatisfactory results."—E. Murmann, *Monatsh. Chem.*, 1911, *32*, 105-115.—*Journal of the Society of Chemical Industry*, April 29, p. 510. (A. W.)

Precipitation of Aluminium, Chromium and Iron.—"The hydroxides of these metals, as usually precipitated by ammonia are voluminous and difficult to wash. Other metals also are co-precipitated and the hydroxides may pass through the filter in the colloidal form. The method proposed is as follows. The solution containing 0·1-0·2 gm. of the metals is, if acid, neutralised with ammonia as far as this is possible without precipitation, and diluted to 250 cc. 20 cc. of a 6% solution of ammonium nitrate are added and the solution is heated to boiling until the smell of nitrogen oxides is not apparent. After heating on the water-bath for ¼—½ hour, the finely divided precipitate is washed several times with hot water by decantation, filtered through a filter of 11 cm. diameter, dried, ignited in a Rose's crucible and weighed. If, in the original solution, there is more ammonium salt than 1% in 250 cc., the precipitation is not complete after longer boiling. In this case ammonia must be added drop by drop, after boiling off oxides of nitrogen until their odour is just perceptible; the solution is then heated in the water-bath and the operation completed as above."—E. Schirm, *Pharm. Zentralh.*, 1910, [22.] *Collegium*, 1911, 99-100.—*Journal of the Society of Chemical Industry*, April 29, p. 510. (A. W.)

Method of Testing Galvanized Iron.—"According to Preece's method, the test-piece is placed in solution of copper sulphate under standard conditions and the number of one-minute immersions necessary to produce a bright, adherent film of copper on the piece is regarded as an accurate measure of the thickness of the zinc coating. The authors recommend the use of a slightly basic solution of lead acetate as affording more reliable results; the solution is prepared by dissolving 400 gm. of crystallised lead acetate in one litre of water, agitating the solution with 4 gm. of finely powdered litharge until most of the latter is dissolved, and diluting the clear, decanted solution to 1·275 sp. gr. at 15·5° C. The weighed test-piece is immersed in the solution for periods of not less than 3 minutes at a time (the adherent lead being removed after each immersion) until a bright surface of iron is obtained. The piece is then re-weighed to obtain the weight of the zinc coating removed, or the latter is calculated from the quantity of lead deposited; in the former case, if iron is found in the acetate solution, the amount is determined and allowed for."—W. A. Patrick and W. H. Walker, *J. Ind. Eng. Chem.*, 1911, *3*, 239—242. *Journal of the Society of Chemical Industry*, xxx. 9, p. 546, May 15, 1911. (A. W.)

Use of Ether in the Analysis of Gold Alloys.—"It has been found that the known solubility of gold chloride in ether may be utilised for the accurate analysis of gold alloys, especially coinage, in which the gold may be determined with a loss of not more than 0·01%. The solution of the alloy in dilute *aqua regia*, after filtering off any silver chloride, is repeatedly extracted with ether (the presence of the free acid being necessary), and the ethereal extract is dealt with as below; the aqueous layer may then be used for the detection or determination of other metals. For the extraction, the solution should contain 5—10% of metal and 5—13% of total hydrochloric acid; the shaking is repeated 4—5 times, a total volume of 50 cc. of ether being used for 20 cc. of solution containing 1 gm. of metal; 10 cc. of water are added to the ethereal extract, the ether is distilled off, and the residual solution is warmed for some minutes with sulphurous acid, after which the precipitated gold is filtered off, washed, ignited, and weighed. Gold may thus be separated from all other metals, and the method furnishes a simple means for the preparation of very pure gold, containing less than 0·001% of impurity. For very accurate determinations, the principal portion of the gold is removed from the solution by two extractions with ether; the aqueous layer is then concentrated, the chlorides are converted into sulphates, and the residual gold precipitated by sulphurous acid or hydrazine; the precipitate is then dissolved in a small volume of acid and the solution is extracted several times with ether. As an example a detailed analysis of gold coinage and suitable apparatus therefor are described, the detection or determination of small quantities of silver, lead, iron, arsenic, antimony, tellurium, nickel, platinum, palladium, and iridium being discussed, and analyses of the gold coinage of various countries as effected by this method, being given; all the coins examined contained, besides copper and silver (0·2—0·4%), very small percentages of lead, iron, platinum, and palladium, and a trace of arsenic. An analysis of certain brittle coins pointed to the presence of a slight excess of lead as the possible cause of brittleness. In the course of the investigation it was found that hydrochloric acid is able to dissolve gold in the presence of cupric chloride; the compound, $CuAu_2Cl_8,6H_2O$, was also prepared, in the form of very soluble lustrous olive-green prisms, by evaporating a solution containing cupric chloride (1 mol.) and chlorauric acid (2 mols.)—F. Mylius, *Z. Anorg. Chem.*, 1911, *70*, 203-231.—*Journal of the Society of Chemical Industry*, May 15, p. 547. (A. W.)

Determination of Lead in Alloys containing Antimony and Tin.—"The author's method in detail is as follows: Treat 0·5 gm. of the finely divided alloy, or such quantity as will contain about 0·3 gm. of lead in a 250 cc. beaker with a minimum of aqua regia, or 5 : 1 hydrochloric acid and potassium chlorate. After solution evaporate the contents of the beaker to a pasty consistency on a steam or water bath. Add 5 cc. of concentrated hydrochloric acid and warm, then add 15 or 20 cc. of water. A few crystals of tartaric acid may be added at this point. Now add 20% sodium hydroxide in slight excess, disregarding any precipitate which may form. Add 50 cc. of sodium sulphydrate, or more if required to produce an excess, and warm gently on steam or water bath with occasional stirring for half an hour. At this point the separated sulphides should be perfectly flocculent and should subside readily. Decant through an 11 cm. C. S. and S. filter No. 597, and wash by decantation with hot water containing 10 cc. of the sodium sulphydrate per litre. The precipitate is finally entirely transferred to the filter, and washed until 15 cc. of the filtrate give a separation of milky sulphur containing no coloured sulphides when acidified with acetic acid. Now transfer the filter and precipitate to a 500 cc. Jena Erlenmeyer flask. Add 25 cc. of 2 : 3 nitric acid and boil, best over the free flame, manipulating the flask in a holder, until the filter is entirely disintegrated and no black particles remain. Add 15 or 20 cc. of concentrated sulphuric acid, and continue the boiling until white fumes of sulphuric anhydride appear, and the filter paper is thoroughly carbonised. Care must be taken here to continue the boiling until no solid carbonaceous matter remains, and the sulphuric acid has acquired a black or caramel colour, a procedure which requires only four or five minutes at the most. Remove the flask from the flame, allow to cool slightly, and cautiously add 0·1 or 0·2 gm. of pulverised potassium permanganate, and heat again. Repeat these operations until the brown colour of the sulphuric acid is entirely destroyed, and the acid is coloured slightly pink or green. Allow to cool, and add 50 cc. of water, bring to a boil, and drop in sulphurous acid or a 10% solution of sodium bisulphite while boiling, until the colour due to the excess of permanganate is completely discharged. Allow the flask to stand hot for 15 minutes, then cool to room temperature with the addition of 25 cc. of alcohol and filter. Wash with cold 10% sulphuric acid, concluding with one wash of pure cold water. It is by no means necessary nor desirable to transfer all of the lead sulphate to the filter. The C. S. and S. No. 551 black filter is admirably adapted to use here, for with the white filters it is often difficult to be certain that all of the lead sulphate has been removed in the subsequent operations. Place the flask containing the remaining lead sulphate under the funnel, open the filter carefully, laying it against the inside of the funnel, so that its whole interior is exposed, and wash its contents into the flask with a boiling solution of sodium acetate contained in a wash bottle. The greatest care must be taken that every trace of lead sulphate is removed. Now add sodium acetate solution to the contents of the flask, and warm until the whole of the lead sulphate has dissolved and the solution is perfectly bright and clear. In case there is a persistent turbidity, an incomplete sulphide separation is indicated. Add 20% sodium hydroxide drop by drop to the solution, until a cloud forms, which just does not redissolve If the percentage of lead is low, it is advisable to add a few drops of phenolphthalein and add the alkali until a pink colouration is produced. Now add acetic acid drop by drop until the solution just clears or the colour of the phenolphthalein is discharged. Add 10 cc. of a 10% solution of potassium dichromate (the powdered commercial article), heat the mixture to boiling, and boil gently for one or two minutes or until the precipitate becomes of a deep orange colour and settles fairly well on removing from the heat. Longer boiling does no harm, but is unnecessary. Filter through an 11 cm. filter and wash filter and precipitate ten times with a hot dilute solution of sodium acetate (50 cc. of a cold saturated solution of the commercial salt diluted to 1 litre), using perhaps 5—6 cc. each time. Now again place the clean flask under the funnel, remove and open the filter as before, and wash the precipitate back into the flask with a jet of cold dilute hydrochloric acid (1 : 2). Continue the washing until every trace of residual lead chloride is dissolved, and no trace of colour remains on the filter. Dilute the filtrate with cold water until the flask is nearly two-thirds full, and add 2 cc. of a 50% solution of potassium iodide. Mix gently and titrate without delay (to avoid possible loss of iodine by volatilisation) with standard sodium thiosulphate solution until the brown colour becomes faint; then add sufficient starch liquor to produce a strong blue colour, and finish the titration very slowly, finally a drop at a time, allowing each drop a little time to produce its full effect, until the solution becomes a clear pale green with no tinge of blue. The end point is sharp, but care must be exercised or it may easily be passed by a drop or two.

Standardise the thiosulphate solution on pure lead foil. Dissolve 0·3 gm. of lead foil in a little 2 : 3 nitric acid contained in a 200 cc. flask. Boil the solution, add 5 cc. of concentrated sulphuric acid evaporate to fumes of sulphuric anhydride, cool and add 50 cc. of water. Boil for a moment and cool again with the addition of 25 cc. of alcohol. After filtration treat the washed lead sulphate as described above.

A close approximation to its lead value may be obtained by multiplying the normality of the thiosulphate by 0·06799.

'Bismuth in small amounts does not interfere with the above method, since it remains in solution as sulphate when the lead sulphate is filtered off. When, however, several per cent. of bismuth are present, an indefinite amount is liable to remain with the lead as basic sulphate and to raise the result. The bismuth may be detected during the neutralisation of the sodium acetate solution by the failure of the white cloud to redissolve in excess of acetic acid. In such a case add a few drops of phenolphthalein solution, make again slightly alkaline with the sodium hydroxide, then faintly acid with acetic acid, add the dichromate, and boil for a minute or so as usual. Now remove from the heat and add about 2 gm. of citric acid dissolved in a little water and mix by agitating. Any bismuth chromate will go into solution at once. Filter without delay, lest the composition of the lead chromate change somewhat by standing in the hot acid mixture, and finish as usual.' Solutions required :—

Sodium sulphydrate : 500 gm. sodium sulphide are dissolved in 1 litre of water. Portions of this solution are saturated with hydrogen sulphide, and filtered as required for use.

Sodium thiosulphate : The usual $N/10$ solution.

Sodium acetate : 90 gm. lead-free sodium acetate and 8 cc. of 99.5% acetic acid per litre."—*Journal of the Society of Chemical Industry*, May 15, p. 519. (A.W.)

METALLURGY.

THE EFFECT OF THE VARYING COARSENESS OF GOLD UPON THE ACCURACY OF THE ASSAY OF GOLD ORE.—"The object of the experiments and examinations recorded in the following pages is given sufficiently in the title.

For the purpose of the assays approximately 50 assay tons each of silica were mixed with 250 milligrams of each lot of gold filings. The first lot of the 40-mesh ore was mixed for about 30 minutes and four assays of 1 A.T. each were made. The assays were repeated in lots of 6 after remixing following each assay in the hope of attaining more uniform results, and to get an idea of the time necessary to be spent in mixing. After the fourth lot was assayed, it was concluded that the maximum amount of time necessary had been exceeded.

In general, the results indicate :

(1) If gold ore is classified by a set of fine screens, the average weight of the gold particles, free, will be about one-third the weight of the largest gold cube which should theoretically pass through the aperture of the screen.

(2) Wire-cloth screens in the sizes examined contain many apertures whose smaller dimension is at least 1¼ times the length of the side of the theoretical square aperture and the ratio of the area of the largest measured aperture to the square is from 1·3 to 2·9 with selected cloth.

(3) Beyond a certain length of time, further mixing will not reduce the number of gold particles in excess or deficit likely to enter the assay sample, and below 120-mesh with at least No. 43 wire of the Old English or London standard, the size and number of these particles are likely to cause an error of $5 per ton in the 100, 90 and 80-mesh sizes, with chances of their increasing to double this sum, in $100 ore. In the sizes below the error would rapidly increase until with 40-mesh screenings we should expect an error of $18 per ton, with the possibility of $30 error.

(4) The average percentage error is about 17% with a 40-mesh screen, decreasing in the sizes above to a minimum of 1·3% with the 150-mesh material, the maximum percentage error between the limits mentioned being about 30% and 1·7%, respectively."—T. D. TOBIN.—*Pacific Miner*, May, 1911, p. 156. (H. A. W.)

CRITICAL MOISTURE IN TUBE MILL FEED.—"In Mr. John W. Bell's letter of February 28, published in *The Mining Magazine* of April, 1911, it is stated that 'the percentage of water which should be present in the tube mill feed is numerically equal to the percentage of voids in the sand inside the mill.'

Apparently Mr. Bell is confusing percentage of water in pulp by weight with percentage of voids by volume. Actual experience shows that the coarse grit fed to tube mills on the Rand occupies about 20 cub. ft. to the ton, when settled under water. Taking the specific gravity of the dry material in such a product at 2·67, a ton occupies 12 cub. ft.: hence the voids in such a product settled under water occupy 20 cub. ft. less 12, equal to 8 cub. ft. or 40% of the total volume. When however such voids (8 cub. ft.) are filled with water, this water only weighs 0·25 ton, equivalent to 20% of the total weight of sand (1 ton) and water (0·25 ton), or a total weight of 1·25 tons of pulp.

If the pulp fed into the tube mill contains 40% by weight of moisture, such pulp, on the solid being allowed to settle, would have more than half the water standing above the settled solid. Hence assuming that 40% of moisture is desirable in the pulp fed to a tube mill, this is greatly in excess of the percentage by weight which is required to fill the voids in the settled sand, and it is not apparent that there is anything more than an accidental coincidence between the desirable proportion by weight of water in a tube mill feed and the proportion by volume of voids in the settled sand.

It may be added that in practice the desirable proportion of moisture in tube mill feed is largely dependent upon the tonnage of sand treated, the general rule being that the smaller the tonnage the lower is the percentage of moisture both permissible and desirable. The apparent reason for this is that with a large tonnage of thick tube mill feed (say, 400 tons solid per day to a mill 22 ft. long by 5½ ft. diameter inside shell) the mill cannot discharge at an adequate rate and consequently becomes over full, so that large masses of sand and pebbles are projected at once instead of the pebbles individually, as would be the case with a more dilute pulp. With a small thick feed (up to 200 tons of solid per day as a pulp containing 27% by weight of moisture) over-crowding of the tube mill does not take place and more efficient crushing is obtained than with a similar small feed of solid as a more diluted pulp "—W. R. DOWLING. —*The Mining Magazine*, June, 1911, p. 439. (A. R.)

STANDARDIZATION OF CONVENTIONAL SIGNS FOR FLOW-SHEETS.—"The complex character of most of our modern metallurgical processes has made it very difficult for a writer to give a written description of a system of treatment in a way which will not tend to confuse the reader. For this reason it has been necessary to introduce the flow-sheet into technical writing. The flow-sheet is a sketch containing a number of conventional signs representing various machines, connected together by lines which indicate the character of the material and the direction in which it is moving.

The advantage of the flow-sheet over a written description is that it permits the reader to grasp the whole scheme of treatment at a glance. If one understands the meaning of the conventional signs the interpreting of the flow-sheet is a simple matter. If he does not he must have recourse to the text as in the case of the conventional signs shown in connection with this article. To a certain extent the value of the flow-sheet is lost when the reader is compelled to take time to decipher it.

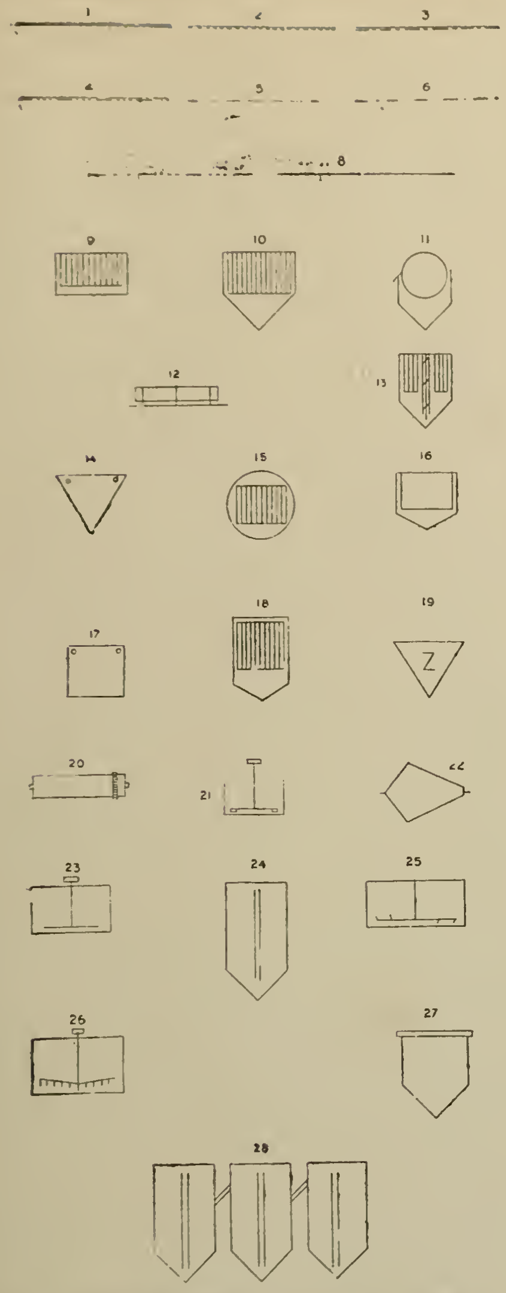
The purpose of this article is to point out the great advantage which would accrue to the profession if engineers would agree upon standard conventional signs for representing machines and flow-lines which would tell the character of the material without having to study out their meaning. At the present time every engineer has his own particular ideas about laying out flow-sheets so that every time the reader comes across one in a technical journal he feels as if he must learn a new sign language. There is no logical reason why the conventional signs used in flow-sheets should not be standardized. It is merely a question of some technical body, such as the American Metallurgical Society, taking the matter up and after due consideration, giving its official approval to certain signs. I believe some such action of this kind is very necessary and that it will meet with the hearty approval of the members of the profession, who will gladly co-operate with us by introducing them into their work.

In order to present the subject in a way which will permit of discussion, I have gathered together a number of conventional signs which, with more or less modifications, have been in use for some time.

As a general rule, engineers employ the ordinary straight unbroken line to represent flow-lines which makes it necessary to accompany them with a description of the material which it represents. This tends to make the preparation of a flow-sheet a laborious task and furthermore it fills the drawing up with a lot of matter that tends to confuse the reader. This has led to the introduction of the broken line to represent a certain class of material, which is a step in the right direction. The next move should be to standardize these lines.

In the accompanying drawings, Figs. 1 to 8 inclusive, are flow-lines, each of which indicate a certain kind of pulp or solution. It will be noticed that the first four flow-lines are double, while the remaining four are single. The double lines are intended to represent pulps and the single ones liquids. The advantages of employing a system of lines which will enable one to determine at a glance whether he is following a flow of pulp or solution is obvious. The objection to the double line is that it is clumsy, but I think the points in its favour outweigh this. A series of single broken lines could be used in place of the double lines but this would lead to some confusion because of the multiplicity of lines of this type.

It is not contended that the eight lines in the accompanying sketch cover all cases that one meets with in metallurgical practice. I have simply chosen a few of the common ones, leaving to others the task of making additions, and improving on those submitted. Fig. 1 is intended to represent ordinary unclassified battery pulp. It might be well to have a line which will represent an unclassified pulp from which the free gold has been removed by amalgamation and also one to represent a pulp from which the concentrate only has been taken. We might have still another to show an unclassified pulp from which both free gold and concentrate have been removed. The objection to having so many different lines is that it will tend to complicate flow-sheets which is, of course, the very thing we are trying to overcome. In a mill of the character of the Goldfield Consolidated if one were to attempt to introduce a flow-line to represent each change in the pulp there is no doubt but that a great multiplicity of lines would have to be employed with attendant confusion to the reader unless he had become proficient in our hieroglyphical language, but there are also numerous cases



Machines Represented by Conventional Signs.

Fig. 1—Wet unclassified pulp; Fig. 2—Wet sand—a product of classification; Fig. 3—Wet slime—a product of classification; Fig. 4—Ore—a product of dry crushing; Fig. 5—Strong cyanide solution; Fig 6—Weak cyanide solution; Fig. 7—Wash water; may or may not contain a little cyanide; Fig. 8—Water; Fig. 9—Moore vacuum filter; Fig. 10—Butters' vacuum filter; Fig. 11—Oliver Continuous filter; Fig. 12—Burt revolving filter; Fig. 13—Hendryx combined vacuum filter and agitator; Fig 14—Merrill's filter-press for slime treatment; Fig. 15—Kelly filter-press; Fig. 16—Sweetland filter-press; Fig. 17—Ordinary filter-press; Fig. 18—Blaisdell pressure filter; Fig. 19—Merrill filter-press used for zinc-dust; Fig. 20—Tube-mill; Fig. 21—Grinding pan; Fig. 22—Hardinge conical tube mill; Fig. 23—Mechanical agitator—paddle type; Fig. 24—Pachuca tank; Fig. 25—Trent agitator; Fig. 26—Dorr pulp thickener; Fig. 27—Settling tank; Fig. 28—Pachuca tanks in series.

where the treatment of the ore is quite simple and the use of a number of different flow-lines would tend to simplify the understanding of the process. It is for this reason that I suggest a discussion of the advisability of employing a number of lines in addition to those shown in Fig. 1 to 4.

It is very desirable to employ lines, which, while indicating the direction of flow, will also clearly explain the character of the solution used at each stage of the process. Fig. 5 to 7 inclusive represent cyanide solutions of various strengths and Fig. 8 represents pure water. It would be well if we could go a step further and introduce a sign into a flow-line which will indicate whether or not a solution has been precipitated. This could be easily done by introducing a new line but this is objectionable for the reason that it will bring about confusion in the flow-sheet. For example, it would never do to have a line like Fig. 5 be shown as entering the zinc-boxes and not leaving it. A strong solution starts from the storage tank always and the line should be maintained throughout the cycle until it enters it again to be standardized. It would be advisable to add some character to it as, for example, when leaving the zinc-boxes to show that it is barren. One will know of course that a solution has been freed of its precious metal after passing through zinc-boxes, but we have frequent cases where a rich solution is sometimes passed around the boxes and used a second time before being precipitated. My idea was to attach conventional signs to flow-lines representing solution which would indicate their composition. This is a subject which I shall leave open for discussion.

It would be a very simple matter to get up a conventional sign to represent a machine of standard make and yet I have experienced a great deal of trouble in doing this in a way satisfactory to myself. A sign for the purposes for which we want it must be simple and easily drawn; it must possess a characteristic individuality which will enable one not thoroughly posted on the subject to tell what it represents without having to read the article which it illustrates; and it must be a view taken from a position which will permit of the receiving and discharge ends being shown so that there can be no mistake regarding the character of the material represented by the flow-lines and the point from which the material started.

To illustrate what I mean by this, consider the case of a dewatering tank represented by a conventional sign which shows a plan of the tank. With a sign of this type, it would be very difficult to tell which flow-line is entering and which one is leaving the tank. If we represent it as shown in Fig. 27 by a side view, it is easy to see that a flow-line touching the centre of the tank at the top represents a thin sludge entering; that a line leaving the projecting overflow launder is clear water, and that a flow-line leaving the apex is thickened sludge.

This brings us back to the point as to whether it is best to show machines in plan or elevation in our conventional signs. This is a question which will have to be determined in each case. There are a great many machines of which I shall cite the Wilfley table as a case, where it will be necessary to so show the machine in plan. The Wilfley table has a character shaped so that it is an easy matter to determine which is the head and which is the foot. In all cases the point to be kept in mind is to produce a view which will embody some characteristic feature of the machine which will help to identify it and which will permit of the receiving and discharging point being shown. This is not as difficult a matter as it might be thought to be at first glance.

Not all engineers and metallurgists are expert draughtsmen, and for this reason it is very essential that the conventional signs used be of such a character that they can be easily drawn. Straight lines, circles and simple curves easily reproduced with the compass should be used to the total exclusion of curves of the conic section type. It is absolutely unnecessary that complicated curves should be employed in our conventional signs in order to make them clear. In the latest model of the Oliver continuous filter the bottom is rounded and the sides are allowed to flare out giving the end section an outline very much resembling an hyperbola. Fig. 11 is a conventional figure which I have proposed for representing this machine. If this was used in a flow-sheet without any comment as to what it represented, I believe very few engineers would be in doubt as to what it stood for.

In the accompanying sketches I have shown a number of conventional signs for use in flow-sheets describing cyanide practice. It will be noticed on close examination that each of these possesses some characteristic feature of the machine it stands for. The square, flat-bottom tank with the suspended filter-basket is typical of the Moore filter represented by Fig. 9. The Butters' filter, which differs from the Moore only in the shape of the tank, is shown in Fig. 10 with the characteristic pointed bottom. Fig. 11 is the Oliver filter, of which mention has already been made. Fig. 12 is the Burt filter—a long tube mounted on trunnions. Fig. 13 shows the Hendryx combined filter and agitator. These five machines are practically the only vacuum filters used in this country. The conventional signs proposed for representing them are entirely unlike and therefore there is absolutely no chance of one making a mistake in the identification when he sees it shown on a flow-sheet.

Figs. 14 to 19 inclusive represent the principal pressure filters employed in metallurgical work. With the exception of the Blaisdell filter shown in Fig. 18, the machines may be all classed as filter-presses. Since making the drawing, it occurred to me that it might be advisable to place on each one of these drawings, two horizontal lines, one on each side, as shown in Fig. 17. The horizontal lines represent the lugs cast on the sides of the frames which rest on the side rods which support the series of frames. These lugs are characteristic of all filter-presses and for that reason if they were shown on the conventional signs, they would serve at once to identify the machine as a filter-press.

The purpose of this article is simply to draw to the attention of the members of the society the necessity of standardizing the conventional signs employed in flow-sheets. From the points which I have brought out I believe that most of the members will agree with me that it is a subject worthy of our attention and I therefore leave it to others to take up where I leave off." James Speers. *Transactions American Metallurgical Society*, May, 1911, p. 7. (H. A. W.)

The Nahl Intermittent Slime Decanter.—"A description of a slime decanter, used by the Progreso Mining Company, Triumfo, Baja California, and adapted to the treatment of its low-grade oxidized silver ores, used instead of canvas leaves is given below. A 60° cone is constructed inside a wooden tank. The inside surface of the cone is constructed of brick and mortar finished with a thin surface of cement. Radial reinforcing walls one foot thick extend from the inside of the cone to the sides of the tank, and the space between these walls and the tank is filled with sand, or jig tailing.

The scrapers are balanced on both sides and are supported by a frame which runs on wheels on a track around the upper edge of the tank, and are steadied by a pin at the bottom. The slime enters the centre and settles on the sides of the cone, whence the coagulated slime is assisted to the bottom by the slow-moving scrapers without any stirring action. The clear solution overflows at the periphery of the tank. The alkalinity of the solution must be carefully watched to obtain the best results.

The frame carries a foam fence three feet high above the solution, and running down to six or eight inches below the surface. It is made of 1/32 in. steel, and is riveted to the inside of the sprocket band. The foam accumulates in this and dries, and drops in gobs to the bottom when it is too heavy to float. In this way it does not interfere with the clear overflow by being carried over the edge by winds.

The sprocket is a home-made affair. The pins are riveted in a band of ¼ × 3 in. iron. The pins are riveted about a foot apart and are set exactly to engage a No. 67 sprocket chain. It is revolved by a 10-in. sprocket. A smaller driving sprocket is used and the whole scraping apparatus is revolved by a worm gear at the bottom instead of the bevel gear. The chain is guided on the driven and also the driving sprockets by small rollers.

The slime accumulates, and is compressed by its own weight in the bottom discharge pipe, and is discharged intermittently through an appropriate gate at the bottom of the pipe into a car. If it is inconvenient to revolve the scrapers by mechanical power, a small boy can turn them around slowly every two or three hours or so by a crank fastened to the worm shaft."—Arthur C. Nahl, *Engineering and Mining Journal. The Pacific Miner*, March, 1911, p. 104. (A. R.)

Titanium Steel Rails.—"Titanium steel rails have now been in use experimentally on the New York Central and Hudson River Road for three years. The results have been so successful that the company's rail orders for 1911 include a large proportion of such rails. The titanium steel rails are hard and show a greater resistance to wear than ordinary rails. The specifications for these rails call for the use in the steel of 1% ferrotitanium, carrying from 10% to 15% titanium, making 0·1% to 0·15% titanium in the rail."—*Engineering and Mining Journal*, May 20th, 1911, p. 1010. (M. T. M.)

The Status of Commercial Electric Furnace Design and Practice in U.S.A.—"To illustrate the lack of appreciation of the significance of the remarkable developments of the last few years, on the part of men who might reasonably be expected to know better, Mr. Snyder tells a story of a man who was asked about the electric furnace paper on the programme of a recent meeting of an association of engineers representing an industry which uses a large amount of high-grade steel in various forms. This man, who had heard the paper and the discussion, characterised it as 'the usual optimistic hopes expressed by the agent of an electric furnace inventor.' He emphasizes the fact that the electric furnace has grown beyond a question of this inventor or that inventor; that it has grown beyond a question of hopes to one of commercial use, and is now a matter of engineering and finance.

And beyond steel making, the electric furnace has been commercially reliable so long in the aluminium industry that the general patents have run out. The commercial reliability of the electric furnace in the manufacture of carbide has for years been in evidence with each user of acetylene light. Less widely known, but commercially profitable, has been the electric furnace manufacture of carbon-bisulphide, of phosphorus, of graphite, of abrasive, of magnesium for flashlights, of ferro-alloys for steel refining, and in the last few years we have seen the reliability of electric furnace operations pledged in the investments of the millions of dollars that are going into the construction of huge electric furnace plants for the production of fertilizers. And it is the commercial reliability of these older electric furnace applications that is behind the reliability of the newer application of the electric furnace to steel. In them was developed the demand for electrodes which justified the building of electrode factories that now are available for steel furnaces. In them was developed the special engineering ability to handle the large electric currents of furnace work.

'Technically, reliability is a matter of apparatus and men and supplies. These react on one another. Experienced and skilled men can force poor supplies and indifferent apparatus to yield a commercial output. Apparatus that is inherently self-regulating will permit the use of such men as can be had at a permissible cost. Supplies that have a factor of safety in their construction will carry the equipment through an overload. Involved in the question of commercial reliability is the element of time. Results must be prompt. For this, reliable electric furnace supplies are now available and tested. Electrodes can be had in sizes as large as furnace requirements call for. Refractories have been developed with an ability to stand up well under the higher temperature of the electric steel furnace. Skilled electric furnace men are scarce. The rapid installation of furnaces more than absorbs the few who have had an opportunity to become experienced with electric furnaces working in commercial operation.

'But a reliable furnace and reliable supplies and reliable melters are not all that are concerned in the commercial reliability of electric steel. The steel produced is different from crucible steel. It is different from open-hearth steel. If it were not different, the electric furnace would not be used. It is true that under favourable conditions steel can be made cheaper with an electric furnace than in other ways, but the fundamental reason that is speeding electric furnace construction is that the steel is better. Being better means differing chemically, differing physically from the steels of the other furnaces, and this difference chemically and physically must be reckoned with in the subsequent processes the steel goes through. In the foundry the moulding must be done for this new material. It is freer from gases. It usually is hotter. The tendency is to lower the carbon for the same tensile strength, and this makes it set at a higher temperature. The improved elimination of phosphorus and sulphur tends in the same direction. In the forge and rolls, electric steel shows its additional strength and toughness. It must be handled from the point of view of its characteristics. It is a mistake simply to consider the electric steel furnace as a substitute for some other kind of melting furnace. Its introduction can be counted on to mean changes of method as far as the steel is followed in manufacturing processes, and unless this is clearly recognized there will be disappointment, and the reputation of the electric furnace for reliability will suffer.

'From a still narrower technical point of view, reliability means freedom from actual breakdown, from the enforced necessity of change in construction and the consequent loss of time and productive capacity. As to this, electric furnace design is now on an engineering basis. Dependence on experiment is now a matter of volition on the part of the investor. It is no longer a necessity. As a piece of engineering apparatus, an electric furnace can now be more closely designed to meet narrow specified operating conditions than can an open-hearth furnace. In a recent instance, an electric furnace and a small open-hearth were installed side by side in a new steel-casting shop. The open-hearth gave trouble and has been replaced by a better design. The electric furnace, although not of most modern design, has operated without need of alteration. Those details in which an electric furnace differs from established open-hearth practice, the electrode holders, the electrical regulation, the contacts with the bath of steel, are now rugged and simple. Electrode holders are in evidence that operate with less than one hour's lost time in 100 heats.

'It is to be expected that the success of this new and flexible tool of industry will lead to over-enthusiasm and its application in wrong ways and in wrong places, and that its failure as a panacea for troubles it has properly nothing to do with will bring setbacks. But behind its present success is the cause of that success, the steadily lowering cost of electric current. Thirty years ago the furnace of Siemens worked as an apparatus. It was not until the cost of current dropped below the figure at which the electric furnace was commercially reliable that its industrial application began and its engineering developed. This dropping of the cost of electricity is still going on, and as it goes on it brings a greater and greater motive force to bear on the installation of electric furnaces. At the same time, unfortunately for industry in general, but fortunately for the electric furnace, the cost of fuel for direct and gaseous heat is slowly but steadily rising.'"—*Engineering Magazine*, June, 1911, p. 488. (J.A.W.)

PREPARATION OF ASSAY SAMPLES.—"Of late much has been written regarding the accuracy of mechanical samples but no articles have come to my attention on the inaccuracies which are liable to occur in reducing large mechanical samples to assay pulps.

When mechanical samplers are properly designed and operated the resulting sample represents the entire lot in every respect. The weight of the mechanical sample will depend on the size and character of the ore. When an ore is spotted or non-homogeneous, high-grade, or if the lumps are comparatively large, the resultant weight of the mechanical sample must be relatively large. Great stress is laid on the proper weight of the sample with reference to its size and character but of still greater importance is the final preparation of the assay pulps. When the assays of buyer and seller do not check a third pulp is assayed by an umpire. This third assay may or may not check the previous assays and it may be necessary to resample the ore. These erratic results may be due to poor sampling, poor assaying, or to use the common excuse: the ore is spotted. In my opinion it is in the final preparation of assay pulps that most of the errors are made. If a spotted ore is ground sufficiently fine and the pulp thoroughly mixed there should be no trouble in obtaining check assays from different pulps on the same sample. Assay samples for the buyers and sellers taken from the same pulp should and will check when the samples are properly prepared.

After the mechanical sample has been obtained it is customary to reduce it in size and weight by hand, using sample grinders and riffles. Usually the man whose duty it is to perform this reduction and prepare the final pulp sample knows little or nothing about the inaccuracies which are liable to occur by the improper use of riffles, or the absolute necessity of thoroughly mixing the sample after each crushing and screening operation. Inaccuracies due to the improper use of riffles have been investigated by me and the results published. The object of the present investigation was to determine if there is an appreciable separation of the rich ore from the poor ore in pulverizing through a given mesh.

Separation in Pulverizing.—For the following investigation 100 kg. of Cripple Creek ore were selected. This ore is considered difficult to sample and from the occurrence of the rich tellurides along the fracture planes and the great difference in value between the coarse and fine, it can be considered a "spotted ore." The following samplings prove there is no difficulty in getting correct results providing great care is exercised in the various operations. The entire lot of ore was broken through ½ in. and riffled into two portions. These portions were in turn sampled and assayed, giving a result of 1·44 oz. gold. About 3,000 gm. of 20-mesh ore were selected from the rejects in sampling and treated as follows: All of the ore under 100 mesh was screened out and marked No. 1; the oversize was put through a coffee-mill sample grinder and the ore under 100 mesh again screened out and marked No. 2; this operation was repeated until all of the ore passed 100 mesh. Ten samples were obtained varying in assay from $48 for the first undersize to $8·20 for the last undersize, proving that, with ores containing the values in the brittle minerals, instead of the ore mixing in pulverizing, a statement often made by assayers and samplers, the rich ore is entirely separated from the poor ore if several passes are required in pulverizing. It is extremely difficult to mix pulps varying in assay such as are shown in the table and the larger the sample the more difficult it becomes. The necessity of mixing such fine pulps before the final subdivision into assay samples should be avoided.

Effect of Screening on Gold Segregation.

No.	Percentage by weight.	Assay.	Percentage of Content.	CUMULATIVE.		
				Weight.	Assay.	Content.
1	27·1	$48·00	45·20	27·1	48·00	45·20
2	14·5	37·60	19·20	41·6	44·20	64·40
3	7·8	29·60	8·05	49·4	41·80	72·45
4	7·1	22·20	5·50	56·5	39·50	77·95
5	8·7	18·60	5·60	65·2	36·60	83·55
6	11·6	16·20	6·55	76·8	33·70	90·10
7	12·8	13·80	6·15	89·6	30·80	96·25
8	4 5	11·20	1·75	94·1	29·90	98·00
9	2·9	11·60	1·15	97·0	29·33	99·15
10	3·0	8·20	0·85	100·0	28·66	100·00
Tl.	100·0	$28·66	100·00	……	……	……

The computed assay of the 10 samples, 1·434 oz. of gold, checks the original assay of 1·44 oz. The possibilities of error due to carelessly mixing fine ore before the final subdivision into assay samples can readily be seen by examining the accompanying table. Sample No. 1 contains only 27% by weight, 45% of the total value, and assays $48, whereas samples 7, 8, 9 and 10 contain 23% by weight, only 10% of the total value, and assay $12·35.

Avoid Screens in Pulverizing.—To secure accurate assay samples of fine pulps and avoid as far as possible the separation of the rich ore from the poor ore in sampling, I offer the following suggestions: The proper weight and size of the original mechanical sample must be determined for each ore. From here the theoretical reduction in weight will be the cube of the reduction in size; that is: If 2,000 lb. of 1 in. ore is the correct weight for an accurate sample, when this ore is reduced to ½ in. size the weight should be the cube of one-half, or one-eighth, which is 250 lb; when this is reduced to ¼ in. the weight should be one-eighth or 31 lb.; ⅛ in. should weigh 4 lb. and 1/16 in. should weigh ½ lb. The first weight of 250 lb. for ½ in. ore has always appeared high to me, and after sampling several lots I have found that original and duplicate samples check on lots of 50 lb. at ½ in.; for ¼ in. ore I prefer to have about 25 lb. which in turn checks the theoretical weight. Below this size I never follow the theoretical weights, as I consider them too small.

With these weights there is no trouble in securing check samples; ½ in. ore, 50 lb.; ¼ in., 25 lb., 10 mesh, 6 lb.; 20 mesh, 3 lb.; 40 mesh, 24 oz., and 60 mesh, 5 to 6 oz. The last weight of 5 to 6 oz. is the assay sample. This can be pulverized through 100 mesh before assaying, but I do not consider it essential for correct results on gold-silver assays where the valuable metals are carried by brittle material. I have had a large number of assays run on 60 mesh Cripple Creek ore, and by slow fusion and pouring at a high heat have had no difficulty in checking assays made on 100 mesh samples. I prefer assay pulps pulverized through 100 mesh to assist in the furnace work but not for the accuracy of results, providing the fusions are made properly. The errors I consider due to screening and repulverizing the oversize thereby separating the rich particles from the poor ore. Taking the results from the table we find the following conditions. If 41·6% is pulverized through 100 mesh in the first operation the undersize will assay $44·40; in recrushing, if 48% passes 100 mesh the product will assay $18·80; the product from recrushing this the third time contains only $10·80. These three products must be thoroughly mixed otherwise the resulting pulps will not check.

Thoroughly Mix after Each Crushing.—It is customary in reducing the smaller samples in size to use screens and recrush the oversize; this is especially true in reducing the 24 oz. sample through 100 or 120 mesh before riffling into four assay samples.

The table shows the great possibility of error by the use of screens as the ore is immediately separated into lots of different values. For this reason screens should be avoided and the crushing and pulverizing machines should be kept in such condition as to give the desired product without the necessity of screening and returning oversize. After each crushing operation with the smaller sizes the product should be mixed by rolling on rubber sheets before riffling. Coning should never be used for mixing or sampling. Self-discharging riffles of the Jones type should be used. Flat-bottomed riffles should only be used on the smaller weights when there is no possibility of overloading. By carefully observing the above suggestions there should be no difficulty in obtaining four assay samples or original and duplicate samples from the same lot of ore which will check."—L. D. Huntoon, *Engineering and Mining Journal*, June 24, 1911, p. 1249. (A. R.)

Electric Power to Melt Metals.—"Whether electric power is or is not economical for melting metals, is not alone a question of kilowatt-hour cost, but is also a function of the relative efficiencies of the fuel-using and the electric furnace. This question was recently discussed before the American Brass Founders' Association, by Prof. J. W. Richards, who gave the accompanying table of constants. The column 'just melted' shows the number of Calories required to bring a kilogram of the metal to bare fusion, while the column 'superheated 10%' shows the number of Calories required to bring a kilogram of the metal to a temperature 10% higher in Centigrade degrees than its actual melting point.

CONSTANTS OF FUSION.

	Melting Point.		Heat in Melted Metal.	
	Degrees C.	Latent Heat of Fusion.	Just Melted.	10% Superheat.
Ag	962	24	89	96
Al	645	100	258	278
An	1054	16	51	55
Bi	267	12	21	22
Ca	780	53	186	202
Cd	322	13	32	34
Co	1500	68	329	359
Cu	1085	43	162	176
Fe	1600	69	336	368
Ir	1950	28	113	123
K	58	16	27	28
Li	180	73	242	257
Mg	750	58	246	271
Na	97	32	78	81
Ni	1450	68	290	319
Pb	326	4	16	17
Pt	1775	24	89	102
Sb	632	40	74	78
Si	1430	128	555	622
Sn	232	14	28	29
Te	455	19	46	49
Th	290	6	16	17
Zn	419	23	68	73

Dr. Richards gives the efficiency of good oil furnaces at about 17%, coke furnaces at about 18%, and the electric furnace at 50%. On these figures, which would, however, have to be determined for each individual installation, coke at $4 is equal to oil at $6, and electric current at $12·25 per horse-power-year."—*Engineering and Mining Journal*, April 15, 1911, p. 765. (M.T.M.)

Metallurgy at the Globe and Phoenix.—"The Globe and Phœnix mine is so well known as the premier gold producer of Rhodesia that it is not necessary for me to give its history; suffice it to say that the Globe is not now being worked and that the Phœnix suddenly acquired fame when an enrichment was found extending several hundred feet above and below a horizontal dyke at a vertical depth of 1,700 ft. The increase in the value of the ore may be gauged by the fact that the reserve at the end of 1910 was 178,221 tons, averaging 33·9 dwt. per ton, as compared with 106,155 tons averaging 11 dwt. at the close of 1907. The ore contains sufficient antimony in the form of stibnite to make its treatment difficult both as regards cost and recovery. Much time and money have been spent in studying the question, and as the new plant is giving good results, I feel that the publication of details will be helpful to others confronted with a similar problem. It is not claimed that the method now adopted is in the nature of a new departure in metallurgy, but rather a combination of known processes applied to suit local conditions.

The ore is a hard white quartz, and, as the following analysis shows, with the exception of antimony, is singularly free from base metals:—

	Per cent.
Silica and insoluble	76·32
Antimony	0·73
Iron	2·58
Sulphur	0·91
Lime (CaO)...	1·21
Magnesia (MgO)	6·44
Arsenic	Trace
Copper	Trace
Gold	32 dwt. per ton.
Silver	3 dwt. per ton.

The stibnite is found in irregular patches throughout the mine, at some places fairly massive and free from gold, and at others intimately mixed with the quartz. In the rich ore (3 oz. gold per ton and over) the proportion of stibnite increases up to 3% and the pyrite is high in gold. Generally about 75% of the gold is free and recoverable by amalgamation; a considerable proportion of the 'rusty' or 'coated' gold is not caught on the plates but can be recovered in grinding pans.

An ore low both in antimony and gold presents a hopeless problem unless the gold can be liberated and caught by fine grinding. The antimony has an injurious effect on cyanidation owing to its easy solubility in alkaline solutions and its property of robbing the cyanide of the oxygen required. An example of this can be given in the case of the ore under discussion where, in cyaniding sand and slime with everything conducive to a good extraction, the presence of only 1% stibnite makes an extraction of more than 20% of the gold practically hopeless. The concentration of antimonial gold ore is also difficult owing to the tendency of antimony to slime. In the present instance the question was further complicated by the fact that a reduction plant was already in operation at the mine, and, on ore assaying up to 12 dwt. gold per ton, was giving good results. Moreover, the surface space available was so limited as to preclude a radical re-arrangement of the plant. The prospects of the rich ore continuing had to be carefully considered, and the position in June, 1909, when experiments commenced, made it necessary for a metallurgist to proceed with caution. Fortunately the ore has proved persistent and a further extension of plant will probably be made.

The original plant, erected ten years ago, was designed on the lines usual when dealing with a free milling ore and consisted of a 40 stamp mill, a cyanide leaching plant, and a slime decantation plant. The presence of antimony soon began to make itself felt, and in the sand and slime plant the extraction dropped to 40% and 30% respectively. It was discovered, however, that if the slime was run direct, without treatment, to dams, and stored there for about six months, and allowed to 'weather,' or oxidise, and then broken up and pumped to the slime plant, the extraction was raised to 75% or 80%. This course was henceforth adopted as far as the slime was concerned, and it was naturally imagined that the same result would ensue as regards the sand. But extended trials proved otherwise. After a close investigation of the sand then being treated, it was found that the antimony and pyrite were in a much coarser state than in the slime; moreover, the operation of the sand plant was handicapped by the presence of coarse particles of ore due to broken screens. Finally 'rusty' coarse gold was beginning to appear, and as there were then no grinding appliances to save it, a further loss in the sand residue was inevitable. Numerous experiments were made in aeration, length of treatment, and fine grinding and amalgamation, but all with unsatisfactory results, although the last mentioned method proved encouraging. An endeavour was then made by the erection of two grinding pans in the mill to regrind the coarsest of the sand, and, by the additional installation of sorting belts and jigs, to eliminate as much antimony as possible before the ore reached the stamps. The pans did fairly good work as far as saving of the coated gold was concerned, but were found insufficient in number to cope with all the coarse sand, while the sorting and jigging department (owing to the tendency of antimony to shatter into minute fragments) prove disappointing.

The extraction obtained by treating 74,492 tons during the year 1909 was:

	Extraction dwt. per ton.	Percentage of content extracted.
Extracted in mill	15·92	75·66
,, from sand ..	1·17	5·56
,, from slime partly weathered ...	0·76	3·61
,, total	17·85	84·83
Left in sand residue ...	2·77	13·17
Left in slime residue partly weathered	0·42	2·00
Original assay of ore ..	21·04	100·00

The extraction of 84·83% for the year 1909 is not alarmingly low, but occasionally when the mill for several days was engaged in crushing development ore from the enrichment close to the dyke, with a pulp assaying from 60 dwt. to 100 dwt. per ton, the extraction dropped (because of the antimony) to about 75%, leaving a residue containing about 15 dwt. As the reserve of rich ore had increased, the management decided to take steps to improve the extraction. In doing so the following points had to be remembered:

(1) That a plant already existed at the mine.

(2) That about 75% of the gold content could be obtained by amalgamating and grinding.

(3) That a good extraction could be obtained on slime up to 6 dwt. by storing in dams and weathering for about six months before cyanide treatment.

(4) That advantage must be taken of the brittleness of the antimony to grind the sand finely, and so force the antimony into the slime, where, by weathering, it would be rendered harmless. The fine sand remaining to be leached would then have but little antimony left to interfere with its treatment.

(5) That the ore reserve in the mine was increasing in value.

To make the experiments on a large scale 10 stamps were set apart, with two grinding pans followed by Wilfley and canvas tables. Trials were made on ore assaying from 11 dwt. to 25 dwt. per ton. In all cases the results showed an extraction of over 90%, in fact, on ore up to 15 dwt., the sand caught after fine grinding was of so low a value that cyaniding was unprofitable. The conjecture that finer grinding would drive most of the antimony into the slime proved correct, while the slime (50% of the tonnage milled) invariably had an original value about 50% higher than the sand, and as the slime by exposure to the atmosphere proved amenable to cyanidation a big advance was made in the successful solution of the problem.

All roasting of the ore also gave good results. If the ore increased greatly in value this process would probably have proved the best, but its adoption meant a dislocation of existing arrangements, besides a large capital outlay.

Armed with the experience thus furnished, the alterations were commenced early in 1910 and the full plant was running in December of the same year. The ore is now trammed to the crusher house, to fall on grizzlies with bars placed 2 in. apart. The fine passes into a storage bin; the oversize is partly shovelled and partly runs into two 15 by 9 in. Blake-Marsden crushers, and thence falls into another bin. This bin has iron doors discharging to an 18 in. belt conveyor, which takes it to a washing trommel. This trommel is 10 ft. 6 in. long by 5 ft. 6 in. diam. at the large and 2 ft. 9 in. at the smaller end, perforated with 1½ in. holes for the first four feet of its length and [illegible] in. holes for the remainder, and revolves at 18 r.p.m. The fine drops through the 1½ in. holes to another 18 in. belt conveyor, while the oversize passes along the trommel and, after being washed, to a 33 in. sorting belt travelling at 100 ft. per min., allowing part of the massive stibnite (½ ton to 1 ton per day) to be picked and stored pending further treatment. This sorting belt delivers the ore to the same belt as the fine from the trommel, and is conveyed to mill bins having a capacity of 500 tons. The ore is distributed in these bins by means of two 18 in. shuttle belts delivering into two revolving chutes. The mill consists of 40 stamps, each of 1,250 lb., and has been in use now for ten years. Each stamp drops 102 times per minute and crushes 5½ tons per day through a 25 mesh screen, followed by the usual amalgamated copper plate table. Three feet of the bottom end of the table is covered with a blanket to catch the coarse particles of 'rusty' gold that refuse to amalgamate. This product is afterwards amalgamated in a barrel. The tailing, on leaving the plates, runs into classifiers each 36 × 36 × 30 in. (one to each 10 stamps) where the slime is separated, while all the sand passes to 6 Record vanners (5 in use and 1 spare). Concentrators were installed to extract as much of the antimony and pyrite as possible, and also to act as a check in case any sudden addition of rich ore might raise the value of the tailing unduly. The concentrators precede the re-grinding plant, as the antimony is saved easiest when in coarse particles. It was also deemed advisable to impoverish the slime passing to the storage dams for weathering.

The fine slime overflowing the classifiers passes direct to the blanket-tables, while the sand goes to the vanners. The antimony and pyrite, containing about 70% of silica and other insoluble matter, pass from the tables to launders with a steep incline to a Frenier pump, by which they are elevated about 16 ft., and run to two settling tanks at the concentrate-roasting plant. The tailing from the vanners then passes into 10 grinding and amalgamating pans running at 55 r.p.m. These pans require about 8 h.p. per pan and treat 160 tons per day. The fine pulp from the pans, together with the overflow of the classifiers at the head of the vanners, passes over a blanket table 58 ft. wide by 38 ft. long divided into strakes each 24 in. wide. The blankets are removed at regular intervals and washed in pointed boxes. The fine concentrate from these boxes runs into the same Frenier pump as the concentrate from the vanners, and is pumped to the roasting plant. The pulp now finally leaves the mill building and flows to the sump of the tailing pump.

The following figures, taken over a week's run, illustrate the extraction obtained. In crushing such rich ore it is difficult to prevent violent oscillations in the grade (varying from 20 dwt. to 60 dwt. per ton) with corresponding variations in the value of the sand residue.

Extraction in Mill and Accessories.

	Gold Dwt.	Stibnite per c.	Pyrite per c.
Screen discharge	36·0	1·02	1·69
Leaving plates	15·0		
Slime overflow from classifiers (20% total tonnage) ...	12·0	2·02	2·30
Sand from classifiers (80% of tonnage	16·0	0·76	0·86
Sand leaving vanners and entering pans	11·25	0·17	0·54
Pulp leaving pans	6·25		
Pulp from pans running on blanket table and slime from classifiers	6·50	0·67	1·20
Pulp leaving blanket tables and entering cyanide plant ...	6·00	0·51	1·10
Total percentage extraction ...	83·50	50·00	35·00
Sand caught in sand collectors, 50% of mill tailing ...	4·5	0·4	0·85
Slime run to dams for weathering and subsequent treating	8·5	0·75	1·36

Sizing Results.

	Mesh Used. +40 p.c.	+60 p.c.	+100 p.c.	+150 p.c.	−150 p.c.
Battery Screening ..	23·4	18·3	22·4	10·4	25·5
Feed to pans ...	32 8	20·6	25·0	7·6	14·0
Discharge from pans...	0·2	1·6	34·2	19·8	42·2
Pulp leaving mill ...	0·1	1·0	17·8	15·4	65·7
Sand in collectors ...	0·1	0·8	24·2	29·2	45·7
Slime to dams ...	Nil	Nil	Nil	2·3	97·7

The pulp finally leaving the mill flows to a Forwood-Down Pump. A tailing wheel had been in use for ten years, but as the alterations necessitated a much higher elevation of the pulp, and space being limited, a pump was installed. After being elevated 51 ft. by the tailing pump the pulp flows down a launder to a large spitzkasten 61 × 6 × 6 ft. where the fine sand is separated for the sand collectors, while the slimy water overflows to the slime collectors. The sand collectors consist of 4 steel tanks, each 30 ft. diam. × 7 ft. deep, superimposed over 4 steel leaching tanks, 30 ft. diam. × 8 ft. deep. The sand produced is about 50% of the tonnage milled, and owing to its fineness the collectors are only filled to a depth of 6 ft., each charge being subjected to about six to eight days actual treatment in the lower tanks. Sizing tests of the sand show :

Mesh		per cent.
+ 40 ...	...	... 0·1
+ 60 ...	..	... 0·8
+ 100 ..	...	... 24·2
+ 150 ...	...	... 29·2
− 150 ...	...	... 45·7
		100·0

As already mentioned the extraction on the sand had been unsatisfactory, and even after the alterations the residue from high-grade ore (over 30 dwt.) is too valuable to be rejected. Numerous attempts have been made to reduce the sand-residue to an economic limit : for instance, several tons were exposed to the action of the atmosphere for months but with an indifferent result. The largest loss is in the *minus* 150 product, which contains practically all the fine antimony and pyrite.

As before and after the alteration of the plant it had been found necessary to crush rich ore on various occasions, the following schedule is interesting.

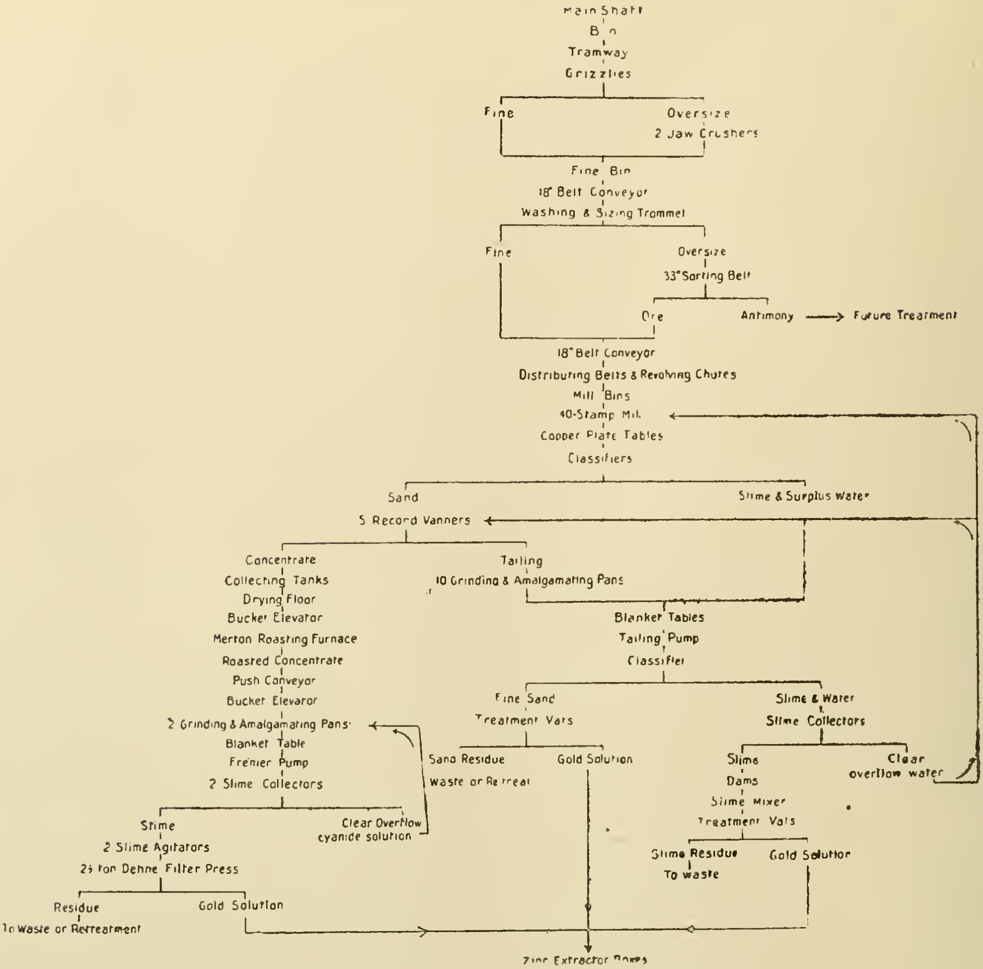
It will be observed that as soon as the antimony in the sand charge exceeds (on high-grade ore) 0·2%, the extraction drops rapidly. As before the alteration to the plant the sand treated constituted 80% of the tonnage milled, as against only 50% at the present time, the general loss in residue per ton milled is now obviously much lower in proportion than the above figures indicate.

Results of Rich Ore.

Battery screening dwt.	Antimony in sand per cent	BEFORE ALTERATIONS. Sand charge dwt.	Sand residue dwt.	Gold extracted per cent	AFTER ALTERATIONS. Sand charge dwt.	Sand residue dwt.	Gold extracted per cent
15	0·1	4·0	2 5	37·5	1·5	·50	66·6
17	0·1	5·5	3·5	36·3	2·0	·75	62·5
20	0 15	6·0	3·75	37·5	2·5	1·00	60·0
25	0 20	8·0	5·00	37·5	3·7	1·50	60·0
30	0·26	10·0	7·00	30·0	4·5	2·0	55·5
40	0·42	12·0	9·50	27·7	5·7	3·50	38·6
60	0·60	17·0	12·00	29·4	7·5	5·00	33·3
80	0·81	20·0	16·00	20·0	8·75	6·50	25·7
100	1·10	25·0	20·00	20·0	12·00	8·50	29·1

The overflow of slime and water from the large spitzkasten placed over the sand tanks goes to two 35 × 14 × 17½ ft. collectors where the clear water is drawn off and pumped back to the mill supply tank, while the thick slime is drawn from the bottom and pumped to the storage dams. As soon as the accumulation has attained a thickness of about 6 ft. and contains about 10,000 tons of dry slime, the stream of thick untreated current slime is turned into another empty dam and so on. At the end of two months of dry weather the surface of a dam is solid enough to permit of its being ploughed to a depth of about a foot, and after a further week this ploughed portion is harrowed and allowed to stand for another month or so, or until the loose slime appears perfectly dry and oxidized. The slime is then shovelled into trucks and goes to a mixer. To permit of continuous weathering of the slime before cyanidation, it is necessary that several dams be in use at once, so that one is being filled, one is drying, and two are being ploughed and harrowed.

The mixing of the slime is done in a steel tank, 4 ft. diam. × 3 ft. 6 in. deep, in which is placed a steel cone with the large end rivetted to the top edge of the tank, while the apex of the cone has an opening of about 18 in. diam. and reaches to within 12 in of the bottom of the tank. In this opening revolves, at a speed of 200 r.p.m., a four-bladed propeller connected to vertical shafting and driven by means of pinion and spur wheel gearing. The slime has to pass the blades of the propeller and is then forced up between the side of the cone and tank, passing out of a 6 in. opening in the side of the



Flow Sheet Illustrating Scheme of Treatment.

latter. Cyanide solution is added to the mixer to make a pulp of about 1½ parts of solution to 1 of slime. The pulp on leaving runs into a tank or steady head, connected to the suction of a double-acting centrifugal pump, which elevates the pulp into the decantation plant. The capacity of the mixer itself is about 12 to 15 tons of dry slime per hour, and 20 h.-p. is required to operate the entire apparatus. The plant for the actual cyaniding was erected about 5 years ago and is of the usual decantation type, consisting of two collectors 35 × 14 × 17½ ft., six treatment tanks, also 35 × 14 × 17½ ft., three clarifying tanks 20 × 5 ft., with pipe connections, and centrifugal pumps for transfer of slime and decantation of solution into clarifiers and sumps. This plant has a capacity of about 3,000 tons of slime per month of an original value of 4 dwt. per ton, but when richer material is treated the capacity will be less. To obviate this the erection of a vacuum-filter plant is under consideration. The total cost of treating slime is 3s. 6d. per ton.

The extraction obtained on weathered slime of a value of 4 dwt. is from 75 to 80%, while on the same slime if taken direct from the mill the extraction drops to about 30%. For an extra expenditure of 6d. per ton therefore in the handling and weathering of slime in the dams, a further extraction of 45% is effected.

The concentrate from the vanners and blanket-tables is elevated by means of a Frenier pump into two small collecting-tanks 12 ft. diam. × 4 ft. deep, provided with filter-bottoms, and an overflow launder round the periphery, which takes the clear water and delivers into the sump of the tailing-pump. The concentrate collected each day is allowed to drain, and then shovelled upon a sloping floor to dry for 24 hours and thence fed by means of a chain-and-bucket elevator into the top hearth of the roasting-furnace. This is a Merton 3-hearth furnace with a finishing hearth-extension. On each hearth are four solid knife-edge rabbles, while the finishing hearth has three water-cooled rabbles with diamond-pointed feet. All the rabbles run at two revolutions per minute.

The successful treatment of an antimonial gold concentrate is always difficult. Various experiments on the raw concentrate gave extremely poor results, but roasting experiments held out a much better prospect, especially when compared with the only other course open, namely, bagging and shipment. When an ore or concentrate containing antimony is roasted, the volatile trioxide is first formed, which on coming into contact with air forms the infusible and stable tetroxide. Again, when antimony or stibnite is roasted in presence of the sulphides of other metals, antimoniates of these metals are formed. It is therefore impossible to get rid of all the antimony, while it is also found that, when a large percentage of this substance is present, the ore on heating swells considerably and becomes 'sticky.' These difficulties can be partly overcome by constant rabbling in the presence of a large proportion of sand or silica. Trials showed that 90% could be recovered, and it was hoped that by exposing the residue to the atmosphere in dumps from six to twelve months, and running it through the large decantation slime-plant, this extraction could be raised to at least 95 . The total cost of the treatment of the concentrate on a large scale was estimated to be about 25s. per ton. These figures have been borne out in practice: a comparison with bagging and shipping to England, freight alone costing £6 per ton, emphasizes the advantage of roasting the concentrate at the mine.

The roasted product is discharged into a push-conveyor, 30 ft. long with flights 12 in. wide running at 18 strokes per minute: this cools and conveys it to a small chain-and-bucket elevator, which delivers it to two 5 ft. grinding-and-amalgamating pans, running at 45 r.p.m., in which cyanide solution is used.

After grinding, the pulp flows from the pans over blanket-tables (which catch any fine 'rusty' gold or 'floured' mercury), thence into a Frenier pump, to be elevated 18 ft. to a classifier, which returns any oversize back to the pans, while the overflow passes into two 16 ft. × 6 ft. collectors provided with overflow launders. The overflow clear solution gravitates to a storage-tank supplying the pans, while the slime settles to the bottom and is discharged every 24 hours into two 16 ft. × 6 ft. tanks having agitators running at 14 r.p.m. After 24 hours agitation, a three-throw pump forces the pulp into a 2½ ton Dehne filter-press, where the gold solution is removed and the cake washed and discharged (containing about 12% moisture) into trucks, and dumped for oxidation and future treatment if the gold contents warrant. The capacity of the furnace is about 11 tons per day (depending on the proportion of antimony and sulphur present) while the capacity of the remainder of the plant is double this, so, in the event of any increase in output, the addition of another furnace only would mean that 600 tons of concentrate per month could be roasted, slimed, and cyanided. The whole of the floor around the pans and furnace and under the filter-press is cemented, so as to minimise loss of gold in solution or dust.

An analysis of the raw concentrate shows about 65% silica and insolubles, 5% antimony, and 9% sulphur, while the roasted product is found to contain 3% antimony (chiefly as antimoniate) and 0·72% sulphur (0·6% existing as sulphate and 0·12% sulphide). It will be noticed that while a good roast as regards the expulsion of sulphur is obtainable, the antimony is not so readily driven off. The loss in weight in roasting varies between 25% and 30%, depending on the quantity of sulphur in the raw concentrate. The gold in the concentrate varies between 2 and 15 oz. per ton, and in the residue from 2½ to 25 dwt. per ton. The extraction has ranged between 85% and 90% and an experiment on the roasted residue exposed for only three months to the atmosphere indicates a further extraction. The firing of the furnace is done with wood by native firemen, one white man per shift being in charge of the whole roasting-plant, although the same man could easily superintend the working of two more furnaces. The total labour is 3 white men and 15 natives per 24 hours. The running time is about 94% of the possible, and so far as the repairs on the whole roasting unit have been extremely small. The cost on a basis of 320 tons per month is as follows:

	Per ton. s.	Per ton. d.
White and native labour ...	10	0·26
Fuel for roasting	2	1·71
Cyanide	3	3 46
Power (including compressed air)	3	9·58
Assays, etc.	1	5·09
General stores (zinc, acid, filter-cloth, oil, etc.)	2	2·10
Total	22	10·20

With the addition of an extra roasting-furnace, it is anticipated that on 600 tons of concentrate per month, the total cost would be reduced to 17s. 6d. per ton.

It has been found that the concentrate requires to be brought to a high temperature very slowly, and as the knife-edge rabbles are rather against this, steps are being taken to change most of them for the solid rabbles with diamond-pointed feet.

About 60% of the gold is obtained by amalgamation in the pans and 40% by cyanide. The gold from both processes is melted into one bar, about 965 fine.

As the new plant has been in full operation about three months at the time of writing, sufficient data are available to estimate the average extraction, although, owing to the storage in dams of the slime for six months, the total extraction, including that on the actual slime produced each month, cannot be given. As thousands of tons of weathered slime have been treated, it is an easy matter to calculate the recovery as if the slime actually produced in any one month had been treated during that period.

Taking January last as an example, during which 6,539 tons were milled, the estimated extraction was as follows (allowing 80% recovery on the slime, which would be the result after six months lying exposed to the air):

Source.	Dwt. per ton milled.	Percentage per ton milled.
Stamps and pans ...	28 94	79 86
Concentrate ...	2·41	6·65
Sand	0·39	1·08
Slime	2·15	5 91
Total extracted ...	33 89	93 50
Concentrate residue ...	0·21	0·59
Sand Residue ...	1·60	1.49
Slime Residue ...	0·54	4·42
	36·24	100 00

The actual figures for January were an extraction of 91·31% on an original assay value of 35·29 dwt. The discrepancy in these figures and those given in the above schedule is due, as stated, to the fact that the weathered slime from the dams during the month was only about half the value of the actual slime produced.

Working Cost.

	s.	d.	s.	d.
Mining	6	10·17		
Development	4	0·00		
Hauling and pumping	3	0·71		
			13	10·88

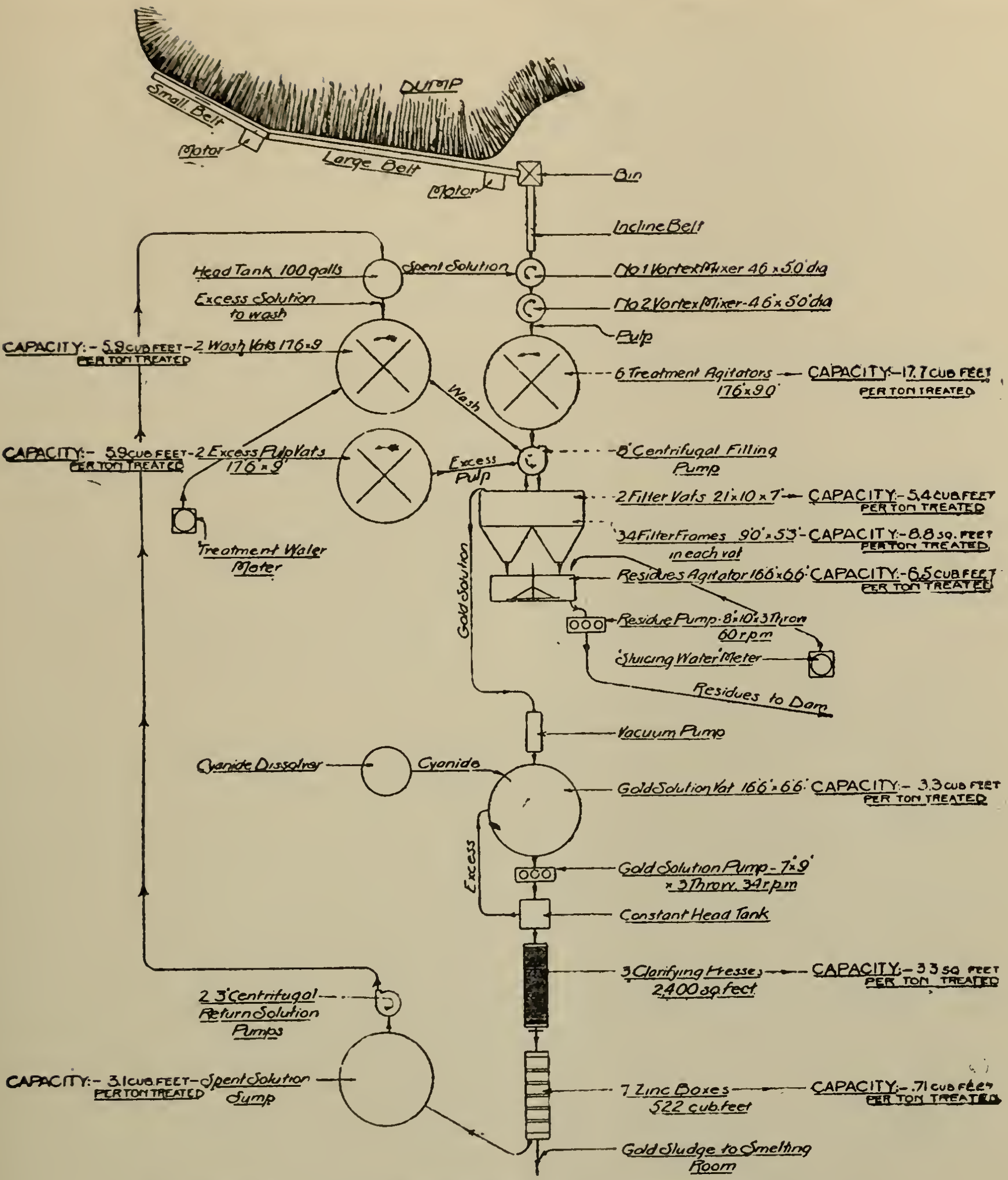
Transport and rock breaking ...	1	1		
Milling (inclusive of fine grinding and concentrating) ...	6	6		
Sand treatment	1	9·40		
Slimetreatment	1	7·26		
Concentrate treatment ...	1	1·15		
General repairs and maintenance		9·11		
Bank charges on bullion ...		4·94		
			13	2·86
General administration ...			2	9·33
Total cost			29	11·07

As the sand residue shows an unsatisfactory extraction on the high grade ore, future improvements will be required. One proposal is to eliminate the sand plant altogether, grind the sand in tube mills, and convey it together with the present slime to dams, for weathering. Against this, there is the question of surface room, as a large space is now appropriated in exposing to the air the present slime (3,000 tons per month) and to increase this to 6,000 tons per month necessitates double the dam area. The question of native labour (always a scarce commodity), in attending to the dams, affects the decision; there is also the question whether this sand can be slimed and in six months oxidize sufficiently to become amenable to cyaniding. As six months must elapse before this question can be ascertained (exposure to the atmosphere being the essential factor, not to be accelerated by any artificial means), it is obvious that if at the end of that time results are unsatisfactory, the position becomes awkward and a fairly large capital outlay is undertaken for no result. Another suggestion is to erect a roasting furnace of 100 tons capacity, together with necessary pans and tube mills, take the sand direct from the collectors and roast, slime, and cyanide it. This scheme would be a little more expensive than that just mentioned, but it is more certain of success as far as gold extraction is concerned.

It is suggested now to erect three large roasting-furnaces, the necessary pans, tube mills, and filter-plant, cut out the present sand-cyanide plant (except the collectors), roast, slime, and cyanide 200 tons of old accumulated sand together with 100 tons of current sand, or a total of 300 tons per day. The cost of treatment is not expected to exceed 8s. with a loss of gold in the residue of under 3s. per ton. The whole question is a matter of profit per ton and capital expenditure involved, with due regard to local conditions, and should the last named proposal be finally adopted—roasting the whole of the old accumulated and current sand—it must be admitted that it will be a unique undertaking."—H. T. BRETT. —*Mining Magazine*, July, 1911. (W. R. D.)

RE-TREATMENT OF TAILING AT OROYA-BROWNHILL.—"In the *Monthly Journal* of the Chamber of Mines of West Australia for March, T. B. Stevens and W. R. Degenhart describe the method and plant used for re-treating the dump at the Oroya-Brownhill, Kalgoorlie. The plant was erected at the end of 1908, and the dump then contained about 500,000 tons of tailing from the old filter-presses. The tailing consisted as to 92% of an all-slimed sulphide product that had been treated with bromo-cyanogen, and as to the remainder of roasted concentrate. The grade is described as the finest of sand, and only 2½% remains on a 100-mesh screen. The content of gold in the tailing averages 2 dwt., of which ½ dwt. is in the form of dissolved gold contained in the moisture. It was found after experiment that of the 1½ dwt. remaining undissolved only ½ dwt. could be extracted. An increase in the strength of the cyanide did not improve the extraction; in fact a greater amount could not be recovered without re-grinding, the cost of which of course would make such a policy out of the question. The company is therefore content with an extraction of 1 dwt. by means of this re-treatment plant. The capacity is 730 short tons per 24 hours: the recovery is worth 4s. per ton and the cost is 2s. 1d. per ton.

The tailing is in the form of a conical heap and was originally stacked by a belt-conveyor. The treatment plant was built immediately adjacent, and the material, on being broken with a pick, runs down in rills on to travelling belts, the position of which can be altered as the dump grows smaller. The tailing is first discharged from the belt into a bin, and then it goes on an inclined belt to a vortex mixer, where it is broken up and mixed with spent solution. The mixer is 5 ft. diam. and 4 ft. 6 in. deep, and rapidly revolving cast-iron blades are driven by a vertical central shaft. Heavy foreign bodies, such as pieces of iron, accumulate at the bottom, and occasionally they are large and heavy enough to cause the breakage of a blade. Manganese-steel was tried for these blades so as to prevent breakage, but the result was that the shafting was distorted if the blade did not break, a more expensive accident. The pulp passes by a launder to another mixer of the same type, and on the way floating matter such as wood is skimmed off by means of screens. From the second mixer the pulp goes to six agitation vats, made of galvanized iron, the sides being corrugated. Circulation is effected by means of four angle iron arms revolving at 12½ rpm. The spent solution entering the mixers contains an equivalent of 0·04% KCN. Owing to the presence of magnesium salts, little protective alkalinity is possible, and only 1½ lb. of lime per ton is added. The pulp is agitated for 8 hours, after which the cyanide content corresponds to 0·02% KCN. The specific gravity of the pulp is maintained at 1·61, as thinner pulp gives no increase in extraction. The consumption is 0·47 lb. of sodium cyanide. The pulp is then pumped to two vacuum-filters each 10 ft. wide, 21 ft. long, and 7 ft. deep, the bottoms of each being divided into two discharge-hoppers. Underneath residue agitators are built in the ground. There are two storage reservoirs for excess pulp, that is, pulp remaining between the filter-frames and in the discharge hoppers after the cakes are formed. In filling the filter, it is customary to draw from the excess vats and the treatment vats concurrently. There are also two storage tanks for wash-water. These are fitted with agitator arms, as it is easier to maintain this agitation than to periodically remove the fine sand that would otherwise accumulate. Each filter contains 34 frames, 5 ft. 3 in. deep and 9 ft. wide, and set 6½ in. apart from centre to centre. About 22 short tons of cake are formed per charge in each filter, and 16½ charges are made in 24 hours in each, or 33 altogether. The frames are allowed to stand a day per month in 5% hydrochloric acid, so as to free them from lime. The cloths last 5·3 months, and have then to be discarded because they have become hard with sulphate of lime, which cannot be removed. The vacuum pumps deliver the gold solution to a storage vat, to which is introduced an extra amount of cyanide to the solutions, making it test 0·05% KCN. The object of doing this is that the precipitation is effected better from strong solutions than from weak ones in the zinc-boxes. Before this practice was adopted, magnesium hydroxide was precipitated on the zinc and so stopped its action.


DUMP
Small Belt
Motor
Large Belt
Motor
Bin
Incline Belt
Head Tank 100 galls
Spent Solution
No 1 Vortex Mixer 4'6" x 5'0" dia
No 2 Vortex Mixer - 4'6" x 5'0" dia
Excess Solution to wash
Pulp
CAPACITY:- 5.9 CUB FEET PER TON TREATED - 2 Wash Vats 17'6" x 9'
6 Treatment Agitators 17'6" x 9'0" — CAPACITY:- 17.7 CUB FEET PER TON TREATED
Wash
CAPACITY:- 5.9 CUB FEET PER TON TREATED - 2 Excess Pulp Vats 17'6" x 9'
Excess Pulp
8" Centrifugal Filling Pump
2 Filter Vats 21' x 10' x 7' — CAPACITY:- 5.4 CUB FEET PER TON TREATED
34 Filter Frames 9'0" x 5'3" in each vat - CAPACITY:- 8.8 SQ. FEET PER TON TREATED
Treatment Water Meter
Gold Solution
Residues Agitator 16'6" x 6'6" CAPACITY:- 6.5 CUB FEET PER TON TREATED
Residue Pump - 8" x 10" x 3 Throw 60 r.p.m
Sluicing Water Meter
Residues to Dam
Vacuum Pump
Cyanide Dissolver
Cyanide
Gold Solution Vat 16'6" x 6'6" CAPACITY:- 3.3 CUB FEET PER TON TREATED
Excess
Gold Solution Pump - 7" x 9" x 3 Throw. 34 r.p.m
Constant Head Tank
3 Clarifying Presses 2,400 sq. feet — CAPACITY:- 3.3 SQ FEET PER TON TREATED
2 3" Centrifugal Return Solution Pumps
CAPACITY:- 3.1 CUB. FEET PER TON TREATED - Spent Solution Sump
7 Zinc Boxes 522 cub. feet — CAPACITY:- .71 CUB FEET PER TON TREATED
Gold Sludge to Smelting Room

FLOW SHEET OF PLANT FOR RE-TREATING TAILING AT OROYA-BROWNHILL.

On the other hand, stronger cyanide had no increased effect in dissolving the gold in the agitator-vats, so for the sake of economy the extra cyanide is added after filtration. The remainder of the treatment does not present any special feature. The article is full of working drawings and details that cannot be reproduced here, on account of want of space, but we recommend cyanide metallurgists to study the paper in full."—*The Mining Magazine*, June, 1911, p. 460. (A. R.)

SLIME SETTLEMENT.—"In treatment plants the finer portion, which is always alluded to as slime, never actually fulfils the conditions of any strict definitions, but always contains more or less fine sand. The observations and tests referred to here were made on the slime derived from the reduction plant on the Sons of Gwalia mine. The ore from the mine is mainly schistose in character, and consequently the stamp-milling and subsequent fine-grinding in pans produce a relatively large percentage of slime (50%), of which a great proportion is colloidal in nature and slow-settling under the action of gravity. The separation of the sand from the slime on the above mine is effected primarily by a row of five small spitzkasten classifiers each 3 × 3 × 3 ft. deep; the underflow goes to sand-collecting vats provided with Butters distributors and annular launders, which effect a further separation, while the slimy overflow joins that from the classifiers and is delivered thence to the slime treatment plant. This product, which is termed slime, is characteristic of the slime on most plants and contains nearly all the finely-reduced colloids together with a quantity of fine crystalloids or sand. Taken as a whole, it is slow-settling, and requires an extensive plant to thicken it sufficiently for economical cyanide treatment. The inflowing stream, which contains the equivalent of 6 to 8% of dry slime, first delivers to a nest of spitzkasten 30 × 40 ft. The underflow from these is drawn off through 1 in. nipples (into which wooden plugs with holes bored through them are inserted) and by substituting plugs with larger or smaller bore, as the case may require, it is regulated so as to keep the overflow just running clear. The underflow, which now ranges from 14 to 18% of dry slime, delivers to another nest of spitzkasten 32 × 24 ft., operated in a manner similar to the first, and the pulp is here thickened to from 23 to 27% (a clear overflow being maintained), and then it gravitates to two dewatering sumps working on the vacuum-filter principle, where it is finally brought up to from 40 to 45%. The quantity of slime handled ranges from 150 tons to 200 tons *per diem*.

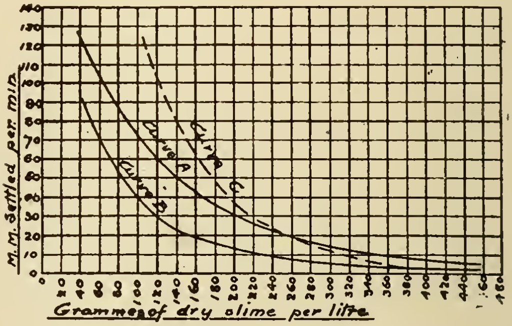
Taking a sample of the slime pulp as it comes from the sand plant, agitating in a glass cylinder, and then allowing to stand and settle, it was noticed in every case that a portion of the solids fell rapidly to the bottom. This portion, under strict definitions, would undoubtedly be sand. The pulp was then thickened by stages, decanting off part of the water at a time and agitating at every stage, and it was found that the tendency for the sand to separate out by hydraulic classification became less and less, the larger particles only being allowed to fall at intermediate stages, and finally this was eliminated, and the pulp ultimately in the thickest state showed no tendency to classify, but remained practically a homogeneous mixture. By analogy it may be said that all slime, no matter how carefully classified, will contain a certain proportion of relatively coarse particles depending upon the conditions under which it is made; even clays, which are the finest naturally-occurring slime, contain more SiO_2 than is required to satisfy the formula for the silicates present.

A slime may be described as a mixture of small particles of crushed ore of varying sizes, which, when mixed with water, tend to maintain a high degree of homogeneity, and as no degree of dilution has been mentioned under this definition, coarse sand may sometimes become a component part of a slime, or, in other words, there is no distinct line of demarcation. For example, the fine product from the classification at the Sons of Gwalia will separate out 20 to 30% at 8% thickness, as it is delivered to the first spitzkasten, while in the cyanide plant at 45% there is no tendency to classify at all, and it behaves just as any material coming under the strict definitions would. Is it, therefore, any less a slime?

The rate of settlement of a mixture of fine solids and water is governed by laws which have been the subject of much investigation, and it has been stated to depend upon many factors, amongst the most important of which are:—

1. The sizes and shapes of the particles.
2. The density of the solids and of the water.
3. The nature of the solids (colloidal or otherwise.
4. The presence of various electrolytes in the water.
5. The velocity of flow, if the pulp is in motion.
6. The ratio of solid to water, or thickness of pulp.

For any one plant treating the same class of ore these factors, except (5) and (6), remain very nearly constant, and a definite settling rate, determinable experimentally in a laboratory, can be assigned to a pulp at different thicknesses.



Slime Settling Curves

FIG. 1.

On the slime at the Sons of Gwalia mine, having a specific gravity of 2·7, and with water at a specific gravity of 1·05, a number of settling rates were determined for pulps of different thicknesses, and the results plotted to co-ordinates resulting in curve A, Fig. 1. This curve gives the exact nature of the slime dealt with, and will be referred to later on.

Many articles appear at times in the technical press in relation to slime-settling and devices used for that purpose, and in which sizes of particles and figures of results obtained are given; but very often no information is forthcoming as to the nature of the slime and the settling rate. The inclusion in those articles of a curve such as the above would render them immensely more valuable, in that there would then be a sound basis for comparison, as the curve gives the sum total of all the governing factors.

Another sample of slime was dipped from the overflow end of the second spitzkasten. It was taken from this part in order to secure a sample from which the free-settling portion had separated out. A curve was obtained for this (curve B, Fig. 1) and it will be noticed that for equal grammes per litre the settling rate of B is less than A, which is simply explained by the fact that A contained a proportion of free-settling material that in no way affects the rate, and, therefore, for a given number of grammes per litre, there is in reality a less weight of slime upon which the settling rate solely depends. This is clearly demonstrated by the fact that for equal ordinates the abscissa of B is always a fixed proportion of the abscissa of A.

Another portion of this slime (curve B) was taken and made up to 98 grms. per litre, and allowed to settle in a cylinder 3,600 mm. in depth. The falling rate was noted every five minutes.

TABLE A.

104 grammes per litre.

Time (in minutes).	Falling Rate (m.m. per min.)	Distance of Fall (m.m.).
0 to 5	39·5	197·5
5 to 10	39·5	395·0
10 to 15	43	610
15 to 20	42	820
20 to 25	42	1,030
25 to 30	41	1,235
30 to 35	38	1,425
35 to 40	35	1,600
40 to 45	31·5	1,757·5
45 to 50	26·5	1,890
50 to 55	23	2,005
55 to 60	19·5	2,102·5

The falling rate at first increases very slightly, and then rapidly diminishes after the bulk of the pulp is reduced by one-third.

A cylinder of slime was allowed to settle again, and during the operation, portions of the top-most layer were removed by a pipette and the specific gravity determined.

TABLE B.

Distance Settled. Millimetres (approx.).	Grammes per Litre of Top Layer.
nil	99
300	97
600	95
900	92
1,500	97
2,000	140

At the commencement the specific gravity was the same as the whole sample, and then it became less and finally heavier again. The constant removal of portions of slime and the disturbing action caused thereby, render it impossible to make the two tests on a small scale simultaneously, in order that the results might be correlated: but the general trend of both experiments confirm the assumption which is naturally expected from first principles, that is, that the settling rate for any one slime depends upon the density of the topmost layer.

The density of the topmost layer appears not to be altered to any great extent until the pulp has settled some considerable distance.

In a working spitzkasten the action is similar to the settling in a cylinder, save that the pulp is maintained at a constant level by introducing fresh slime, and the thickened slime is continually drawn off at the bottom; therefore, the pulp level never comes into the thicker zones, and the settling rate is that of the inflowing pulp. It is always found on this mine that the specific gravity of the top layer in the spitzkasten equals that of the inflowing pulp, less the free-settling sand.

The secondary spitzkasten in use here offer great facilities for observing the action of settling on the large scale. The settling area is 32 × 24 ft., made up of 12 inverted pyramids each 8 × 8 ft. at the top where they join, and 8 ft. deep. The junction of the boxes is 12 inches below the surface. The slime in this spitzkasten settles down in a very sharply-defined surface, leaving absolutely clear water floating above it, so that it is a simple matter to observe any movement of water or slime near the surface.

The pulp is fed into a half-submerged launder with holes bored at equal distances along the bottom for regular distribution of the feed. The following observations have been made:—

The pulp and water behave as two liquids of different specific gravity, the surface of the pulp is always a true horizontal plane, being always the same distance below the water surface in all parts of the area: the specific gravity of the surface layer of pulp is found to be the same in all parts. The quantities drawn off at the various points of discharge (provided all are kept running) make no difference to the efficiency of the settler, which depends solely on the area and the density, consequently nice adjustment to keep the pulp level at a constant height can be effected by altering the size of one or two plugs only in the whole nest, as may be required.

With the amounts of pulp settled in the two nests of spitzkasten, the movement in the pulp appears to cause little or no diminution in the settling rate, for, as nearly as can be judged by the water overflow, without going to elaborate means of determining quantities, the amount removed appears to be very nearly the product of the area by the settling rate in still water.

It is possible to determine the lateral rates of flow by immersing, in the water and pulp, bodies which just float in them.

Many devices, other than the rectangular nest of spitzkasten, have been brought forward, and it is claimed for some that they have a greater capacity, or produce thicker pulp. Among these may be mentioned the circular vat with central feed and peripheral overflow, and the settler from which the thickened slime is removed by means of slowly-moving submerged belts or chains. In cases of rapidly-settling slime, where movement of the pulp in adjusting its level is likely to interfere, the circular-vat settler would probably be an improvement, as the surface movement of pulp is very small in this case. So far as belts are concerned their sole advantage appears to be that they can be made to remove very thick pulp, which would be troublesome to keep running by the ordinary means of discharge.

At the Sons of Gwalia, the pulp-thickening process is of some moment, as exemplified by the extensive dewatering plant required, and any means to accelerate the rate of settlement would be of great advantage. From an inspection of the curve C (Fig. 1), which deals with the same slime as curve B—but with an excess of lime in the water shows an accelerated velocity—it would at first sight appear that the use of lime would afford an easy solution of the difficulty; unfortunately, however, the presence of large quantities of magnesium salts in the water makes the use of lime a commercial impossibility, as immense quantities would be required to maintain alkalinity. Experiments have, therefore, been made along lines which practically mean the creation of additional area in the existing arrangements. This consists simply of suspending in the pulp in an inclined position a number of sheets of corrugated iron, and it has been found to increase materially the rate of settlement, thereby making it possible to obtain a thicker pulp. Before giving the arrangement in detail, consideration of the following experiment will explain the principle involved.

A glass cylinder of slime is set up in a vertical position and the slime allowed to settle; the rate of settlement is 40 m.m. per minute. The cylinder is again agitated and then placed at an angle of 45° from the vertical; the rate is then 180 m.m. per minute, or nearly 4½ times as fast. The reason for this is quite obvious on close examination, for, on the upper side of the cylinder, a stream of clear water can be seen flowing upwards and joining the supernatant clear water above the pulp. Evidently the particles of slime settling in a vertical direction (indicated by arrows, Fig. 2) must leave upon any part of the tube facing downwards a layer of clear water, and this is immediately displaced upwards owing to its lesser density compared with the pulp that moves downwards to take the place of the displaced water. The inclined sheets of corrugated iron are an application of this principle; wherever there is a surface facing downwards, the pulp must settle away from it, leaving the water to be forced up. The corrugations deflect the flow into narrow streams which break through the surface at points and do not disturb the natural settling tendency of the slime.

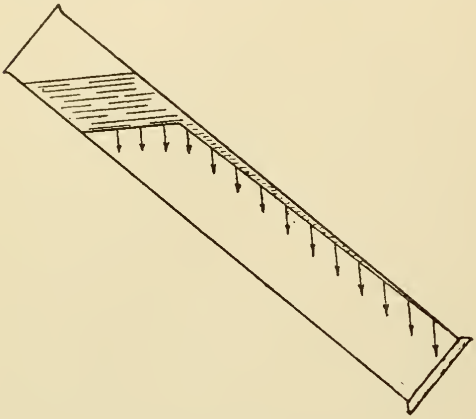


FIG. 2.

Where possible, it is best to set the corrugated sheets parallel with the current, so as to impede the flow of pulp to the least extent. The top edges of the sheet are set just flush with the surface, and by regulating with plugs in the underflow, the pulp level is maintained at from two to three inches below the surface. Small streams of water flow up each of the corrugations to join the clear water above, and the slime tends always to maintain its hydrostatic level—the demarcation of slime and water being a horizontal plane over the whole area—with the sheets of iron projecting through it to a greater or less extent according as the pulp is high or low. The natural-settling area is scarcely interfered with, the small jet of water in breaking through affecting it only to a slight extent, so that there is practically the original settling area plus the total area of the sheets by the cosine of the angle they make with the vertical, or the projection of their area to the horizontal plane.

The following figures show the comparative efficiency of the ordinary spitzkasten compared with the section fitted with the accelerating device:—

TABLE C.

No. of Test.	Spitzkasten 24 ft. × 8 ft. = 192 sq. ft., with settling device.		Spitzkasten 24 ft. × 24 ft. = 576 sq. ft., without settling device.	
	Amount Settled lb. per min.	Percentage of Dry Slime.	Amount Settled lb. per min.	Percentage of Dry Slime.
1	47·3	42·6	168·0	25·0
2	50·4	40·0	139·3	24·4
3	40·6	40·9	144·3	27·2
4	54·4	41·7	144·5	27·2
5	53·0	40·0	182·0	26·6
Average	49·1	41·1	155·6	26·1

These tests were all made on slime resulting from sulphide ore, the inflowing pulp being from 15 to 18%. The conditions were, that a clear water overflow had to be maintained, and that the pulp level had to be stationary at 2 to 3 inches under the surface, and maintained in that position for at least half-an-hour before any measurements were made. Endeavour was always made to have an amount of pulp proportional to their areas running into each section. The figures were obtained by catching the underflow in a receptacle over a measured time and by weighing the amount caught; from this and the specific gravity, which was also determined, the pounds per minute of dry slime were calculated. The averages show that the feed is very nearly proportional to the areas, so that the net result is to improve the thickness of the pulp from 26 to 41%.

It is proposed to fit the whole of the nest with the accelerators, and it is expected that the present mechanical unwatering plant will be wholly or, at least, partly dispensed with.

The area of the experimental spitzkasten is 192 sq. ft., and the total area of the inclined sheets used is 648 sq. ft.; they are inclined at 45°, so that the effective area is 648 cos. 45° = 453 sq. ft. Taking the inflow at the average value of 17% for both columns in Table C, and calculating the amount of water removed, then with the figure so obtained, divided by the area, the fall in inches per minute can be determined.

For the unaltered spitzkasten the fall is 0·107 in. per minute; for the spitzkasten provided with accelerators the fall is 0·168 in. per minute. The increase in the settling rate is not proportional to the increase in effective area. The reason for the difference lies in the fact that, of the sheets dipping some 18 in. into the pulp, the bottom portions are surrounded by a mixture of greater density than the surface layer, and consequently there is less settlement per unit area for those portions; moreover, the small uprising streams of water mix again, to a certain extent, with the pulp, and this tends to decrease the efficiency.

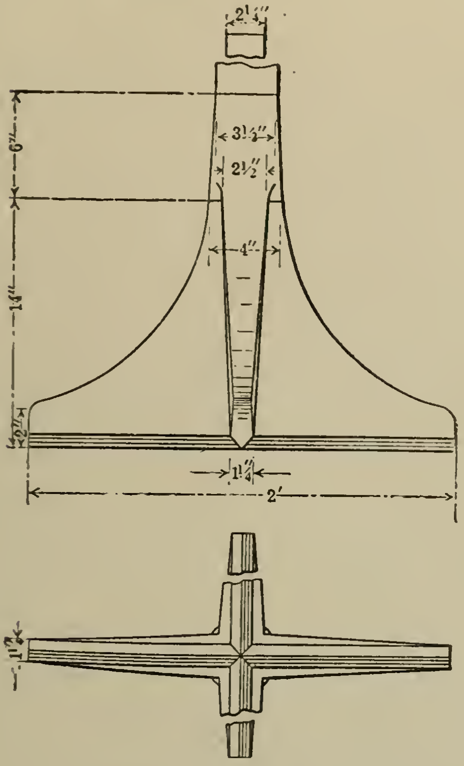
The chief merit of the device is that it can be applied to existing spitzkasten at a very small cost, and that a given amount of settling can be performed in a smaller space. It may be used for increasing the thickness of the pulp or enabling larger quantities of pulp to be handled, and should be very effective if applied to the decantation process, the greater rate of settlement increasing the capacity of the plant considerably.

The principle certainly has its limitations. Depending as it does upon the behaviour of the slime and water as two immiscible fluids, its effect is not so marked on thin pulp—the thin seeming to mix more easily than the thick—and the difference in density is not so noticeable. In all cases the simple laboratory test of the inclined glass cylinder gives a very good idea of the possibilities. By determining the settling rate in a vertical position and then inclining at different angles and calculating the increased effective area and the settling rate, a good idea can be formed of the results likely to be obtained on the larger scale.

The method of slime settlement described in this paper has been adopted at the Sons of Gwalia mine with complete success."—B. L. GARDINER, B.Sc.—(See also Richards "Ore Dressing," Vol. I., p. 459). *Monthly Journal of the Chamber of Mines of Western Australia*, May 31, 1911, p. 120. (H. A. W.)

MINING.

Cutting Hitches with Large Bits.—"In building the No. 10 dock of the Pennsylvania Company, in Ashtabula Harbour, Ohio, certain posts were to be set 4 ft. in the solid bedrock 27 ft. below mean lake level. According to *Engineering News* of May 18, 1911, the holes in bedrock were cut by three Ingersoll-Rand drills that had been used in drilling 2 in. holes. These were rigged with cross bits as shown in the accompanying illustration to drill 24 in. holes; the length of cutting edge was 24 in. from out to out. A small jet-pipe leading into the hole was used to wash out the cuttings.



Drill Bit with 24 in. Cutting Edge.

The successful use of such a large bit is interesting to miners, inasmuch as it suggests trying similar or perhaps smaller bits for cutting hitches for timbers. It might be difficult to use so large a bit for drilling horizontal holes, but for hitches of ordinary size considerable time might be saved by using a 10 in. or 12 in. bit."—*Engineering and Mining Journal*, June 17, 1911, p. 1197. (A. R.)

Results of the Fathomage System.—"We started with a reorganised method of mining, and we expected good results therefrom. The results obtained have surpassed our expectations. We are getting more gold and more profit out of the mine, and with less rush and wear and tear and straining to give record tonnage returns. What is it that has been sacrificed to do that? The only sufferer is that proportion of the tonnage that used to occupy a part of the plant and part of the labour of the underground workers—the tonnage that we found was of no assistance whatever in providing funds for the dividends you are receiving. In other words, we are now working only for our profit, and we gain that as reasonably and regularly as it is possible to do. A remark has been made by the chairman and by the consulting engineer as to the shortening of the life of the mine. From the point of view of getting all the profit we can out of the remaining life of the mine, there is no doubt that under the present system we are gaining a greater total profit than we should have otherwise done. There is one other point which I would mention to you. I wish again to call your attention to a seeming paradox which has been referred to before in connection with this change of policy, namely, that by working to profit instead of to low mining costs a considerable proportion of the unprofitable ore to-day eventually will become profitable. I have just got out a list of the percentages of payable and unpayable ore developed for the past two and a half years, and I find that since we started on our new policy our percentage of unpayable ore developed has systematically been reduced. During four of the last seven months the unpayable ore returns of development were absolutely nil, which is a record in the history of this company."—H. F. Marriott, Village Main Reef G. M. Co. Report for 1910.—*Transvaal Leader*, July 8, 1911. (A. R.)

Gold in Granite.—"In your February issue I notice that you refer to the reef worked on the farm Rietfontein, No. 1228, in the Lydenburg district, by a company (known as the Rietfontein (T.C.L.), Ltd.), recently floated by the Transvaal Consolidated Land and Exploration Company, and that you infer that this is a quartz reef in the dolomite similar to those flat reefs worked by the Transvaal Gold Mining Estates and other companies in the Lydenburg district.

I desire to point out, however, that the Rietfontein reef is actually a vertical reef in the older granite; and, as such, is of some special interest as being the only instance where development to any extent has been carried out on a quartz reef in granite—at any rate in the Transvaal, and, as far as I know, in South Africa.

This reef has been known for many years, but the local prejudice that 'no payable reef could be expected to live in granite' prevailed to such an extent that it was never properly prospected until two years ago when work was started on it and gold found to occur at intervals along the outcrop over a distance of 10,000 ft. Subsequent development has fully justified the confidence displayed, and certain well defined shoots have already been located, which led the chairman to make the statement the other week that 'from the results being obtained to-day it is now possible to state definitely that we have proved the existence of a payable gold mine here, and it is probable that in a few months' time the question of the erection of a 10-stamp battery will be considered.'

The importance of this occurrence is considerable when the enormous extent of the older granite—at present wholly unprospected—found in this country is considered, and it might be of interest if some of your correspondents could give experiences of gold-bearing quartz reefs within granite walls in other parts of the world. Personally, I know of no valid argument against the occurrence of gold in such reefs or their persistence in depth as compared with similar reefs in many other formations."—J. J. Garrard—*The Mining Magazine*, June, 1911, p. 437. (A. R.)

A Rock Drill Test at Cripple Creek.—“Beginning with the first of April, the Portland Gold Mining Co. of Cripple Creek is conducting a test of six air drills under the direction of Fred Jones, mine engineer. The makes represented are the Cleveland, Waugh, Ingersoll-Rand, Western, Shaw and Sullivan. Mr. Jones' statement for May follows :

	Cleveland.	Waugh.	Ingersoll-Rand.	Western.	Shaw.	Sullivan.
Shifts worked	27	29	24	6	28	29
Feet per shift	44·1	50·9	38·2	39	39	51·8
Average drilling time, hours	4·5	4·6	5	4·8	4·9	4·9
Average number drills dulled	22·2	22·1	25·6	24·7	26	20·3
Drills broken, per cent.	2·0	3·9	1·8	4	0·3	4·3
Average feet drilled per hour drilling ...	9·1	11	7·6	8·1	8·1	10·4
Number times in shop for repairs ...	1	1	3	1	3	1

‘These machines have been working almost the entire month of May on the 800 and 900 levels of the No. 2 shaft, in good, stiff breccia rock, with the exception of the Cleveland, which worked during the last week of the month in softer ground on one of the upper levels. The machines have been in the hands of, on the average, three different operators during the month.

‘This trial is still being continued, and the machines will not be calipered for wear until the conclusion of the trial.’

During April and May, the following averages in feet per hour are shown by the company's figures :

	Hard Ground.	Soft Ground.
Waugh ...	7·566	11·239
Cleveland ...	3·200	10·112
Shaw ...	6·833	10·475
Sullivan ...	not tried	10·192
Ingersoll-Rand	3·737	8·937.”

—*Mining Science*, June 15, 1911, p. 623. (A. R.)

Special Uses of Concrete in Mining Operations.—“In Europe, mine operations have been regarded as permanent operations ; hence for many years shafts have been sunk using brick or stone for linings, such shafts being circular or elliptical in cross-section. Where it has been necessary to use iron or steel casings to pass through water-bearing strata, permanent inner linings of brick have been used. For such inner linings, in some recent shafts concrete has been substituted.

In this country, until the past few years, it has been without exception the practice to employ wood linings, the shafts being rectangular in cross section. Water-bearing silts or quicksand locations were generally avoided, and where water was encountered in shaft sinking, no attempt was made to keep it out as in the European shafts.

In some instances where the surface quicksand proved shallow, concrete dams were constructed inside of timber linings or coffer dams were circular or ellipical in section and massive so that reinforcement was incidental. However, in sinking the air shaft of the Panama mine, Illinois, the writer used a reinforced-concrete lining of rectangular section through the surface silt of sufficient strength to resist the water pressure, so as to keep the air shaft dry. This is an important matter if the shaft is to be used for “intaking” aid since in a wet shaft, masses of ice accumulate in cold weather, thus restricting the ventilating current or stopping it with serious results.

The inner lining was put in after the shaft had been forced through the soft silt, and the outer wood curbing rested on this rock. This rock was a fissured limestone so the shaft was sunk about 8 ft. further into the shales below—here a good foundation was prepared for the concrete lining.

As the water spurted out above in streams in putting in the massive concrete foundation, the sinking pump was temporarily checked in putting in the successive courses, and the water allowed to rise above the level of the course to be laid, so that the concreting materials could be placed in water without currents behind the form. The concrete was then tightly rammed against the shale and rock to make tight joints.

When this foundation had been brought up to the level of the wood curbing, preparations were made for building up the thinner reinforced concrete walls, which were but 15 in. thick at the bottom, and 9 in. thick at the top, about 40 ft. higher. Diagonal wood strips were nailed on the inner face of the wood curbing and on these a tight lining of tongued and grooved flooring.

This deflected the water streams to the bottom of the lining, and short pipes were placed through the concrete forms at a number of points around the shaft so as to discharge the water from behind the wood lining into the open shaft. By this method, the reinforced walls were erected in comparative dryness.

The reinforcement consisted of corrugated bars placed horizontally and vertically near the inner face, *i.e.*, toward the centre of the shaft. The vertical bars served only to bond the courses, the horizontal bars were spaced close together at the bottom, and further apart at the top, according to calculated water and earth pressure, assuming no help from the outer wood curbing.

The horizontal reinforcement bars were bent around at the corners of the shaft, which were beveled. The shaft had two compartments about 9 × 10 ft., and 9 × 5 ft. in the clear, respectively, with an 8 in. concrete partition between.

When the concrete had sufficiently hardened, the drainage pipes were plugged, and the walls then served their object as a dam. There was a slight seepage at several points, but this gradually stopped.”—G. S. Rice.—National Association of Cement Users. *Mining World*, May 20, 1911, p. 1027. (A. R.)

The Word's Mineral Reserves.—“Of iron ore there is no lack, nor need anyone be apprehensive regarding the supply of this metal, but before very many years have passed the yield of the ore will have decidedly declined. While the falling off will be gradual, it will undoubtedly tend in the long run towards 40 %. This change is in itself important because, unless otherwise neutralised, it will raise the cost of production. It makes necessary the melting of more barren materials in the furnace, so that the consumption of fuel rises with respect to the amount of iron produced. It means also the mining and freighting of an additional burden which yields no return. From whatever point of view we regard it, other things being equal, the cost of production rises. The great reserves of lower grade ore than at present mined are in the Lake Superior district. They are silicious ores, and will require in smelting the admixture either of limestone or other iron ores high in the bases. The Clinton ores of Alabama are of this type and except for the hitherto unfortunate percentages of phosphorus which they might add to Lake Superior ores, they would doubtless make an advantageous mixture with the latter. But the southern ores are remote from the northern. In order to meet them at or near the supplies of fuel a long railway haul would be necessary. While this is not impossible it would add to the cost so greatly as to be highly improbable. There is one further consideration. The greater part of our pig iron is used in the manufacture of steel. For this purpose, in the two processes until recently most extensively employed, we need respectively either a very low or a fairly high percentage of phosphorus. If our irons are in between, and like the church at Laodicea, neither hot nor cold, they are ill-adapted to steel manufacture. Unless the growth of the open-hearth process introduces great changes, the mixture, therefore, of southern basic ores and northern silicious ones is not altogether promising for this reason.

The greatest cause of apprehension as regards present processes of iron manufacture lies in the supply of coking coal. We have built lofty furnaces, and in their use we place upon the fuel as it progresses downward in the furnace a heavy load of overlying ore and limestone. We need a very strong coke to stand up under the burden. The coals which yield these high grade cokes are found in a small portion of the total coal-bearing area, and the life of the supply is one of the very serious phases of the present situation. I do not know what the amount of reserves may be.

While these physical and chemical factors operate to increase costs, there is always the possibility of improved processes and of greater efficiency to keep them down. The improvement oftenest in people's minds to-day is the utilisation of water power to generate electricity which in turn may supply heat. Now, in a blast furnace smelting iron ores, one-third of the fuel is employed in reducing the iron oxide and two-thirds in developing the necessary heat for the reaction. Were we able with water power to economically furnish electricity and from it derive the necessary heat we might save the two-thirds of the present amount of required fuel. We might reduce costs. The remaining one-third of the fuel we should always need, but it is possible that poorer grades than high quality coke might answer. The saving would lie, of course, in the difference between the cost of the fuel and the cost of the electric current, provided the latter could be furnished more cheaply than the former.

The water powers in our own country, or at least in the more thickly settled portions of it, have not failed to attract attention, nor have they gone altogether unutilised. The more conveniently situated ones are already harnessed to the dynamos. But in countries like Norway and Sweden, where there are large water powers still available; where there are rich deposits of ore; and where coal fails, the applications of electricity to iron smelting are likely to be first worked out successfully. Data may be furnished in the lifetime of many of us which will cast light upon these improvements in their world-like relations.

The only other apparent possibility of reducing costs lies in the labour charges. Wages at present are not unduly high, and unless the increasing population of the country brings to pass an inevitable struggle for existence, which will cause the greater sub-division of tasks at lower proportionate returns; or unless the general reduction of expenses for subsistence makes lower wages possible, there would seem to be slight prospect of change in this item. In any event, the reduction from this cause cannot compensate the falling off in the yield of iron as foretold above.

Suppose iron goes up in cost—other conditions of our daily life remaining the same—transportation and all manufacturing based on machinery would become more expensive; and less freely carried on. Undoubtedly an appreciable pressure would be developed to turn our people back to the rural districts and to tilling the soil for a livelihood. The tendency under the stimulus of manufacturing development has been the other way. The migration of late years has been toward, not from, the cities. Shall we perhaps find in the long run, in the increasing cost of iron and steel, a partial solution of a much vexed problem? Will the cry, 'back to the soil,' receive support in a way not generally anticipated? The question is an interesting one for speculation.

The general inferences regarding copper are that the pinch of higher cost of production will be felt sooner than in the case of iron. We have no knowledge of such enduring reserves of copper ores as we have of iron. On the other hand, copper, despite its vast importance, is not the fundamental necessity that iron is. It is used in less quantity in machinery, and its increase in cost would less vitally effect manufacturing industries based on machinery. Advancing cost would cut it out of much ornamental work of inferior æsthetic merit. The most serious effect would be found in raising the expenses of service in the applications of electricity. Electrical transportation, telegraphy, and telephony would be more expensive than to-day. Unless wireless methods of transmission eliminate copper, or unless some discovery in the domain of physics which we do not now foresee furnishes a substitute for the omnipresent copper-wire of to-day, we may find ourselves face to face with some curtailment in these modern aids to the easy conduct of life's affairs.

The cases of lead and zinc are even more emphatic than that of copper. We have still fewer assured reserves, and the pinch of increasing cost may manifest itself at an earlier date The two metals are not, however, quite such vital factors in modern life as is copper, and the larger effects would be less apparent. Zinc is a necessary component in the manufacture of brass, the industry which absorbs the greater part of the copper output. A curtailment of either lead or zinc would cause inconvenience, but would scarcely occasion fundamental changes.

Silver will be very seriously affected by a decrease in the output of either copper or lead. Gold will feel these changes in an appreciable but far less degree. There will always be sufficient, however, of each of the precious metals for coinage, and beyond this use their applications, except perhaps in photography, concern luxuries rather than fundamental necessities. We cannot attribute to them any profound possibilities in their influence upon civilisation should the contributions of the mines decline. In the recent past we have been more apprehensive regarding a too great supply of the precious metals than regarding one too small.

With the increasing interest in the discussions of the conservation of natural resources, there has been an increasing disposition of the authorities to assume supervisory powers over mining and metallurgical operations. The old time idea that to the Crown, or, as we are accustomed to say, to the State, belongs the mineral wealth of the earth, is experiencing something of a revival. The disposition to restrict the waste of valuable metals or minerals in process of manufacture is commendable, and after careful demonstration that it is feasible and that it is just to the operating concerns, it may be wisely done. The rejected product of concentrating mills, or the 'tailings' so called, when provided with appreciable percentages of metals, may well be stored where they can be utilised by future generations, if processes improve so as to make them available. That is to say, they should not be run into rivers, or placed where they will be dissipated. The same remark applies to slag from metallurgical works. The moderns, for example, are now working over the lead-bearing slag left by the ancients at the great lead mines of Laurium, Greece. Even the slag of early smelters in the West and Mexico may again pass through the furnace."—J. F. KEMP. *London Mining Journal*, April 22nd, 1911, p. 410. (A. R.)

THE LIFE OF RAND MINING.—"The pastime of guessing at the duration of the life of the Rand mining industry has proved attractive to more than one of the various leading engineers who have had more than usually favourable opportunities for studying all the conditions surrounding this wonderful series of auriferous conglomerates. Although it is every day becoming more difficult to assign bounds to the economic limits of mining operations, upon which factor alone, as far as any of us are aware, the continuation of our advance underground depends, it has apparently not deterred Dr. Hatch from adding another forecast to those already recorded. If recent cablegrams do not misrepresent the substance of his utterances, there still remains to be obtained an additional output from the Witwatersrand mines amounting in value to 1,046 millions sterling, assuming a vertical limit of 6,000 feet. It is now about twenty years since Mr. Hamilton Smith almost staggered humanity by announcing that he foresaw the possibility of descending as far as 3,000 ft., and calculated that to this level it would be possible to extract about 200 million pounds' worth of gold, Bergrath Schmeisser afterwards admitted the feasibility of getting as far down as 4,000 ft., and estimated the productive result of this enterprise at 360 million pounds sterling. Later on again, M. de Launay set about the calculation, and arrived at the conclusion that from a depth of 3,000 ft. alone there should be extracted no less than 560 millions, and subsequently Messrs. Hatch and Chalmers, undaunted—encouraged, perhaps, by these vast globular sums, did not hesitate to prophesy that by going down another 600 ft. or so the total gold obtained might be as much as 700 millions. It was according to the nature of things that someone should go a little better, and in due course Mr. Hays Hammond, in 1901, weighed in with something like 800 millions down to a depth of 6,000 ft.

The Limit to Date.—Mr. Bleloch was the next of whom we have any reliable record, and, after the somewhat restrained efforts of the previous computers, it rather took one's breath away to learn that down to 7,000 ft. on the Central Rand, and from 3,000 to 6,000 ft. outside this boundary, or an average, say, of 6,000 ft., the recoverable gold would probably total as much as 2,800 millions. This prognostication represented the absolute limit apparently, for Dr. Hatch, again attacking the problem, this time with the assistance of Mr. Leggett, could not see his way to go beyond 1,240 millions sterling as the very outside of the Rand's future production, an operation which would probably occupy some forty years or so. Although there has been much talk since then of the lowering of grade and the possibility of treating ore which was formerly thought too poor to consider for a moment, it is to Dr. Hatch's credit that he refrains from the temptation to make our hair stand up by improving to any notable extent upon his 1901 estimate. He has allowed for the fact that in the meantime a very great deal of those 1,240 millions has been extracted from the earth, so that the latest calculation is of necessity rather lower than the previous one.

The Depth of the Deepest Deeps.—All these guesses are, of course, interesting, but, as we have already remarked, everything depends mainly upon the depth to which it will be possible to go down in the uttermost levels. The idea seems to be that 6,000 ft. is a safe figure to base calculations upon, and although it would be deeply interesting, and possibly very instructive to learn why this particular figure has been decided upon, it is precisely the point upon which information is wanting.

In the discussion upon Mr. H. Behr's paper upon 'Winding Plants for Great Depths,' Mr. H. H. Webb remarked that 'it would be rather daring to attempt to predict at what depth mining on the Rand would become unprofitable. Mr. Jennings admitted that a moderate profit might be expected at 8,000 ft., but considered this a sufficient depth for making plans. He (Mr. Webb) considered that sufficient reasons were not given why 8,000 ft. should be taken as the extreme depth. As engineers, he thought they were trying to work out a problem for the best and most economical means of attacking reefs, not only at 8,000 ft., but at 10,000 or even 12,000 ft., if their values warrant it.' Mr. Webb's view was that the extreme depths would be reached through inclined shafts. The fact of the matter is that no one can possibly say how long the Rand mining operations are going to be protracted, and, therefore, what amount of gold is likely to be extracted from the Main Reef series. There is not sufficient data at present for arriving at any conclusion upon the point which would be worth the paper it is written upon. The whole matter cannot be put better than it was two-and-a-half years ago by Mr. R. W. Schumacher, in an article which we then printed. Here are Mr. Schumacher's conclusions: 'In the future there will be a number of large groups of mines which, for reasons that have not yet been fully understood, and perhaps have not been thoroughly explained, will have lives of almost indefinite length. The reef at the lowest boundaries of these big groups will, as a general rule, be so deep—perhaps 8,000 ft. or more—that no new companies could profitably start work further to the south. The capital outlay would be too enormous, without even considering the mining risks involved. For instance, it is unthinkable that independent work will be started by holders of claims to the south of the proposed Crown Mines, the enlarged Village Deep, the City Deep, the Knight Central, or of the East Rand Proprietary Mines. In all these instances, which are typical of others, the reef will be followed to its utmost limit of payability by existing companies, and it will always be possible for them to acquire further areas to the south, should they desire to do so. In other words, all the claims to the south of these big groups will, in the natural course of events, gradually fall to them without any effort on their part. On the whole, it may be confidently stated, therefore, that it will be possible in some form or other to extend almost indefinitely the life of an enormous percentage of the existing gold-mining securities of the Witwatersrand. There will naturally be some exceptions, but they will be very few indeed.

When working under favourable conditions, as just outlined, the life of the Witwatersrand mines will be limited by one factor only, *i.e.*, the margin of profit between the value of the gold won and the cost of its production. As long as the ore is payable work will continue. Questions of deep haulage, ventilation, inflow of water, etc., need not be feared; they are comparatively minor details, and will be settled by mechanical means without serious difficulty. The following vital questions, therefore, naturally present themselves to people interested in or dependent on the Transvaal gold industry: —

For how many years will the reefs remain payable?

Will the life of the fields be long, or are there perhaps already indications that the end is not far distant?

Fortunately it is possible to give a most reassuring answer. The end is entirely out of sight, and there is no single disquieting feature on the horizon. Indeed, the prospects have never been as bright as they are to-day. Working costs have recently been greatly reduced, and it is certain that further important economies will be effected. An immense tonnage of ore hitherto considered unpayable, and neglected in most calculations, is thus brought within the scope of profitable treatment. Then, also, an overwhelming amount of information has been acquired during the last few years, all leading to the conclusion that the gold reefs will prove to be payable to a far greater depth than was thought only a little while ago. Let us for a moment consider the values of the deepest developments of the whole Witwatersrand from east to west. As is known, very promising results have been obtained recently in the extreme eastern portion of the Rand, notably in the deeper levels of the New Modderfontein mine, in the Geduld property, and in the eastern shaft of the Brakpan Mines, where most encouraging results have been obtained at a depth exceeding 3,500 ft. Coming westward, we find that in the Cinderella Deep some of the values have been of distinctly high grade. In the Central Rand, recent developments in the City Deep have exposed values considerably exceeding the present average of the outcrop mines, and the reefs in the South Rand property are also proving rich. In the Western Rand, the lowest levels that are being exploited are exposing ore of good grade, notably in the Main Reef West, the Durban Roodepoort Deep, and several other properties, extending right up to and including Randfontein. It is safe to say, therefore, that the very lowest workings throughout the Witwatersrand goldfields have, during the last year or two, exposed ore of a promising nature; and while poor patches will undoubtedly be encountered from time to time, as is always the case in gold mining, there are no indications to hand which would lead one to fear that the reefs payable to-day are likely to become permanently impoverished within any measurable period. It is well to bear in mind also that the lateral limits of the Witwatersrand are not defined; there are great areas of ground unproved to-day, especially on the eastern fringe, which may be profitably worked before very long. One can, in fact, see no limit of life to these goldfields, and there is no justification whatever for arbitrarily imposing, as has often been done, any fixed term of years, such as fifty, a hundred, or even considerably more, as a maximum period of their activity and welfare. There is reason to believe that these facts are not generally known, and that the great improvements that are being brought about by working with larger units, on a bigger scale, and with the prospect of an almost endless life, are not fully realised. When they become clear there will arise a sense of security and permanency that has hitherto not existed in regard to the gold industry in South Africa, and this factor is bound to react favourably on the fortunes of the whole country.' "—*The South African Mining Journal*, July 8, 1911, p. 775. (A. R.)

MINING ACCIDENTS AND THE RENDERING OF FIRST AID.—"The author deals with the various accidents arising in mining operations, and offers suggestions as to the prevention of those accidents that are due to causes over which no one has control. These accidents are much less frequent than accidents attributable to preventable causes. As regards the danger of gas containing carbon dioxide, carbon monoxide, nitrogen, fumes of nitro-glycerine, etc., carbon-monoxide poisoning is probably the most fatal. Cases of gas-poisoning may be divided into two classes, namely, acute and chronic: the first class resulting where a limited quantity of the gas is inhaled, and the second where a considerable quantity of gas is inhaled in a very short time. The symptoms in the acute cases are (1) Beating in the head; (2) flushed condition of the skin, especially of the upper part of the body; (3) giddiness; (4) nausea, and usually vomiting; (5) difficulty of respiration; (6) restlessness; and (7) fullness and compressibility of the pulse.

When a large amount of the gas is inhaled, there ensues usually giddiness, passing into unconsciousness in a short time. After the unconscious period, if the patient recovers therefrom, follows a drowsiness or languor and an intermittent pulse, with considerable difficulty in respiration.

In the chronic cases of gas-poisoning where men are constantly inhaling small quantities of the gas, the four main symptons are:—(1) Slight persistent headache, (2) cough, (3) indigestion, and (4) nerve disorders.

The gas may be absorbed by carrying dynamite in the boots, or by rubbing the face with the hands after handling dynamite. One instance is quoted of a man having eaten an apple after having cut a stick of dynamite with a knife, and the same effects were reproduced. The writer is of opinion, however, that the main symptoms are due to a volatilization of the fumes of nitroglycerine rather than to absorption through an unbroken skin.

Dealing with the treatment of such cases, the author recommends the removal of the patient to fresh air as quickly as possible; but if, for any reason, such a course is impossible, to force air into the immediately vicinity. This should be followed by keeping the patient's body warm with blankets, but the head cool, and by his removal to a room where there is an abundance of fresh air.

After dealing with the treatment of venous and arterial hæmorrhage, head wounds, fractures, and eye-wounds, the author points out the importance of the prevention of shock to the patient, and makes the following recommendations in the case of an accident of a serious nature:—(1) Place the patient in as warm a room as possible ; (2) keep the patient lying down ; (3) allow the patient water, diluted whisky, or alcohol, if any is at hand ; (4) do not allow too many other persons in the rooms in which the patient is placed ; and (5) prevent, if possible, the patient from learning of the seriousness of his accident."—C. H. HAIR.—*Transactions of the Institution of Mining Engineers*, 1911, p. 616. (M. H. C.)

DRIFTING RECORDS.—The Vekol Mining Co., near Casa Grande, Pinal County, Ariz., is advancing a drift in hard blue limestone from the 120 ft. level. The size of the drift is 8 × 6 × 4½ ft., and the working face is now about 500 ft. from the shaft. The drilling is done with a Sullivan 2¾ in. air drill on a double screw mining column. Two drill runners recently had a contest to see which one could break the most rock in five shifts, with the results shown in Table I.

The time for setting up included the time required by the men in walking from the shaft station on the 120 ft. level until the air was turned on at the drill. The total working time included setting up, drilling, tearing down, loading, and blasting. Seventy-five pounds of 1¼ in. powder were used per round of holes. A Sullivan straight-line, two-stage compressor furnished power for the drills at 110 lb. receiver pressure.

Shifts.	1	2	3	4	5	Averages.
Minutes to set up	26 min.	16 min.	19	21	17	19·8
Number of holes drilled ...	11	11	12	10	12	11·2
Number of feet drilled ...	94	95	102	85	92	93·6
Average depth of holes in feet	8·54	8·6	8·5	8·5	7·66	8·36
Total working time ...	7 h. 54 m.	6 h. 25 m.	7 h. 15 m.	7 h. 10 m.	7 h. 2 m.	7 h. 9 m.
Feet of advance per shift ...	7 ft.	6 ft. 11 in.	6 ft. 9 in.	5 ft. 1 in.	5 ft. ½ in.	6 ft. 2 in.

The above information was furnished by W. J. Forbach, manager of the Vekol Mining Co."—*Mines and Minerals*, May, 1911, p. 606. (A. R.)

COLLECTING GOLD FROM PANNINGS.—"In estimating gold in gravels in India and West Africa, there are certain visible physical properties, such as difference in colour and form due to a slight variation of hardness, which, although the eye is in good order and the judgment clear, may lead to wrong estimates until check samples are taken and the eye accustomed to the new condition of the metal.

With the amount of black sand common to most alluvials mixed with fine gold the operation of forming a clean tail of gold is often a compound operation. The greater bulk or mass, the crystalline form, and the consequently greater friction of heavy black sand, causes it to be comparatively sluggish in travelling on the surface of the contrivance used for panning and swirling the sample in the washing process, while the tendency of minute specks of gold, often polished and of a hardness which gives them a suitable form for moving comparatively quick in relation to their density, is to work their way into the natural crevices of the black sand.

Until my sense of judgment becomes sufficiently trained to fix the relative size and weights of gold when sampling in a new field, I have for years collected samples for check as follows :

(a) If a large amount of black sand is left in the concentrate and a rich tail of gold is formed, I take off the bulk with a small globule of clean mercury, manipulating so that it does not mix with the base metal in the concentrate. I then pour off the amalgam, and with a few spoonfuls of water carefully bring down about half the concentrate to a lower point in the pan. After working up, if no gold shows, I brush it off the pan, transferring it to a white enamelled shallow plaque about 10 in. diameter, when the finest gold can be easily and quickly seperated, the white plaque intensifying the black of the base concentrate. I then pick off again with the mercury, clean and weigh the gold.

(b) Pan down to a small bulk, transferring on to a plaque to finish off, when the gold can be quickly taken off with a small straw or glass quill. The quickest and simplest way of taking it off when the gold is clean, is by means of a piece of ordinary wax candle worked soft with the fingers, and formed into small elongated pills or pencils for taking up solitary specks. One wax candle will take and hold firmly 80 to 100 ordinary samples ; the wax sets quickly and can be wrapped in thin grease tissue paper, and will keep for an indefinite period, away from heat. Subsequently the wax can be dissolved off quickly with heat and will leave the gold clean.

(c) Pan off to a small bulk, dry, and take out all magnetic particles with magnet ; be careful that fine gold is not lifted and carried up with the iron ; roast, amalgamate, and then weigh.

A temporary roast in the bush on a wet day can be carried out quickly for minute samples, as follows: Mix up nitre, charcoal and sulphur to form a combustible mixture ; make into a stiff paste, place the charge in the centre, ignite, and the roast is complete in a few seconds.

A paste made of gunpowder or a small piece of dynamite, with care, can also be used ; in the latter cases, a plate held a few inches above the charge prevents loss from sparks.

A good method is to collect the results from half a dozen pans, which greatly facilitates weighing with bush instruments or portable scales. The plaque as described is, in my opinion, the best appliance for finishing off small samples ; it is light, simple to use, and requires a minimum of water, and, with a common wax candle, samples of gold and other metals can be quickly and accurately taken and reduced when convenient."—JOHN SAXTON.—*Institution of Mining and Metallurgy.*—*Mining World*, April 8, 1911, p. 740. (A. R.)

WORLD'S DEEPEST BORING.—"What is claimed to be the world's deepest boring is a hole 2,240 metres, or 7,347 ft., deep, drilled in Upper Silesia. The boring was undertaken to determine the relations of the coal beds in the Knurow royal mining district, and was conducted by the Schöneback division of the Prussian Royal Drilling Bureau. The locality of the drilling is 1·3 km. north of Czuchow village, in the Rybnik district. The drill used was a crude combination of the scoop or wimble, chisel, and diamond-bit types, in conjunction with water rinsing. Operations were begun September 25, 1906, but the outfit was not installed until October 15.

The mouth of the hole was some 440 mm. wide, exclusive of casing, but the diameter gradually diminished to 92 mm. at a depth 979·93 metres, which also marks the lower limit of casing. From this point down to 1727·54 metres depth the diameter was 91 to 92 mm. : down to 1,749 metres, the diameter was 67·5 mm. : down to 2,088 meters, the diameter was 50 mm. : the remainder of the hole is 48 mm. wide.

A total of 982 days was consumed in drilling, including 169 Sundays and holidays, and 119 days in accessory operations as in drill repairing, etc., leaving 694 net working days. The maximum speed attained was a 24 hour record of 16·73 metres with the 91 mm. diamond bit. Altogether 704 3/32 carats of diamonds were consumed. The total cost of the drilling was 323,712 marks and 47 pfennig, or $77,043·57, representing an average of 144·53 marks per metre, or $1·04 per ft.

The list below shows the Prussian drillings exceeding 1,600 metres (5,250 ft.) in depth, and the cost per metre in marks (the German mark is 23·8 cents.) :

	Depth (metres.)	Cost per M. (marks.)
Hoetmar	1,634·76	78·26
Schladebach	1,748·40	121·43
Ottweiler	1,803·36	81·58
Everswinkel	1,814·50	60·47
Paruschowitz V	2,003·34	37·55
Schubin	2,149·45	68·51
Czuchow	2,239·72	144·53

Mining World, May 27, 1911, p. 1074. (A. R.)

MISCELLANEOUS.

VALUATION OF TIN ORES.—"Your Cornish Correspondent, in your issue, May 6 (p. 471), has touched upon an interesting subject when quoting from Mr. C. F. Thomas's paper, read before the Chemical and Metallurgical Society of S.A. In view of the not altogether justified outcry which from time to time has been raised by various correspondents and others against the charges of the tin smelter, it may be interesting to compare the smelting charges made in various parts of the world. It must, however, be borne in mind at the outset that two essential factors must determine the smelting charge. (1) The purity of the ore—*i.e.*, absence of deleterious elements and minerals such as sulphur, arsenic, wolfram, rutile, magnetite, &c. (2) The relative position of the supply of ores and the ruling price of white metal.

Mr. C. F. Thomas's valuation of ores is a somewhat complicated one, and to my mind suffers from one weak point. His calculation is to allow a margin of £2 to £3 below cash price of tin, and then a deduction of 6·25% is to be made from the assay value of the ore. Presumably this 6·25% is to cover smelting charges, including loss of metal. With the present price of white metal at, say, £190 per ton, this equals nearly £12, which is not far out as a total returning charge, including the 3 lb. Cornish draftage. Let us, however, take the white metal at, say, £120 per ton, then 6·25% equal about £7 10s., which obviously is too low. No hard and fast rule as to returning charges can be established, as pointed out before : but yet figures more or less approximate can be arrived at, as will be seen below.

(1). *Cornish Smelting Charges.*—Your Cornish correspondent gave an illustration of the charges made by Cornish smelters in your issue of November 19, 1910, the net result of which is that, including the 3 lb. allowance per 112 lb. demanded by the smelters, that the total charge per ton of concentrate assaying, say, 65% Sn. amounts in round figures to about £11. This charge must needs vary within certain limits as the buying of the Cornish tin ores by the smelters at the ticketings is a competitive one ; but it will be found that from £10 to £11 generally covers the smelting charge made per ton of concentrate.

(2). *Continental Smelting Charge.*—As an example, I give the following price obtained in May, 1908, for an ore assaying 71·4% Sn dry assay, with the white metal at £140 2s. 8d. The Continental smelter paid M. 181·11 per 100 kilos, and as £140 2s. 8d. equals M. 280·27 per 100 kilos, this amounts to a total smelting charge of, say, £9 10s. per ton of concentrate. This is somewhat lower than Cornish charges, but as the ore was high grade and pure, probably the Cornish charges would have been approximately the same.

As another example: Another smelter quite recently charged £12 per ton returning charge for an ore testing about 67% Sn. The tin contents were paid for in full, the basis of the tin valuation being the average lowest price of either cash or three months during the week following arrival of mineral. In this case cash was paid, whereas in the previous example a three months' bill for value was given.

(3). *Australian Smelting Charges.*—The Mount Bischoff Company (Tasmania), who are doing customs smelting besides treating their own ores, I understand, make the following charge : Smelting charge, £3 per ton, and deduction of 2% from the assay to cover smelting losses. For instance, for a 72% ore they return 70% tin, or say, 14 cwts. of metal per ton of mineral. A heavier deduction is, however, made for ores under 70%. Here, again, it must be borne in mind that this low total charge of, say, under £7 per ton, with tin at £190, is on account of the great purity of the Tasmanian ores.

(4). *Straits Settlement Smelting Charges.*—As far as one can gather, the following is the charge made by the Straits Trading Company at Pulo Brani, which concern handles by far the greater portion of the ores in the Malay. The total deduction for ores testing 70% Sn is about 2½%, and proportionately more for lower grade ones : for instance, for 65 to 67·5% Sn 4% is deducted. From the weight of the dry ore and reduced assay value thus obtained, the weight of the tin is calculated, and is paid for in cash at current rates, less a returning charge of about £2 10s. to £3 per ton of ore. These figures are only approximate, and not of quite recent date ; and, perhaps, one of your readers may give us more definite particulars.

In looking at the various charges one point must not be overlooked—viz., the question of the cost of carriage of the ore, which may considerably modify the actual amount obtained by the mines."—O. J. STEINHART.—*London Mining Journal*, May 27, 1911, p. 575. (A. R.)

THE SAFETY FACTOR OF WIRE ROPE IN HOISTING, ETC.—"In the application of wire rope to the transmission of power, and for purposes of hoisting or haulage where it has to run about sheaves, it is subjected to two stresses, tensile stress due to the load, and transverse stress due to bending. In an article on "The Internal Stresses in a Wire Rope," by Walter C. Kretz, it is stated that the bending stress does not affect the ability of the rope to carry external loads : in other words, that the factor of safety of a given rope lifting a given weight is identically the same whether that rope be bent over a sheave or not, and that this has been proved by tests time and again. The explanation given is that a tension on one side of the neutral axis due to flexure means a compression on the other side, and that the sum of these stresses is nil. On the other hand, it is further stated that while bending does not affect the strength, it does affect the life of the rope by reason of the continual stressing of the outermost fibres of the wires which causes the material to deteriorate rapidly.

What is the factor of safety of a wire rope? Are we to infer that because certain tests may have shown no impairment in the tensile strength of wire ropes subjected to bending that its factor of safety is independent of the conditions governing its life? Can we say that a wire rope in use on a mine shaft, for instance, is just as safe running over a sheave half the diameter of the one that it actually does run over? When a wire is bent free of contact, the neutral plane of flexure is undoubtedly coincident with its axis, but is this true with a wire bearing against other wires as it does in a wire rope, or against the rim of a sheave?

We know that a wire rope subjected to undue bending after a time shows the effect of this in the fracturing of the wires. That this result is due to the overstraining of the wires in bending is confirmed not only by the observations of wire ropes in actual use, but by experiments made to determine the effect of bending about sheaves of various diameters on the life of wire ropes running under tension. The fractures may occur without flattening of the wires from abrasion, and are generally flush with breaks at the crown faces of the strands; that is, the points are not tapered with cupped ends like those of wires broken under direct tensile strain. Occasionally, but rarely, such fractures will occur, especially with wire ropes of the plow-steel grade, where the wires of one strand cross those of an adjoining strand. It has not been observed that breaks of this nature have been accompanied by any marked indentation of the wires, and it is probable that they are caused by some molecular disturbance produced by the friction of one wire against the other. It is difficult to account for this phenomenon in any other way. It is rarely that a core wire breaks unaccompanied by breaks in the outer wires, notwithstanding the fact that a theoretical demonstration of the stresses sustained by the individual wires shows that the proportion of load borne by the central core wires is somewhat greater than that by any of the surrounding wires. Such breaks have been observed, however, and can be readily detected by the drawing-in of the surrounding wires. It is rarely also that fractures occur in the outer wires of the strands in the interior of the rope or where these wires bear against the hemp core.

Sufficient consideration is not generally accorded to the effect of bending upon wire ropes, and it is too often the case that sheaves are used of too small diameter for satisfactory results. The strength of the sheave itself is frequently the chief consideration, its diameter apparently being of little or no moment. The rope soon goes to pieces, and it is either condemned as "rotten" (a favourite expression with many), or the natural inference is that it is too small, and a rope of larger diameter is tried, which goes to pieces even more quickly. Had a smaller rope been substituted instead, better results might have been obtained, as has been demonstrated on several occasions where such substitutions have been made. This is explained by the fact that the difference in the maximum safe stresses in such cases was considerably more than compensated for by the greater difference in the bending stresses.

When a body is placed under stress certain deformations occur in the fibres or molecular structure of the material which are proportional to the amount of the external load up to a certain point known as the limit of elasticity. Within this limit no change takes place in the molecular structure, and upon relieving the body of stress the molecules assume their original arrangement. Now, if the load is increased beyond that corresponding with the limit of elasticity, the deformations in the molecular arrangement increase at a faster rate, and upon release of the load do not at once assume their original arrangement, and are said to take a set which may be a permanent or temporary one, according to the nature of the material. If the load is still further increased, a point is reached where the molecules begin to separate, or, in other words, lose their power of coherence. This is known as the yield point. With some materials rupture may take place immediately upon reaching this point. With iron and steel, however, the ultimate stress is somewhat greater. Beyond the yield point there can be no restoration of the original molecular arrangement, and it makes little or no difference whether part or all of the fibres or molecules are subjected to such a stress; as far as the general effect is concerned, the material has been permanently weakened and should be considered unsafe for use. How far the factor of safety of a wire rope running about sheaves may be independent of the bending will depend upon the conditions under which the bending takes place.

The question is, can any material be regarded as safe that is loaded beyond its elastic limit, but within the yield point? The answer to this should take into account not only the molecular condition of the material, but also its fluidity. That a certain movement of the molecules one about the other takes place with deformations in the molecular arrangement is an accepted theory, and although the laws governing the molecular flow of solids may not as yet be thoroughly understood, we have reason to believe that certain disturbances affecting this flow have a deleterious effect on the material, and that the mobility of the flow may be affected not only by temperature, but in other ways from the manner in which the deformations under stress are produced.

In fact, the distinction between a solid, liquid, and gas, as far as the molecular flow is concerned, is only one of degree. Water, as we know, under the effect of temperature assumes all three conditions. In a gas the molecules are so sensitive to external forces that they are mobile in the highest degree, and we can feel the movement although we may not see it. In liquids the movement can be seen in the wave motion, which may be active or sluggish, according to its viscosity. We cannot see the movement in solids, but there is no doubt that such a movement actually does take place under stress.

A material that has been strained or loaded beyond its elastic limit, although a permanent set may have taken place, is not necessarily injured if the load is so applied that the molecules have time to assemble themselves into positions corresponding with the new shape, so that their coherence remains intact, and tests made under such circumstances would develop no impairment in the strength of the material. But if the loads are so applied that the molecules have not time to accommodate themselves to the changes of form, the material may be injured and become unfit for use. It is well known that iron and steel bars subjected to rapidly recurrent stresses, even though these may not be applied suddenly, will eventually break with loads considerably less than the ultimate static stress.

The theory of molecular flow above enunciated would account for this phenomenon, and may also account for the fact that mild steel or even iron has been found to give better service in the case of recurrent loading or intermittent stresses, than a higher grade of steel, the tensile strength of which is considerably greater.

The maximum safe stress of a wire rope may be considered as two-thirds of the elastic limit of the wires, which for iron and steel is about two-thirds of the ultimate stress, or, in other words, the maximum safe stress may be considered as four-ninths of the ultimate stress: a good rule under ordinary conditions is to take it at two-fifths. In any case the effect of bending should be taken into consideration in estimating the total stress. Whether this should be added to the load stress, wholly, in part, or not at all, will depend on the circumstances of the case. With iron, owing to its ability to recover from strain more quickly than steel, the maximum safe stress may be taken at a higher ratio, and it will not be amiss in most cases to consider this as three-fourths of the elastic limit of the wires, as Mr. Kretz states, or one-half of the ultimate stress.

We conclude from the foregoing that while the safety factor of a wire rope in most cases may be considered as an empirical ratio between the load and ultimate stresses, it really has no definite limitations and may vary according to the conditions under which the stresses are applied. The maximum safe stress for a wire rope on a mine shaft, for instance, where an accident might result in serious consequences, can hardly be as great as on a haulage line or derrick where the damage is not likely to be serious.

Furthermore, while bending under certain conditions may not affect the ability of a wire rope to sustain external loads, it is surely incorrect to infer from this that the safety factor of a wire rope is entirely independent of the conditions that govern its life."—WILLIAM HEWITT, M.E., *Industrial Engineering.—The Mining World*, May 20, 1911, p. 1031. (W. R. D.)

WHAT IS A MINERAL?—"The definition of the term 'mineral' is a very elastic one, and over a score of different definitions might be cited. Prof. S. Dana's definition is:—'A mineral is a body produced by the processes of inorganic nature, having a definite chemical composition and, if formed under favourable conditions, a certain characteristic molecular structure which is exhibited in its crystalline form and other physical properties.' Rutley defines a mineral as 'a homogeneous substance formed without the agency of man, and under conditions in which neither animal nor vegetable life has taken part.' Again Sir Archibald Geikie says that 'A mineral may be defined as an inorganic substance, having theoretically a definite chemical composition, and in most cases also a certain geometrical form.' Lastly, Nuttall's Dictionary gives the meaning of the word 'mineral' as 'A body destitute of organisation, and which naturally exists within the earth or its surface, a term including all inorganic substances' Now, none of these definitions would give the ordinary man much enlightenment, as they seem so contradictory, especially if they were referred to the two substances, free-tone and shale. Freestone cannot be said to have a definite chemical composition, nor in most cases has it a geometrical form, while oil-shale cannot be said to be an inorganic substance or a substance in the formation of which neither animal nor vegetable life has taken part, for it is frequently charged with organic remains. But the difficulty does not end here, for all along there have been two distinct meanings taken out of the word 'mineral'; the scientific meaning or definition, and the commercial definition, the latter according to Lord Justice Fletcher Moulton being 'any substance which can be got from within the surface of the earth which possesses a value in use apart from its mere possession of bulk and weight which makes it occupy so much of the earth's crust.'"—*Iron and Coal Trades Review*. March 17, 1911, p. 420. (A. R.)

PROPERTIES OF CONCRETE.

Proportions.					Volume in place cub. ft.	Barrels cement per cub. yard.	Cost cub. ft. broken stone concrete.	Cost cubic foot gravel concrete.	Ultimate compressive strength broken stone concrete, 1 month old, per sq. in.	Ultimate compressive strength per sq. in. of brickwork.
Cement.		Sand.		Stone.			$	$		
1	:	½	:	1	6·8	3·97	·370	·340	5,340	
1	:	1	:	2	9·7	2·79	·314	·274	4,000	
1	:	1½	:	3	12·6	2·14	·283	·237	3,200	835
1	:	2	;	4	15·6	1·73	·264	·214	2,700	
1	:	2½	:	5	19·0	1·42	·246	·195	2,300	
1	:	3	:	6	22·8	1·18	·230	·179	2,000	
1	:	3½	:	7	26·6	1·02	·219	·169	1,785	486
1	:	4	:	8	30·4	·89	·210	·160	1,625	
1	:	4½	:	9	34·2	·79	·205	·154	1,500	
1	:	5	:	10	38·0	·71	·200	·149	1,400	
1	:	5½	:	11	41·8	·65	·195	·145	1,320	347
1	:	6	:	12	45·6	·59	·192	·141	1,250	

—*Mines and Minerals*, May, 1911. (A. R.)

THE INTERNATIONAL CYANIDE TRUST.—"It is understood that the trust has obtained such general control of gas works waste in Germany and in other countries under long-term contracts as to render it practically impossible for competing concerns to be organized, and it is believed that the few not already in the trust who might produce cyanide of potassium are paid to refrain from doing so. Indeed, according to the published report of the Deutsche Gold und Silber-Scheideanstalt for 1909, a payment was made in the year covered of $121,201 'according to special agreement.'

Some years ago, before the organization of the trust, cyanide was manufactured and was being sold with at least a fair profit for from 62 to 70 marks ($14·76 to $16·66) per 220 pounds. The price has been driven up from this level to 118 to 122 marks ($28·08 to $29·04) for cyanide of potassium and 122 to 128 marks ($29·04 to $30·46) for cyanide of sodium. Both of the materials named are indispensable in modern gold and silver reduction processes, the demand being particularly great in the United States, Mexico, and South Africa. It is believed that not a pound of either material is now shipped either from the German producing factories or from the associated concerns in other countries except under conditions dictated by the trust, which produces also many other chemical products, not all of which are controlled so absolutely as cyanide.

The published report of the Deutsche Gold und Silber-Scheideanstalt, which is both a manufacturing and holding company, states that it is the owner of the stock of the three American concerns as well as of eight other companies in Europe.

For the fiscal year ended March 31, 1909, the Deutsche Gold und Silber-Scheideanstalt paid out dividends amounting to 30%, and in the succeeding year there was a distribution of 33%, or $785,000. These enormous dividends were made possible after paying the 'tantieme,' or non-producers graft, 'according to special agreement,' amounting to $121,201, and a further 'tantieme' to the board of supervising directors of $51,032. The report contains the statement that the American companies in which the parent concern was interested were obliged to reduce the scale of their operations, and could not show as good results as in previous years. This situation was considered to be merely temporary.

The exports from Germany of cyanide of potassium and sodium, for the first ten months of 1910, and for the full calendar years 1909 and 1908 were as follows:

Countries.	1910 Tons.	1909 Tons	1908 Tons
United States ..	1,098·3	1,349·2	1,394·8
Great Britain ..	117·4	147·0	148·7
Russia	199·9	140·3	217·5
South Africa	1,638·0	2,600·7	2,564·0
Japan	87·2	133·7	105·6
Dutch Indies	139·8	199·1	125·3
Mexico	1,065·6	1,118·1	19·1
Australia	110·2	216·1	77·9
All others ..	548·6	377·8	233·7
Total exports ..	5,005·0	6,282·0	4,886·6
Value	$1,548,190	$2,093,210	$1,628,150

—*Mines and Minerals*, April, 1911, p. 568. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Albe, E. E. Fournier d'. Contemporary Chemistry, a Survey of the Present State, Methods, and Tendencies of Chemical Science. Cr 8vo, pp. 196. *Constable.* Net 4s.

Geological Survey. United Kingdom. General Memoir. Summary of Progress of the Geological Survey of Great Britain and the Museum of Practical Geology for 1910. *T. F. Unwin.* 1s.

Green, A. M. Pumping Machinery. 8vo. *Chapman & Hall.* Net 17s.

Hazard, D. L. Directions for Magnetic Measurements. Engravings. 8vo, 3d, pp. 131. *Wesley.* Net 4s.

Henkel, F. W. Weather Science: an Elementary Introduction to Meteorology. Cr 8vo, pp. 336. *Unwin.* Net 6s.

Hobart, H. M. The Design of Static Transformers. 8vo, pp. 190. *Constable.* Net 6s.

Hurst, G. H. Lubricating Oils, Fats, and Greases: Their Origin, Preparation, Properties, Uses and Analysis. 3rd Edition Revised. 8vo, pp. 406. *Scott, Greenwood.* Net 10s. 6d.

Journal of the Institute of Metals, The. Vol 5, 1911. Edited by G. Shaw Scott. 8vo, pp. 423. *Institute of Metals.*

Kilner, W. J. The Human Atmosphere: or the Aura Made Visible by the Aid of Chemical Screen. Illustrated. 8vo, pp. 329. (Diagnosis Sheets Separate and Screens in Box.) *Rebman.* Net 30s.

Leechman, D. Systems of Electric Ignition for Motor Cars. 2nd Edition entirely revised and rewritten. Cr 8vo, pp. 192. *Car.* Illustrated. Net 1s. 6d.

McDowall, S. A. A Laboratory Note-Book of Physics. 2 Pictures. 4to, pp. 174, interleaved. *Dent.* Each net 2s. 6d.

Miers, H. A. The Growth of a Crystal. Being the 18th Robert Boyle Lecture delivered before the Oxford University Junior Scientific Club on May 20, 1911. Cr 8vo. *Frowde.* Each net 1s.

Oldham, F. M. Elementary Quantitative Analysis. Cr 8vo, pp. 98. *Mills & Boon.* 1s. 6d.

Smith, C. C. Economy in Sewage Disposal, together with a History and Description of the Sutton (Surrey) Sewage Works. 8vo, pp. 32. *Sanitary Publishing Co.* Net 1s.

Smithsonian Physical Tables. 5th Revised Edition. Prepared by A. E. Fowle. (Smithsonian Miscellaneous Collections. Vol 58, No. 1) New Edition. 8vo, pp. 318. *W. Wesley.* Net 8s. 6d.

Thompson, M. de Kay. Applied Electro-Chemistry. 8vo. *Macmillan.* Net 9s.

Tisdale, C. W. Walker. Milk Testing. A Simple Practical Handbook for Dairy Farmers, etc. Revised and Enlarged. 12mo, pp. 86. *Dairy World Office.* Net 1s.

Tookey, W. A. How to Manage a Suction Gas Producer. A Practical Handbook for Engineers and Attendants. 12mo, pp. 90. *P. Marshall.* Net 1s.

Twelvetrees, W. Noble. The Practical Design of Reinforced Concrete, Beams and Columns. Cr 8vo, pp. 178. *Whittaker.* Net 6s.

Wood R. Physical Optics. New and Revised Edition. 8vo. *Macmillan.* Net 22s.

Whyte, A. G., Electricity in Locomotion. Roy. 16mo. pp. 152. *Cambridge University Press.* Net 1s.

Abstracts of Patent Applications.

(C.) 216/11. Oscar Clarence Beach (1), Arthur Judson Hodge (2). Improvements in disc pulverisers. 13.4.11.

This invention relates to disc pulverisers of the type where the material is ground between two abrading discs, the surfaces of which are pressed together, while at the same time one of the discs revolves. This action tends towards the formation of annular grooves, which diminish the effectiveness of the surfaces. To overcome this the present invention provides for the axis of the revolving disc to be parallel to, but at a distance from, the axis of the fixed disc and while the moving disc revolves on its spindle, the axis of the spindle and revolving disc is made to move slowly in an orbit round the centre of the fixed disc. To obtain the two motions of the revolving disc, there is a sleeve (the outer circumference of which is concentric with the fixed disc) which is rotated slowly by means of an epicyclic train of gearing operated by the spindle of the revolving disc. Through the sleeve there is a hole to receive the spindle of the the disc, the axis of the hole being parallel to, but at some distance from, the axis of the sleeve.

The grinding motion between the surfaces of the discs is obtained by one of the discs being revolved on its own axis, and the displacement of the revolving axis as a whole is obtained by the rotation of the sleeve through which the driving spindle pass eccentrically.

(C.) 237/11. Charles Washington Merrill. Process for precipitating and recovering valuable materials from solutions, and particularly metals from hydro-metallurgical solutions. 26.4.11.

This relates to improvements in the process of zinc dust precipitation and to apparatus for completely excluding atmospheric oxygen both before and while in the filter press. The idea of subjecting the precipitant to mechanical attrition for cleansing purposes is also claimed.

(C.) 238/11. Charles Washington Merrill. Process for treating a semi-solid material in the containers of filter presses, and for removing the same therefrom. 26.4.11.

This refers to improvements in apparatus for removing solids from filter press frames by sluicing out and to arrangements for treating the same *in situ* with any desired fluids.

(C.) 256/11. Richard Benard Rinne. Improvements in apparatus for automatically signalling to the driver of a hoisting or hauling engine the fact that a skip, cage or like vehicle has left the track or guiding means. 6.5.11.

This invention relates to a method of electrically signalling to a winding engine driver the fact of his skip having left the rails. Boards are placed immediately above and below each divider, and contacts are fixed to board and divider in such a manner as to be closed by the impact of the derailed skip. The carrying away of a divider with its contacts is provided for by providing at each divider spring operated contacts.

(C.) 270/11. Wilbur Alson Hendryx. Improvements in and relating to apparatus for separating solids and liquids. 12.5.11.

This patent refers to an apparatus comprising a conical vat, a delivery tube connected to its lower portion containing the means of forcing the solids accumulated at the bottom of the vat through it, and the means of regulating the pressure and discharge of the material in the tube, the purpose of the invention being to separate solids from liquids.

(C.) 290/11. John Cuningham Cuningham. Improvements in apparatus for cooling water by evaporation. 23.5.11.

This application is in respect of an arrangement of water trays and air deflectors to be used in cooling tower construction. The object of the device is to admit air up between the water trays so as to strike the air deflectors and thus come into more intimate contact with the descending water.

(C.) 301/11. Charles Eber Baker. Improvements in the treatment of ores. 26.5.11.

This application describes a method for the treatment of zinc, lead or copper ores consisting in subjecting the crushed partially roasted, or, mixed sulphide and oxide, ore to the action in a moderately heated barrel of a mixture of halogen acid, oxygen and chlorine, whereby metallic chlorides are formed.

(C.) 310/11. Frank Franz (1), Eugene R. Day (2). Improvements in ore concentrators. 2.6.11.

This patent refers to a concentrating appliance or table somewhat on the lines of the Wilfley, Ferraris and other concentrating tables. The main features are the riffles which extend in a direction parallel to the infeed side of the table, a closure wall at the concentrate discharge end of the table which extends at an obtuse angle with respect to the infeed side and at an acute angle with respect to the discharge side of the table, a smooth concentrate discharge end having a covering that laps up against such end closure wall, precipitate slots extending at right angles to the concentrate discharge end closure wall, a trap at the infeed side of the table and sundry other means for regulating the delivery of the concentrates and the motion of the table.

(C.) 403/10. Reuben Jones. Improvements in stamp tappets. 17.8.10.

This invention has reference to stamp tappets of the kind which are retained in position on the stamp stem by means of cotters passing horizontally through the body of the tappet and acting upon gibs to force the latter against stem.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 402/11. Martin McHale. Improvements in rock drill making and sharpening machines. 17.7.11.

(C.) 403/11. Hugo Crob. Improvements in the regulation of electric tensions. 17.7.11.

(P.) 404/11. James Lester Clark. Improvements relating to sand tanks. 18.7.11.

(P.) 406/11. Lars Pedersen. Improvements in respirators. 19.7.11.

(P.) 407/11. Siegfried Reinheimer. Rotary rock drill. 19.7.11.

(P.) 408/11. George Anderson Robertson. Improvements in mills or machines for crushing ore and other materials. 21.7.11.

(C.) 409/11. Edwin Henry Messiter (1), Weigher Patent Company (2). Improvements in and relating to continuous weighing mechanism. 21.7.11.

(C.) 410/11. Edwin Henry Messiter (1), Weigher Patent Company (2). Improvements in and relating to continuous weighing mechanism. 21.7.11.

(P.) 413/11. Peter Norman Nissen. Improvements in caps for cam shaft bearings. 22.7.11.

(C.) 414/11. Ivan Werlin. Process for manufacturing a refractory and abrasive product 22.7.11.

(P.) 415/11. Petrus Jacobus Pienaar. Improved water wheel automatically driven by a centrifugal pump and vice versa. 25.7.11.

(P.) 417/11. Francis Ernest Dunnett. Improvements in paint and compositions of a similar nature to paint. 25.7.11.

(P.) 419/11. Charles Henry Wilkins. Improvements in supports for pneumatic hammer tools. 25.7.11.

(P.) 420/11. Arthur Marinus Alexander Struben. A lock nut for locking, nuts studs, screws and the like 26.7.11.

(C.) 423/11. John Hines. Improvements in apparatus for heating, drying and otherwise treating stone and other material for use upon roads and like surfaces. 28.7.11.

(C.) 424/11. Dr. Fritz Rothe. Process for the production of disc-like electric flaming arcs for carrying out gas reaction. 28.7.11.

(C.) 425/11. Robert Swan Crawford (1), James Scott (2). Improvements in tools or machines operated by a reciprocating element and in means for drying them electrically. 28.7.11.

(P.) 427/11. James Grant Gibson. Improvements in machinery for making concrete blocks. 1.8.11.

(P.) 428/11. Charles Arthur Coles (1), Johannes Wilhelmus Bykelenboom (2). Dust collector for rock drills. 1.8.11.

(C.) 429/11. Otto Stuhaug Jonnesen. Improved shapes of drill bits. 2.8.11.

(P.) 430/11. Arthur Marinus Alexander Struben. An improved pipe joint. 2.8.11.

(P.) 431/11. Edmund Paul Millington Jerrard. Improvements in and relating to synchronous reproduction of sound and light wave effects and apparatus therefor. 2.8.11.

(C.) 432/11. Mose Wilbuschewitsch. Process and apparatus for converting fatty acids, fats, oils, fish oils, and waxes into like bodies of higher melting point. 4.8.11.

(C.) 433/11. Jabez Muskett. Improvements in hose for the transmission of fluids. 4.8.11.

(C.) 434/11. Guy St. Farbe Sladen Watkins. An improved method and means for cleaning the combustion chambers of internal combustion engines. 4.8.11.

(P.) 435/11. James Ferguson (1), Edward James Rule (2). Improvements in chucks or tool holders for rock drilling machines. 4.8.11.

(P.) 436/11. James Ferguson (1), Edward James Rule (2). Improvements in rock drilling machines. 4.8.11.

(P.) 438/11. John Frederick van der Velde. Improvements in machine and hand rock drills. 5.8.11.

(P.) 439/11. Joseph Crosfield & Sons, Ltd. (1), Karl Emil Markel (2), Wilhelm Normann (3). Improvements in the treatment of unsaturated fatty acids and their glycerides for the production of saturated fatty acids or glycerides, and in the preparation of catalysts for use in such treatment.

(P.) 443/11. James Albert Grayson Harrison. An adjustment for regulating the discharge of fluids over crest weirs and the like. 8.8.11.

(P.) 445/11. Charles Edward Bennington. Improved ambulance dividing sheet. 8.8.11.

(P.) 446/11. Thomas Henry Chad (1), George Daniel Hook (2). A new and improved method of securing rails and the like to sleepers or any other bearer. 10.8.11.

(P.) 448/11. James Watson (1), John William Petrie (2). Improvements appertaining to depth indicators for hoists and the like. 11.8.11.

(P.) 451/11. Eugene Louis. Improvements in explosives. 11.8.11.

(C.) 452/11. James Henry Alling. Process for recovering metals from ores. 11.8.11.

(C.) 453/11. Dora Kathleen Stobie. Respirator. 11.8.11.

(P.) 455/11. Randall Birch Canning. An improved apparatus for silencing intake or egress orifices of air or gas compressing machinery. 12.8.11.

(C.) 456/11. Joseph Albert Hill. Improvements in and relating to the furnaces, furnace fire-bars, and furnace fire-bridges of steam boiler furnaces and like furnaces. 12.8.11.

(P.) 457/11. John Henry Skinner. Invention for reinforcing concrete block walls. 14.8.11.

(P.) 458/11. George Henry Seedon. Improvements in and relating to the form and construction of positive seated valves for rock drills and the like. 15.8.11.

(P.) 459/11. James Wilshir (1), George Twomey (2). Improvements in lubricators. 15.8.11.

(P.) 460/11. Basil Vivian Blundun. Improvements in sieving apparatus. 15.8.11.

(P.) 461/11. Robert Jennings. Improvements in means or devices for repairing the inner tubes of pneumatic tyres, hose pipes and the like. 15.8.11.

(P.) 462/11. George Buchanan. Improvements in the tools and tool rests of lathes used for cutting zinc. 16.8.11.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

ANGUS, W. H., *to* P. O. Box 463, Germiston.

ARDERNE, R. A. S.. *to* Post Office, Pilgrims Rest.

BIDLAKE, A. E., *l/o* Boksburg; c/o Post Office, Bulawayo.

CAMERON, L. D., *to* Darymusu (W. Coast), Dev. Co., Ltd., Naragut, Northern Nigeria.

COCKBURN, P. A., *l/o* Francistown; Bwana M'Kubwa Copper Mining Co., Ltd., N. W. Rhodesia.

DUDGEON, A. F., *l/o* Penhalonga: Associated Mines of Rhodesia, Ltd., P. O. Box 123, Salisbury.

FRY, A. T., *l/o* Wiluna; Kapunda, Western Australia.

GRIFFIN, N. H., *l/o* England; The Chartered and General Exploration and Finance Co., P. O. Box 592, Bulawayo, Rhodesia.

HARRISON, M. J., *l/o* Germiston; Modderfontein B. G. M. Co., Ltd., Benoni.

HOGENRAAD, G. B., *to* Gold Mining Co., Salida, Padang, Sumatra.

HOWE, B., *l/o* Sumatra; 30, Bramham Gardens, South Kensington, London.

MACAULAY, M., *to* Eldorado Mine, Rhodesia.

MANCE, J. C., *l/o* Hong Kong; c/o Mare Dandolo, Esq., Rue Jules Ferry, Haiphong, Tonkin, Indo China.

NICHOLLS, J. C., *l/o* Dutch Flat: Mills Building, San Francisco, Cal., U.S.A.

NIXON, G. H., *l/o* Vaalhoek District; Post Office, Pilgrims Rest.

PENCIER, H. P. DE, *to* 59, Royal Trust Building, P. O. Box 763, Montreal, Canada.

POPHAM, J. L., *l/o* Bulawayo; Northern Copper Co., Ltd., Broken Hill, N. W. Rhodesia.

RICHARDSON, W. P., *l/o* Swaziland; Horo Mine, Louws Creek, near Barberton.

ROSS, A. W., *l/o* Wellington; Mitchell's Creek Gold Recovery Co., Bodangora, New South Wales.

ROTHKUGEL, M., *to* New Kleinfontein G. M., Benoni.

SIMMONS, F. L., *to* P. O. Box 78, Cleveland.

SIMPSON, F. L., *l/o* Rhodesia; c/o W. Thow, Esq., Warraine, Sydney, N.S. Wales, Australia.

SPENCE, A. G. B., *l/o* Denver; Vogelstruis Estates and G. M. Co., P. O. Box 116, Roodepoort.

STOKES, R., *l/o* Johannesburg; c/o 5, Summerland Mansions, Muswell Hill, London.

TAYLOR, M., *to* P. O. Box 46, Roodepoort.

WILLIAMSON, REG., *to* Great Boulder No. 1, Ltd., Mount Magnet, Western Australia.

WILMOTH, L. J., *l/o* Australia; c/o Institute of Mining and Metallurgy, Salisbury House, London, E.C.

THE JOURNAL
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Vol. XII. SEPTEMBER, 1911. No. 3.

Proceedings
AT
Ordinary General Meeting, September 16, 1911.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, September 16th, Mr. W. R. Dowling (Vice-President) in the chair. There were also present :—

46 Members: Messrs. H. A. White, M. H. Coombe, K. L. Graham, J. Gray, Prof. G. H. Stanley, F. W. Watson, Prof. J. A. Wilkinson, A. F. Crosse, J. R Williams, A. McA. Johnston, Dr. J. Moir (Members of Council), W. Abel, W. Beaver, F. W. Cindel, C. Crichton, A. L. Edwards, L. L. Ellis, R. Gascoyne, B. C. Gullachsen, J. H. D. James, A. M. Kratz, G. A. Lawson, L. Marks, J. T. Milligan, P. T. Morrisby, W. J. North, A. C. L. Olsen, E. A. Osterloh, F. S. Pearce, J. Philpott, J. B. Polglase, B. E. Pollard, J. F. Pyles, E. Roberts, H. Ross, K. Sartorius, S. Shlom, G. H. Smith, B. C. T. Solly, S. H. Steels, A. Thomas, Prof. G. R. Thompson, W. E. Thorpe, C. Toombs, and J. Watson.

17 Associates and Students: Messrs. A. R. Adams, G. Bulman, W. E. Burrows, J. Cronin, I. B. Dick, B. Ellis, A. King, E. Lee, G. F. Mathews, F. J. Pooler, R. A. Porter, H. Rusden, P. Scatterty, H. Stadler, H. Ward, A. Willcox and S. A. Woolf.

9 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as published in the August *Journal*, were confirmed.

NEW MEMBERS.

Messrs. K. L. Graham and A. McA. Johnston were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been duly elected as follows :—

James, John Henry Davy, Ginsberg G. M. Co., Ltd., P. O. Box 16, Knights. Assayer.

Neilson, James, May Consolidated G. M. Co., Ltd., P. O. Box 50, Germiston. Cyanide Manager.

Reid, Mathew Henry, Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Cyanide Manager.

Schlesinger, Bruno, Lancaster West G. M. Co., Ltd., P. O. Box 360, Krugersdorp. Mining Engineer.

The Secretary notified that since the las meeting of the Society the following Associates and Students had been admitted by the Council :—

As Associates—

Archibald, Max Stanfield Eaton, Knights Deep, Ltd., P. O. Box 143, Germiston. Shift Boss. (*Transfer from Students' Roll.*)

Ball, Harry Standish, M.Sc. (McGill), c/o Standard Bank, Ltd., 10, Clements Lane, London, E.C. (*Transfer from Students' Roll.*)

Pickburn, William Henry, Rand Club, Johannesburg. Miner.

As Student.—

Green, Morris, S.A. School of Mines and Technology, P. O. Box 556, Johannesburg. Student.

General Business.

A letter was read from the President expressing his regret that he was unable to preside at the meeting, as he had had to visit Lydenburg, and apologising to the Council and members present for his unavoidable absence.

The Chairman: I am sure we all regret Mr. Saner's absence, as I know how he had looked forward to being present at this meeting.

NOTE ON THE USE OF CERTAIN CYANICIDES IN THE ESTIMATION OF UNDISSOLVED GOLD.

Mr. H. A. White (*Vice-President*): The lowest possible residue value in the treatment of slime by cyanide solution can only be obtained when all the accessible gold is in solution before the first settlement commences (or before passing to filters where used), and it is merely wasteful of power and cyanide to continue the treatment after such solution is complete. As pointed out in a contribution to the discussion of Mr. Whitby's paper on assaying,[*] there is a difficulty in securing an accurate estimate of the undissolved gold in such cases, especially in the earlier stages of treatment. This difficulty is, of course, connected with the fact that during the handling of the pulp sample solution of gold may be proceeding under much greater advantages, owing to increased relative surface exposure, than is the case with the bulk under treatment.

Experiments have shown that drowning the sample in a large excess of water, in the endeavour to stop the action of the cyanide by excessive dilution, may fail owing to surface concentration or adsorption effects, although no doubt the greater part of the excess gold dissolved goes into solution during this first wash. The use of freshly boiled water, as then recommended, would naturally be a palliative because of the relative scarcity of oxygen required. It is therefore probable that in many cases a "washed residue" result merely represents some point between the actual undissolved gold sent to the dam or the dump and the inaccessible gold under the conditions of treatment. Again, when samples are taken to determine the length of treatment required for solution of the available gold, the inaccuracy involved in their examination may frequently be a source of loss of possible profit. The assay of the residue for total gold without the use of cuprous salts (Whitby's method) would help to conceal such losses.

While investigating the question of the destruction of cyanide for the purpose of "sand filling,"[†] the main track was diverged from slightly, with the object of determining whether help could not be obtained in this direction, and the suggestive facts were then published that gold, in cyanide solution, was not precipitated by excess of alkaline permanganate, by formaldehyde, by alkaline picrate, or even by ferrous sulphate. (In the latter case it has been stated that some gold is precipitated, but I found that this error was caused by difficulty in washing the voluminous precipitate, which may even have been entirely neglected). The observation that the addition of excess of alkaline permanganate to 0·10% KCN at once and completely destroys its power of dissolving gold leaf was also published at the same time, and later the fact that bleaching powder in alkaline solution has both these favourable properties. Formaldehyde and alkaline picrates failed in immediately inhibiting the solvent action, and are therefore of no use for our purpose. Though the necessary observations were thus made public, attention was not called to their bearing on the purpose we are now considering, though I have ever since then applied them in this direction, after a few preliminary experiments which proved the necessity and the security of the new method.

The following is the method used in examining the course of treatment of slime by cyanide solution and the necessary alteration in other cases will be obvious. In the first place a bucket, half filled with gold-free water, has placed in it sufficient saturated solution of permanganate of potassium to leave a decided pink colour after the addition of the slime sample.

At a definite time during treatment the sample of pulp to be tested is taken by means of a tube with flap valve at the bottom, or by other suitable means, and allowed to run into the prepared bucket under the surface of the solution and with as little exposure to the air as possible. The whole operation is conducted as rapidly as may be. The bucket is then filled with water and allowed to settle. The solution is then poured off and discarded. The operation of filling with water, well mixing and allowing to settle is repeated four times, in each case with the addition of lime water to hasten settlement. Finally the sample is dried for assay. In performing the assay it is especially necessary in certain cases to secure a large button of lead: "washing" with litharge and reducer should not be neglected and fusion should be completed at a high temperature.

Mr. Andrew F. Crosse (*Past-President*): I think Mr. H. A. White deserves our thanks for proposing a method which will help to make accurate estimations of the gold in a slime or sand residue at a given moment during treatment. The treatment of Rand sand and slime is, metallurgically speaking, very simple, but in treating slimes from other parts of the Transvaal. I have come across various cases in which the gold is dissolved but retained with great avidity in the slime; such ores contain ferric-hydrate in various forms. I shall certainly try Mr. White's method.

THE ASSAY OF GOLD-BEARING CYANIDE SOLUTIONS BY ELECTROLYSIS.

By C. Crichton, F.C.S.

Some four years ago Mr. Erskine, chief metallurgist to the Kleinfontein Group, had his attention drawn to an article in an Australian journal, giving a description of an apparatus for the assay of gold-bearing solutions by electrolysis, and as this method, as opposed to the usual precipitation or evaporation processes, seemed to indicate a great saving in labour and time expended, he decided to give it a trial.

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[*] See this *Journal*, Vol. VI., May, 1906, p. 343.

[†] See this *Journal*, Vol. X., pp. 444, 445.

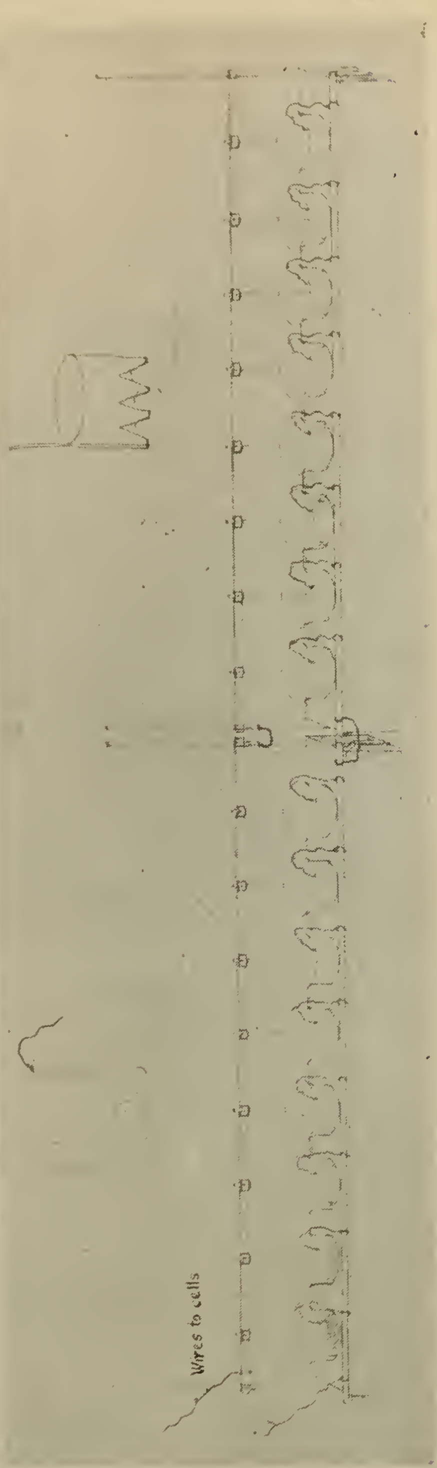
The results obtained from the first experimental run were in every way satisfactory, and during the four years in which, at the Kleinfontein Group Central Administration Assay Office, this process has been in constant use, no inaccuracies in the results obtained have been detected. From time to time check assays of the solutions have been made by means of evaporation with litharge, the values shown by the two methods being identical.

The estimation of metals by means of electrolytic deposition is in general use in most laboratories, the only difference between the determination of gold and, say, nickel or copper, by this means being that in the latter case the increase in weight on a platinum cathode is noted, and in the former a lead cathode is used and the gold obtained by cupellation. The apparatus at present in use at the above assay office consists of four oblong frames, 2 ft. 10 in. × 3 in. × 6 in., connected in parallel, and each holding eight beakers. The frames or boxes are provided with two copper rods, as seen in the sketch.

Anode Connection Rod.—The anodes consist of ordinary $\frac{3}{16}$ in. arc lamp carbons, and are held in position in the centre of each beaker by means of clamps fitted to a horizontal copper bar which runs parallel to and 6 in. above the top of the box. By means of a slot and thumb screw the anode connection rod is attached to two uprights fixed at either end of the box, so that each section of eight carbons may be either lifted clear of the beakers or lowered, as required, in one movement.

Cathode Connection Rod.—This runs along the front side of the frame, slightly above the top, and about 2 in. from it; at suitable intervals along the rod are soldered eight single flexible insulated wires, forming a connection for the lead cathodes.

Description of the Cathodes.—The cathodes are made from ordinary assay lead foil, a suitable length being 9 in., and as the foil is usually obtained in strips 36 in. long, a good quantity of the necessary lengths can be obtained in a short time by cutting the strips of foil into four equal portions. About a dozen of the lengths are placed together and Λ shaped pieces cut out from along the edge intended for the bottom of the cathode, this is to allow for the better circulation of the ions through the solution. Arrangement has to be made for connecting up with the flexible wire from the cathode rod, for this purpose a strip about $\frac{1}{4}$ in. broad is all but severed from one end of the foil, and is folded over, forming a terminal. The two ends of the lead are now brought together and connected by folding the edges; to ensure a smooth surface and circular shape a glass bottle having a slightly smaller diameter than the inside of the beakers in use, will be found to be convenient for this operation.



Very little time is occupied in making the cathodes: a native can, during a very few hours, make a sufficient number for 200 or 300 assays.

The necessary current for the deposition of the gold in solution on to the cathode is obtained from three 2-volt accumulator cells, which being connected in series give a terminal pressure of just over six volts. The current varies, of course, with the resistance offered by the solution through which it passes, *i.e.*, the stronger the solution is in KCN the greater will be the amperage. For example, a solution having a strength of ·3% KCN passes 0·1 of an ampere, and a slime solution (0·02% KCN) will take a current of about 0·04 ampere. The accumulators are charged from a direct current lighting circuit through a suitable lamp resistance, and connection can be effected between the lighting circuit and accumulators, or accumulators and electrolytic apparatus, as desired, by means of two-way switches.

Little remains to be said except that perhaps a few details regarding the *modus operandi* may be of interest. The usual number of solutions assayed each day is twenty-two, which are sent up in marked bottles from the cyanide works at 6 a.m. 10 A.T. portions are at once measured out from each bottle and placed in the beakers belonging to the apparatus, which are prepared the previous evening, so that as soon as connection is made no further attention is required. The time required for the complete deposition of the gold is four hours, after which period the carbons are removed clear of the beakers, the current is switched off and the lead cathodes disconnected and removed to a hot plate to dry. When dry these are folded into a small compass and cupelled with a little silver, parted and weighed, the values being reported to the cyanide works manager by 11 a.m. Although from start to finish this process occupies about five hours, only a few minutes are expended in actual personal attention, and the measuring out of the quantities of solutions can be done either by the cyanide works shiftsman, or as in our case by the reduction works sampler. I might mention that if 20 A.T. of solution are required for use instead of 10 A.T., the same time only is necessary for the complete deposition of the gold on to the cathodes.

Precautions should be taken against having the carbon anodes in contact with gold-bearing solution in the absence of any current passing through: negligence in this particular results in gold being precipitated on the anode.

In the event of a solution offering too great a resistance to the current, the addition of a small quantity of KCN will remedy this, and accelerate the deposition of gold.

It has also been found advantageous to add a little ammonia to solutions which deposit salts.

I regret that I can not give the name of the author to whose article I have already referred, neither have I the name of the journal in which it appeared, and unfortunately at the moment Mr. Erskine, who could probably supply the information, is away on leave.

Mr. A. McArthur Johnston (*Past-President*): I have great pleasure in asking you to accord a hearty vote of thanks to Mr. Crichton for bringing this method of assaying gold bearing solutions before us. I think there is no assay which lends itself to so many diverse methods as this one, and sometimes one would wish—I know some prospective mine managers do—that a standard method were adopted. Mr. Erskine has, to my knowledge, been working this method for some considerable time now, and it has been only his innate modesty which has prevented him laying particulars before us.

The method is itself attractive to some assayers as being particularly clean and requiring practically no attention whilst the action is going on, but in other offices the time necessary for the precipitation of the gold is not always available. In these cases the precipitation method is adopted. I feel certain that assayers will welcome the particulars laid before us this evening by Mr. Crichton, and I would therefore ask you to show your appreciation by passing him a vote of thanks.

Mr. Andrew F. Crosse (*Past-President*): In supporting the motion I only desire to say that I wish the author had given us in his paper a few comparative results between the method described and the other methods which are used on the Rand.

THE TESTING OF TRANSVAAL COALS.

(*Read at June Meeting, 1911.*)

By M. Thornton Murray, M.Sc. (*Member*).

DISCUSSION

Dr. J. Moir (*Past-President*): Having already expressed in general terms my high opinion of this paper, I hope the author will appreciate the situation if I now say that more good will be done if it is submitted to criticism than if it were to get nothing but praise. I thoroughly believe in giving authors something tangible to reply to!

I shall therefore deal in turn with the author's headings.

A. Sampling. — Prof. Wilkinson has dealt adequately with this, so I will not enlarge on it. As regards the analyst's sample, I have no comment to make: but as regards bulk sampling, I wish to amplify what I said in June. The sampling of large quantities of any substance (like coal) which cannot be crushed to distribute the waste evenly over the mass, is so difficult and unsatisfactory that the errors introduced thereby far surpass any that are likely to arise in the subsequent analysis. In this case I very much doubt whether it is worth while going to any great refinement of analysis *of particular samples*, when such particular samples cannot be made exactly representative of the mass to be tested. It is different in the case of gold ores, when the process of sampling is repeated a number of times and the assays averaged to diminish the sampling error; but in the case of coal the sampling seems to be done once only, and any averaging is merely averaging of the bomb tests of the *same* specimen. I should like therefore to challenge one of our local "crack" samplers to put these views to the test by undertaking to sample, say, a 100 ton lot of coal thrice in succession to see if he can get three samples differing only by say $\frac{1}{2}\%$. Please remember I am not casting any reflections on samplers, only saying how difficult the problem is.

B. Moisture.—I understand it is customary to do the bomb tests on samples which have been merely air-dried. This is quite indefensible with a hygroscopic substance like fine-crushed coal. It is all very well of Mr. Murray to praise the climate, but it must be remembered that when it does rain here, it "rains cats and dogs," and an analysis done after rain may be 2% lower than one done on the same sample in dry weather. If the object of the author's paper is to obtain uniformity of analytical results, even to $\frac{1}{10}\%$, then uniformity of drying is of more importance than any standardisation of the bomb process. I suggest as a tentative measure that the 60 mesh sample should be dried for two hours *in a thin layer* at 60° to 70° C., and I am sure that the current beliefs on oxidation of coal and evolution of volatile matter at such temperatures, are sheer superstitions.

C. Ash.—Why does the author suggest allowances for FeS_2 and $CaCO_3$? Surely this correction applies to the volatile matter, and not to the ash, since these substances burn in the same way on the large scale as in the proximate analysis. The case of the combustion in the bomb is of course different, but that is not the point at issue.

D. Ultimate Analysis.—I wish to call the attention of members to a point I discovered in 1903, namely, that there is a remarkable uniformity of composition of the organic part of local coals. This composition in fact only varies by 1% or 2% in any particular coalfield round about the average for that coalfield, although the figure (or mean) for different coalfields varies a good deal. The consequence of this fact is that the calorific value of any coal can be approximately calculated from its ash, if we know what coalfield it comes from. The mathematical expression of this fact is: Calorific value $= \dfrac{100 - A}{K}$, where K is constant for each coalfield, generally about 7, and A is the percentage of ash. I have not come across a greater error than 2% in this simple determination. At the time I discovered this, I was unfortunately not at liberty to publish it, nor even yet have the data on which it was founded been published; but I hope that some of my colleagues, who have, I know, got plenty of new data on the subject, will look into the matter and let us know if the rule still holds good. The scientific explanation of it must be that the organic substance in coal is practically a chemical compound with constant composition. For example, the analysis given by the author corresponds to $C_{50}H_{39}O_3N$, which on heating breaks down into, say,

$$40C + 3CO + 5CH_4 + C_2H_4 + 6H_2 + NH_3;$$

which supposition agrees closely with the "fixed carbon" found and the usual composition of the gas.

E. Criticisms.—Naturally the question of the radiation-factor is the point most open to remark. In the first place, we notice that in the particular experiment given by the author, the temperature rose during the initial period; showing that not only was the instrument not sufficiently in equilibrium with its surroundings, but that it must have been hotter than the water. This radiation question is so complicated that the introduction of a factor due to the start being made too soon is most undesirable in research work. If the author in his enthusiasm must handle and fondle his bomb so closely, he should really have given it time to cool down after the contact! I admit that this might be a "counsel of perfection" in ordinary technical work, but I take it that the author's object was to investigate the radiation correction, and if so, the fewer complications the better.

Might I make the suggestion that light might be thrown on the true correction by burning, say, 1 gm. and then 2 gm. of the same coal in the same bomb? I make this suggestion because I confess I cannot follow the Regnault-Pfaundler formula—partly, I suppose, because it has evidently suffered in the printing in the *Journal*. This formula cannot, as the author says, take account of evaporation, else it would contain a factor involving the latent heat of steam; and secondly, if compared with the author's experimental curve, it obviously gives too small a correction. Going on first principles, it is evident that (as all the heat is suddenly generated in the bomb) the loss by radiation will be got by supposing the final steady rate of loss continued back from the maximum for $n-1$ intervals (n being the number between firing and the maximum). Applying this graphically to the author's diagram on p. 606, I find the top reading without radiation would have been 2·344. On deducting the quantity 0·013 (due to the lack of thermal equilibrium at the beginning), the corrected rise of temperature comes out at 2·331 instead of the author's 2·318. Another source of confusion in this part of the paper is that the author uses the symbol θ_1, in two different senses: also that he makes $N = 6$ throughout the calculations whereas in my opinion it ought to be 5.

In my opinion the sulphur correction can be neglected altogether, because the combustion of the iron in pyrite probably only goes to FeO (namely, $FeSiO_3$) in the bomb, against Fe_2O_3 in the furnace, and this loss of heat (to say nothing of the fusion of the ash) will mostly cancel the heat due to burning the sulphur to SO_3. Nevertheless, the author might investigate the point by "bombing" some pyrite, both by itself, and also with silica, if he can succeed in igniting it with a hot wire.

What seems a more serious error—unless I misunderstand him — is the modification of Mahler's formula which he proposes on p. 607. The author makes $O + N = 100 - C - H$, which I cannot follow. In a decent coal $(O + N)$ is probably not more than 8, whereas $100 - C - H$ may easily be 20. If I am right in this criticism then Mahler's formula is not any better than the others as regards agreeing with the experiments, though, as I have indicated already, there are reasons for thinking that the author's experimental result is too low, and should be in reality nearer the theory.

As regards the thermometer, it is quite unnecessary in technical practice (though perhaps desirable in a research) to read to 0·001°, an amount only $\frac{1}{2000}$ of the whole. The other experimental errors in the bomb-determination are in the region of $\frac{1}{2}\%$, so in technical practice the temperature need not be read to less than 0·005° or even 0·01°. I do not know that the Beckmann instrument is so suitable as the author thinks. It is intended for constant temperatures, and when used for quickly-varying temperatures, it shows a decided "lag" owing to its size and thickness.

As regards the "calibration" of the bomb I wish to put a question. Has the author allowed for iron-wire, sulphur and nitrogen, in calibrating the bomb? If so, then in the coal determinations only a proportion of these corrections should be applied, else they are being counted in twice to some extent.

These criticisms, it must be noted, affect only a very small portion of the wide field which the author has taken up, and it is only right to say that I wholly agree with the rest of his paper. In reality too, I have been criticising myself, as I found when the paper was submitted to me in my editorial capacity, that I knew very little about the subject—so the author can take the credit of having at least made one Johannesburger think!

I should like to sum up my remarks by suggesting a scheme *for technical practice* for the criticism of the author and other members:

I assume that satisfactory sampling from bulk has been accomplished.

(*a*) Dry all samples, say, at 60 mesh for two hours at 60°—70° in a thin layer.

(*b*) Get the bomb and water in complete and permanent equilibrium, except for the minute cooling effect due to evaporation: this cannot be done in a room with an iron roof, for example.

(*c*) Experiment as laid down by the author; record the readings every minute.

(*d*) Calculate the radiation loss by the following approximation: find the *final* rate of steady fall from the last five readings (*i.e.*, four intervals) (at least seven after the maximum should be taken). Multiply this rate by the number of intervals between firing and the maximum (*i.e.*, this number is one less than the number of readings). For example, in the author's experiment the final rate is 0·0075 and $n = 5$ $\therefore$ correction $= ·0375°$. If the maximum lasts for more than a minute, count the second reading in for this purpose.

(*e*) If the water-equivalent of the bomb has been corrected for iron and acids, a further small (and invariable) correction of the calories found may be made with sufficient accuracy by deducting 20 calories from the result, or by taking off about 0·03 from the evaporative power found.

No doubt this scheme is more open to criticism from the exact-scientific point of view than the author's, but it is certainly simpler, and, as far as I can see, the error will always be less than 0·05 lb. on the evaporative power.

Mr. A. McArthur Johnston (*Past-President*): This question of sampling our coal supply and the subsequent analysis of the samples affects very closely the mining industry, and we are indebted to Mr. Murray for raising the subject in his excellent paper.

Dr. Moir has given his views on the sampling of coal and raised some points which the author of the paper seemed inclined to pass over. These points are, to a certain extent, controversial, and the discussion thereon ought to be interesting and should lead to greater accuracy and more complete agreement in the methods of sampling and analysis. With reference to sampling, let me remark that though the work is routine, the greatest care must be exercised and the duty conscientiously done, otherwise the sample is not only useless—it is deceptive. With Dr. Moir I agree that there is great difficulty in obtaining a representative sample from coal in a bunker. The same applies to mine sampling generally, and the only way to get at good results is to have a sufficient number of samples to obtain therefrom an average very near the true value.

Having obtained the sample, however, every care must be taken to secure the best economic results. I use the word economic, because we know that the sample is not sufficiently valuable to warrant us carrying an approximate analysis to the fourth, or even the third, decimal place per cent. In calorific determinations, similarly, I question whether it is advisable to read a difference of more than 0·01° C., though half this may be taken as the limit. Of course, I here refer to routine samples only and not to any special determinations for which the theoretical work in the author's paper is more applicable.

The next point I wish to bring forward is the estimation of moisture and the drying of the sample prior to analysis. The author dealt very lightly with this, and said there was little difficulty connected therewith. Dr. Moir, on the other hand, does not quite agree with him, since he has asked us to-night to consider the advisability of drying the crushed coal samples at 60° C. to 70° C. prior to making the analysis. I would point out that pulverised coal dried at 100° C. is exceedingly hygroscopic and absorbs moisture from the calcium chloride in a desiccator. In one of our experiments, we found that the crushed sample for laboratory use, when placed in a desiccator under sulphuric acid after drying at 94·5° C. (boiling point of water) showed, as soon as it was removed therefrom for weighing purposes, a slight gain each time, even though the porcelain cup had a close fitting lid. A test run for seven days, during which the sample was weighed at irregular intervals, gave from fourteen weighings, an average gain of 0·03% per weighing. These gains were not in direct proportion to the length of time in the desiccator between weighing, but seemed to be dependent on the time taken to weigh. The difference in the readings of the wet and dry bulb thermometers during this time varied between 8° and 10° C.

This result would point to the inadvisability of following out Dr. Moir's suggestion and another series of results still further illustrate the point. Here the sample (A) was cooled under sulphuric acid and weighed without a cover. The amount of pulverised (–60 = 0·010 in. aperture) coal used was 4 gm., which was placed in a porcelain flat-bottomed boat, $2\frac{3}{4}$ in. long by $\frac{5}{8}$ in. wide and $\frac{3}{8}$ in. deep. The results tabulated below were obtained from four Transvaal coals from different localities.

A. This column gives the results of drying in a hot water oven (94·5° C.) for half-an-hour, with a gentle stream of air rising through the bath. This air had been previously heated by passing through a coil placed in boiling water. Further extensions of time for heating failed to show any decrease in weight.

Barometric pressure ... ... 24·760 in.
Thermometer readings } Dry bulb ... 14.0° C.
Thermometer readings } Wet bulb ... 8·9° C.

B. The results in this column were obtained by drying as in "A" for half-an-hour. On removal from this the boats were at once placed in weighing tubes, which were stoppered and allowed to cool. Just prior to weighing, the stoppers were released to adjust the pressures.

C. These figures were got by direct estimation of the moisture in a "U" calcium chloride tube, a jacket of boiling water at 94·5° being used for heating.

Moisture Loss in Coal.

Coal.	A.	B.	C.
	Per cent.	Per cent.	Per cent.
No. 1	4·13	4·40	4·45
,, 2	3·35	3·56	3·58
,, 3	4·80	5·02	5·08
,, 4	2·67	3·09	3·13

These results in (A) show the rapid absorption of water by dry pulverised coal, and the comparison between (B) and (C) leads to the conclusion that at the temperature of boiling water no oxidation nor volatilisation of the coal takes place. It is to be remembered in this connection that Transvaal coal mines are, as a rule, not gassy.

For accurate determinations it seems to me that the only course is, immediately after sampling, to pulverise the coal sample to pass a 10 mesh (aperture = 0·070 in.) and place the samples in air tight vessels. These are tested in the laboratory without further drying, unless the sampler should report that the coal had been wetted either at the pit mouth, during transit or in the bunkers. In such a case, I would suggest that the sample be dried at 15° C. until constant weight is obtained, prior to proceeding with the determinations.

Prof. J. A. Wilkinson (*Member of Council*): As the last speaker has already pointed out, the question raised by Dr. Moir regarding the moisture content of coal is a very thorny one, as the author himself had already remarked. In this connection special experiments have been made in America and the only conclusion to be derived therefrom is that finely powdered coal is a very hygroscopic substance. The method of treatment of coal before testing its calorific value is of very great importance, since this may make, as many are aware, a very considerable difference in the result. The direct aim and object of this particular test is to determine as nearly as possible the number of heat units per unit mass of coal, that can be obtained from any given coal when used under ordinary circumstances in a boiler for raising steam. It is for the generation of heat that coal is generally purchased in South Africa and hence if this can be determined by laboratory tests under conditions as nearly identical as those used in actual practice, then such tests will be of the utmost commercial value and, I may parenthetically add, should receive payment, commensurate with the care and trouble it is necessary to bestow on such work. If this premise be granted, then it will readily be seen that to dry the coal at any temperature above that ordinarily prevailing is a mistake, because such a procedure is impossible in ordinary practice. The question therefore resolves itself into that particular method of air drying the coal in the laboratory, which will coincide most nearly with the conditions under which the coal is to be used for generating steam.

The absorption of moisture by coal was tested during the experiments carried out by the United States Geological Survey at the St. Louis Exposition 1904, and the results can be found in the Professional Paper (No. 48 page 288) which deals entirely with that work.

There is one other point referred to by Dr. Moir namely the simple formula for the determination of the calorific value. This depends wholly on the value of the constant K, but no information was, as far as I could follow, given as to how that constant was derived. If such a simple formula is possible, which I very much doubt, every coal mine manager would be extremely happy but on the other hand it would be a matter for much speculation as to whether a buyer would accept a value based on such a formula.

Dr. J. Moir (*Past-President*): As regards the formula for calorific value from ash, the constant was originally calculated from the average of some 80 samples from five coal fields here, and the point I wished to make was that the individual values of that constant only differed from that average by 1 or 2%.

Mr. A. McA. Johnston (*Past-President*): With regard to the crushed sample, does Prof. Wilkinson intend to expose that crushed sample to the room temperature, or to keep it in an enclosed bottle?

Prof. J. A. Wilkinson (*Member of Council*): I would put it into a glass stoppered bottle.

Mr. A. McA. Johnston (*Past President*): Suppose I had a sample of coal and another analyst the duplicate, these being air-dried without bottling. If I got the calorific value of my sample determined right away whilst there was a difference of eight or more degrees between the wet and dry bulb thermometers, but during the night there was a heavy thunderstorm which altered the atmospheric conditions, would the other man get the same result as I would get?

Prof. J. A. Wilkinson (*Member of Council*): My idea was that the two experiments should be carried out under similar conditions. This is, of course, an extreme case, and no one in their senses would dream that two experiments would check under such conditions.

SOME ASPECTS OF MINE VENTILATION AND TEMPERATURE ON THE WITWATERSRAND.

(Read at May Meeting, 1911.)

By J. Whitehouse, B.Sc., M.I.M.E. (*Associate*)
and W. L. Wotherspoon

DISCUSSION.

Mr. Rowland Gascoyne (*Member*): On p. 545 of the paper the authors mention the fact that at the New Modderfontein a circular shaft is being sunk at the present time which will deliver a supply of fresh air right to the deepest working. Now this practice of delivering the fresh air at the deepest point is one that ought to be practised more than it is to-day. I admit that in a deep hot mine it is difficult to carry out, when the fresh air has to pass through the workings, and a vertical intake shaft to the lowest point is by far the best means of attaining this object. Moreover, a shaft sunk to the lowest point will always conduce to better mining practice which, however, is not a subject properly coming within the scope of this paper.

The authors point out the advantages of circular over rectangular shafts in the matter of water gauge and friction, but I think it would perhaps have been better had this question been gone into more fully, as on the Rand it is becoming a very interesting question. There can be no doubt that as far as ventilation is concerned the advantages of a circular shaft over a rectangular shaft of exactly the same area are most pronounced. It is a recognised fact in mine ventilation that other things remaining the same the quantity of air a shaft will pass depends upon the rubbing surface or friction, or in other words, the quantity of air will increase or decrease in the same proportion as the rubbing surface increases or decreases. Supposing we take the largest rectangular shaft sunk on these fields as an example, a circular shaft 18 ft. in diameter would have the same area, but for the same depth it only presents to the resistance of the ventilation about 58% of rubbing surface even if we regard both shafts clear from obstructions so that a rectangular shaft of equal area and depth to that of a circular shaft will only just pass a little more than half the ventilation. Supposing, however, that the circular shaft was clear, and used solely for ventilation, as may perhaps be found necessary at some future time on the Rand, and compare it, say, with a seven compartment working shaft of similar area, we shall find that under these conditions a circular ventilating shaft will pass three times as much air as a working rectangular shaft of the same area. Clearly then on grounds of ventilation the circular shaft has an immense advantage over the ordinary rectangular shaft.

As this is a discussion essentially on mine ventilation it would be considered improper perhaps to go into other questions such as the ability of a circular lined shaft such as is being sunk at the New Modderfontein at a cost of £20 a foot to stand better through bad ground than a rectangular shaft, and the immense advantage it offers over rectangular shafts in case it becomes necessary to sink through heavily watered ground.

With regard to moisture in the mine atmosphere the action of the fan in drying the atmosphere of the mine must make for increased labour efficiency, for the drier the air the more of course is the capability of the miner to work even in high temperatures increased. I believe it is generally regarded that for temperatures above 25° C. wet bulb it becomes almost impossible for miners to work unless there is a fair amount of ventilation passing through the working place.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Positive Rays and Chemical Analysis.—"There is nothing chemical in this new method of analysis. But the study of the positive rays (Kanalstrahlen) tells the experimenter not only what gases he is actually dealing with in the apparent residual gas of his vacuum tubes, but in what state—atomic, molecular, or molecular aggregations—they are, all deduced from measurements of deflection of the rays, which measurements give a clue as to the atomic weight (or equivalent). This is the strength, and possibly also the weakness, of the new method, which bears some resemblance to spectroscopy.

The new method identifies the elements by their masses. It informs the experimenter that some substance having a certain e/m (ratio of electric charge to mass) is present, and as the experimenter knows with some degree of certainty what substances he starts with, the method affords a valuable help to the chemist, especially as it is wonderfully sensitive and records the presence and mass of elements or radicles which are far too evanescent to be isolated.

Sir Joseph said, in introducing his subject, that he approached a chemical problem with some trepidation, since chemists were fighting men. But his weapons were bullets travelling at 1,000 miles per second. The positive rays were positively electrified particles attracted to the negative electrode. When that electrode was perforated (by Goldstein's Kanal) the particles would pass through the cathode in streams which, in neon, had a beautiful orange-red colour, as was shown. The particles were deflected both by the magnetic and the electric field, and the research depended upon the measurement of these deflections.

There are several arrangements of the bulbs possible, and the bulbs are sometimes very large. In one of the new arrangements the tubular cathode is continued backward in a fine tube, terminating at two electrodes, between which an electric field is established. A magnetic field is also provided so that after passing through the two fields the particles strike the flat screen of willemite, closing the tube.

With the magnetic field alone in action, the original light spot of the undeflected particles is shifted; the deflection being equal to Ae/mv, where A was a constant depending on the intensity of the field and the size of the tube, while m was the mass of the particle, e its charge, and v its velocity. The electric field similarly deflected the particle, the deflection being equal to Be/mv^2, where B was another constant.

When both fields were excited the particles would be shifted diagonally, and it could easily be shown that the ratio of m to e should vary as the ratio of the second deflection to the square of the first deflection. If, now, all the particles were of the same kind and had the same speed, the position of P would merely depend on m/e, and a number of spots P—as many as there were different kinds of particles—would be seen. But the particles of the same kind did not move all at the same speeds, and each spot was drawn out into a curve, which was a parabola if m were constant.

Thus, taking the ratio of m to e as unity in the case of hydrogen, it is possible to calculate from measurement of the curves the relative value of this ratio for the other particles impressing themselves on the plate. Below are figures deduced from the measurements of the parabolas of the discharge through atmospheric nitrogen.

Positive Side.	Negative Side.
1·00 H +	1·0 H −
1·99 H_2 +	11·2 C −
6·80 N + +	15·2 O −
11·40 C +	
13·95 N +	
28·1 N_2 +	
39·0 A +	
101·0 Hg + +	
198·0 Hg +	

The + and − indicated, the lecturer explained, the simple or double charge carried by the particle The H = 1 corresponded to the hydrogen atom carry ing a single charge: the second, H_2=1·99 (instead of 2) to the hydrogen molecule; 11·40 C+ to a carbon atom (C=12): the 6·80 N++ to a nitrogen atom carrying two charges (N=14): the 13·95 N+ to the nitrogen atom; the 28·1 N_2+ to the nitrogen molecule; the 39·0 A+ to the argon atom (A=40): and the two last curves, 101 Hg++ and 198 Hg+ to mercury (Hg=200), carrying one and two charges respectively. The agreement between the deduced values and those assumed by chemists (figures added in brackets) was very close. The hydrogen, carbon, and oxygen found on the negative side are all in the atomic state.

Comparing the new method with spectroscopy, Sir Joseph pointed out that the spectroscopist, observing a new line, was not always certain whether he was not looking at a line of an old known substance produced under special thermal or electrical conditions. In his new method the particles wrote their names on the plate. The method was also exceedingly sensitive. He had traced the presence of helium in this way, when it could not otherwise be detected, requiring only a few particles of the substance confined in a vessel at a pressure of 0·03 mm. of mercury. The accuracy of the measurement was not dependent upon the weight of the substance present.

Although these researches had only been going on for a few months, he realized an accuracy of 1% with 0·01 milligram of substance. No chemist would undertake a determination of atomic or molecular weights under similar circumstances. Another great advantage was that the purity of this substance—the great difficulty of the chemist—was of no consequence. The method sifted the particles, merely giving an additional curve for each constituent. It is further noteworthy that the atom or particle recorded itself within a millionth of a second of its production; thus substances were detected of which the free existence had not, so far, been proved by other means.

The lecturer then exhibited slides of the curves obtained with other gases—chemically-prepared nitrogen, carbon monoxide, carbon dioxide, methane, chloroform, carbon oxychloride, etc., and he reproduced also the tables of the measurements. A few characteristic features may be pointed out. The chemically-prepared nitrogen did not give the argon line, as it should not. A faint mercury curve 65·5 Hg+++ was frequently observed, but somewhat uncertain. The carbon compounds showed 6·0 C++ in addition to the 12·02 C+.

In the experiments with methane, CH_4 (marsh gas), a veil in the neighbourhood of the carbon line had finally been resolved into four curves, 11·95 C+, 12·9 CH+, 14·05 CH_2+, 14·9 CH_3+; these were familiar groups to the chemist, but had never been obtained in the free state.

They hoped to study the transformation products of radio-activity substances in this novel way. In the case of chloroform, $CHCl_3$ and some other compounds, two obscure faint lines were observed, 1·5 and 3, the 1·5 also on the negative side; the 3 might possibly be C++++. The lecturer did not comment on the fact that none of his photographs exhibited any helium curves.

Passing to general conclusions, Sir J. J. Thomson remarked that, leaving accidental impurities out of consideration, this analysis always demonstrated the presence of the respective substance in several states or systems, atomic, molecular, or complex aggregations, carrying one or several positive and sometimes also negative charges.

Oxygen, for instance, was found in nine modifications, comprising ozone, O_6+ (the suggestion is not new to chemists), O—, and uncharged atoms and molecules. In view of this variety, it did not appear surprising that substances gave many kinds of spectra under different conditions: the various systems might give rise to different spectra.

On the negative side were found the atoms of hydrogen, carbon, oxygen, and chlorine, but never in the molecular condition; the last two would naturally be expected on the negative side, and the association of hydrogen (though a highly electropositive element) with negative charge was no longer a novelty.

As regards these curves on the negative side, it would appear that negative particles were carried through the cathode and that they were picked up by neutral particles and held by attraction. Only the atoms, not the molecules, seemed to be able to do this, and the phenomena illustrated the different behaviour of atoms and molecules. Apparently the atom, not the molecule, was capable of exerting momentarily a very powerful attraction. The explanation was probably that the atom had more freedom of motion than the molecule. When magnetism acted on a sphere of a magnetic substance, the electricity within that sphere was free to move and to readjust itself in accordance with the magnetic induction, and a strong attraction resulted. On a sphere of sulphur the freedom was limited and the attraction was weak.

In illustration of this view, about half a dozen compass needles had been mounted, each on its vertical pivot, on a piece of cardboard, which was held suspended by strings in front of an electromagnet. When the electromagnet was excited, the whole system on the cardboard was attracted toward the electromagnet; but when the needles were taken off their pivots and placed on the board, on which they could no longer move freely, hardly any attraction was observed. Similarly, Sir Joseph remarked, there were stiff and freely-moving atomic and molecular systems. It depended upon the numbers of corpuscles in the atom; one corpuscle more than a critical number would (as he had explained on other occasions) impart to the atomic system one freedom of motion, one bond or one valency; two free corpuscles would correspond to two bonds and two valencies. When the atoms joined one another to form molecules, the bonds were linked, the system became rigid, and it failed to attract bodies outside. These researches, Prof. Thomson concluded, might help forward the solution of the problems of chemical combination."—Sir J. J. THOMSON, *Engineering*, April 14.—*Metallurgical and Chemical Engineering*, June, 1911, p. 298. (J. A. W.)

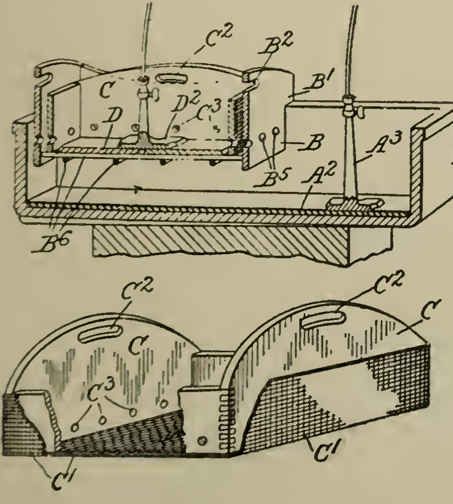
AURIC HYDROXIDE.—"The compound formed on a gold anode consists of auric hydroxide with about three molecules of water. If K_2SO_4 is used as the electrolyte, the deposit contains potassium, probably as $KHAu_2O_4$. The auric hydroxide deposit is translucent, ruby red and apparently crystalline. When either auric hydroxide from the anode or the amorphous form obtained by the hydrolysis of auric nitrate or sulphate is heated it gradually loses water up to 172° but does not become anhydrous at 200°–210°. At temperatures above 172° the compound slowly gives off oxygen but does not yield aurosoauric oxide Au_2O_2 as stated by Krüss or Au_2O_3 auric oxide as indicated by Schottländer. When auric hydroxide is treated with hydrogen peroxide or a solution of sodium peroxide, it is reduced to the metal. The deposit formed on a gold anode in presence of ammonium carbonate contains a fulminate."— W. S. MINTER, *Journal of the American Chemical Society*, 1911, 33, 688. *Journal of the Chemical Society*, Vols. 99 and 100, July, 1911, Abstracts ii., 613. (J. A. W.)

THE SOLUTION OF TIN ORES.—"The writer has discovered that whereas it is very difficult ordinarily to render tin ores soluble by fusing them with caustic soda, the addition of 2% of powdered charcoal, or some similar reducing agent to the melt, will cause it to become soluble with the greatest ease. The action is thought to be *catalytic*."—A. GILVERT, *S. Offentl. Chem.*, vol. 16. *Transactions American Metallurgical Society*, July, 1911, p. 28. (H. A. W.)

METALLURGY.

ELECTROLYTIC PARTING CELL.—"What may be considered as an interesting further development of the well-known Balbach cell for parting gold and silver is patented by Mr. William Thum, of Hammond, Ind., and shown in section in the sketch. A is a tank with an inclined bottom on which rests the cathode plate A^2 (for instance, of carbon), connection to the outside circuit being made through the cathode terminal A^3. The anode cell B is supported in the tank A by means of the projecting ledges B^1. Perforations B^5 are provided in all the walls. The bottom of the cell consists of a grid made up of the bars B^6. The anode basket C, which is located within the anode cell and rests upon the grid B^6, is provided with the canvas bottom C^1, as shown in the lower diagram, the handle holds C^2 and the perforations C^3 in its sides and ends. The anode plate D is situated within the basket C and rests upon the spacing bars, which rest in turn upon the canvas bottom of the basket. D^2 is the anode terminal. A modification of this construction is also described, in which the anode cell and basket are made in one.

If silver bullion containing gold is to be treated an electrolyte of silver nitrate with free nitric acid is used. The electrolyte is contained in the tank A at such a level as to be slightly above the perforations B^5 and C^3 in the anode cell and anode basket respectively. The silver dissolves from D as nitrate and the heavy silver nitrate solution sinks through the canvas and runs down over the full length of the surface of the inclined cathode plate A^2 on which the silver is deposited in shape of loose crystals. The electrolyte, the specific gravity of which has thus been reduced, rises now upward again and flows back across the upper part of the tank toward and into the anode cell through the perforations B^5 and C^3. The gold in the anode plate D is not dissolved and accumulates in the shape of mud or slime resting loosely on the perforated bottom of the basket.



Electrolytic Cell for Parting Gold and Silver.

The cell may be similarly used for refining bismuth. In this case an electrolyte of bismuth chloride and free hydrochloric acid is used. The bismuth is obtained in form of more or less coherent easily removable crystals, having the shape of trees, mushrooms and the like."—W. THUM.—*Metallurgical and Chemical Engineering*, July, 1911, p. 379. (J. A. W.)

PRINCIPLES GOVERNING AGITATION IN PACHUCA VATS.—"In the Pachuca vat, agitation takes place in the body of the vat, during the downward motion of the pulp and does not manifest itself by any vigorous movement. The agitation in the air-lift is negligible on account of the shortness of the time during which it takes place, and its effect cannot be determined experimentally. The air-lift only serves as a convenient means of aerating the pulp, and lifting the solid particles from the bottom of the tank, after they have fallen through a column of solution equal to the height of the tank. This column itself has a downward motion, caused by the action of the lift. What concerns us is to know the rate of displacement of the solids in the solution.

Calling the velocity of the liquid v, and that of the solids v' we should endeavour to make $v'-v$ as large as possible. As v' is a function of gravity, and the retarding influence by the resistance of the medium, it is a constant in every case, determined by the degree of grinding and the consistency of the pulp. The factor v, however, is controllable within certain limits by regulating the quantity and pressure of the air. The greatest displacement of the solids in the pulp is obtained when the current in the lift is just sufficient to keep the cone clear of accumulation of solid particles. Therefore, the less vigorous the circulation the more perfect the agitation.

To determine the value of $v'-v$, a number of experiments were made. An experimental Pachuca vat of 18 in. dia. and 54 in. high, with air-lift, etc., proportioned as in the large vats, was fitted so that the air-lift discharge could be collected for a short time, instead of being returned to the vat. A charge consisting of a weighted quantity of ground ore and a measured quantity of water was made up, and when normal circulation was established, and soundings showed that no ore had settled near the bottom, samples of about 3 liters were taken at intervals from the whole of the air-lift discharge, and each sample investigated separately. As the results closely agreed, the average is used in the argument. Three different charges were used, as follows:—

I. Three parts water and 1 part ore by weight.
II. Two parts water and 1 part ore by weight.
III. One and one-half parts water and 1 part ore by weight.

Each litre of pulp therefore contained solids as follows:

	I.	II.	III.
(*a*)	294 g.	415 g.	524 g.

The solids per litre of air-lift discharge were:

	I.	II.	III.
(*b*)	426 g.	501 g.	580 g.

The solids have therefore moved at a greater velocity than the water and

$v' = \frac{(b)}{(a)} v$. The velocity of the solids was, therefore,

I.	II.	III.
1·45 v	1·21 v	1·11 v

The sizing test of the ore was as follows: All passed 80 mesh: 9·2% was retained on 100 mesh; 27·8% on 200 mesh and 63% was finer than 200 mesh. A litre of pulp contained, therefore,

	Mesh.	I. Grams.	II. Grams.	III. Grams.
(c)	+100	27·05	38·18	48·21
(d)	+200	81·73	115·37	145·67
(e)	–200	185·22	261·45	330·12

The air-lift discharge consisted of the following sizes:

	I. Per cent.	II. Per cent.	III. Per cent.
+100 mesh	14·2	12·5	11·3
+200 mesh	39·4	32·4	30·3
–200 mesh	46·4	55·1	57·4

and the solids contained in one litre of pulp:

	Mesh.	I. Grams.	II. Grams.	III. Grams.
(f)	+100	60·49	62·63	65·54
(g)	+200	167·86	162·32	175·74
(h)	–100	197·66	276·05	338·72

The velocity of the particles of various sizes is,

$$\frac{(f)}{(c)} v, \quad \frac{(g)}{(d)} v \text{ and } \frac{(h)}{(e)} v,$$

for the three classes of pulp, therefore,

Mesh.	I.	II.	III.
+100	2·23v	1·64v	1·36v
+200	2·05v	1·40v	1·20v
–200	1·07v	1·06v	1·03v

The segregation of the particles takes place, not only according to size, but also according to specific gravity. To calculate the velocity of the sulphides, a sulphur determination was made before and during agitation, as follows:

TANK CHARGE.

	I.	II.	III.
Per cent. of solids ..	5·7	5·7	5·7
Grams per litre ...	16·76	23·65	29·87

AIR-LIFT DISCHARGE.

	I.	II.	III.
Per cent. of solids ..	7·7	6·9	6·6
Grams per litre ...	32·80	34·57	38·28
Velocity of sulphides	1·95v	1·45v	1·28v

The most interesting item of the information obtained by these experiments is the displacement of the particles containing silver in the pulp.

TANK CHARGE.

	I.	II.	III.
Grams silver per metric ton ...	995	995	995
Milligrams silver per litre pulp	292	413	521

AIR-LIFT DISCHARGE.

	I.	II.	III.
Grams silver per metric ton	1085	1046	1012
Milligrams silver per litre	446	526	587
Velocity of silver ...	1·53v	1·27v	1·12v

So far we have only considered the relative velocity of the solids. To find the absolute displacement, it is necessary to know the velocity of the pulp in the air-lift. Experiments for the construction of empirical formulæ are in progress. It is certain that those for lifts pumping water to a great height are not applicable.

Calling the velocity of the pulp in the lift V, then, the diameter of the lift being one-twelfth of that of the tank, $v = \frac{V}{143}$.

Assuming V as 100 ft. per minute as correct, and the section of the lift as 1 sq. ft., then 100 cu. ft. of pulp is lifted per minute, which contains 88, 83 or 79 cu. ft. of solution according to the degree of density. Taking the last, as the least favourable, v is equal to $\frac{79}{143}$ = 0·55 ft. or 6·6 ins. per minute.

The displacement of the particles in the solution is then

+100 mesh	0·36v = 2·376 in.
+200 mesh	0·20v = 1·320 in.
–200 mesh	0·03v = 0·198 in.
Sulphides in general ...	0·28v = 1·848 in.
Particles containing silver ...	0·12v = 0·792 in.

Apparently the displacement increases with v and the statement that for efficient agitation v should be kept as low as possible seems contradictory. But the experiments were made with the *proper* amount of circulation. If the latter is increased needlessly, the velocity of the *liquid only* will be increased and the values found above will become smaller.

The important facts brought out by this investigation, incomplete as it is, are: That all of the components of the ground ore continually change their position with regard to each other and the solution, thus presenting the whole of their surface to be attacked by ever changing portions of the latter, leaving behind them a trail of quasi-saturated solution, which is obliterated in the air-lift (if not before) by secondary currents in the tank. The amount of this displacement, even in the most viscous of the pulps experimented with and for the finest and lightest of the particles, is several hundred times their diameter."—A. GROTHE, *Mining World*, May 27, 1911, *Transactions American Metallurgical Society*, June, 1911, p. 26. (H. A. W.)

MODERN ZINC PROBLEMS.—"In his presidential address delivered before the Institution of Mining and Metallurgy, London, Mr. H. Livingston Sulman gave an interesting and instructive review of zinc metallurgy and the problems yet to be solved.

In no department of metallurgy has development been more rapid than in the treatment of 'complex ores.' Just as the terms 'free milling' and 'refractory' applied to gold ores have lost their older significance, so a few years hence the word 'complex' will cease to be used as a prefix to zinc-lead-silver-copper ores, and as a reproach to reduction practice. Many millions of tons of such material have been found, and in some instances partly developed, whose contained metals at current prices render them equivalent in value to 2 oz. gold ore. The 'complexity' usually is due to blende, usually more or less ferruginous, in intimate physical association with galena and other minerals. When it is possible to dress these ores so as to produce a clean galena concentrate and a zinc portion containing 40% to 45% metal, the ores are not 'complex,' but the finer parts and slime are still complex in the sense that their separation still awaits solution. The Broken Hill deposits with their intercrystallised galena and ferruginous blende, rhodonite and garnet gangue, which formerly prevented the separation of zinc, are of decreasing complexity since the advent of flotation methods. The slime reserves, however, still present difficulties of treatment.

A more truly complex problem is that which occurs in Tasmania, where the presence of copper adds to the problem. The difficulty in this instance is that the ore contains but little gangue, and concentration by any method is of little or no effect. Successful reduction must be sought by direct smelting, or the removal of the zinc by leaching processes. Complex carbonates also are known, as those of the Rhodesia Broken Hill region, where zinc and lead occur as mixed carbonates. Again, there are those ores in which the presence of fluorspar prevents the usual retort reduction methods for zinc, and which probably will have to be reduced in the electric furnace.

The methods of attacking such ores may be divided into proximate or mechanical, and ultimate or leaching and smelting processes. Proximate methods are limited by the conditions encountered in crushing heterogeneous materials to free the component parts, and usually are inefficient by reason of having to produce several grades of mineral, or a low grade if all be mixed. Among the proximate methods which have come into prominence is the flotation group.

The simplest type of flotation methods is represented in the De Bavay process used at Broken Hill, and in the Macquisten tube. In these the surface tension of water is used to float off the sulphide particles, which are not easily wetted, whereas the more readily wetted gangue particles sink when a mixture of mineral and gangue is presented to a surface of flowing water. A second variety is the Potter process, in which submerged sulphide particles are caused to rise to the surface of a hot acidified pulp by the fixation of gas bubbles: the gas in this instance being carbon dioxide liberated by the acid from siderite. The third variation includes the well known oil processes of Elmore and Minerals Separation, Ltd., which secure the flotation of submerged minerals by means of bubbles of air, such mineral particles having previously received a slight film covering of oil for the purpose of effecting the air or gas attachment. To Elmore undoubtedly belongs the honour of first bringing oil-mineral processes into the realm of commercial practicability. To Froment and especially to Cattermole we are indebted for the next important development. The latter first demonstrated the efficiency of a very small quantity of oil in place of the large bulk hitherto employed.

The physical reactions which underlie all such methods have been insufficiently explored, and in many respects are still obscure; the striking practical results so far achieved have been reached quite empirically. The author mentions a new fact in relation to measurements of the 'contact angles' which exist between the surfaces of aqueous liquids and those of certain sulphides, silicates and other solids. In investigating previously accepted figures for these, he finds discrepancies in the readings, which on further investigation prove to be due to the existence of a variable range for the contact angle between various liquids and solids, though of constant magnitude of variation for each. This range of angular variation has been called by C. G. Lamb, 'the angle of hysteresis,' and its existence and magnitude in regard to many substances have been independently confirmed and quantified by A. Howard Higgins. It is not a fact of merely academic interest, as it appears to be intimately connected with the ability of a given solid to condense upon itself a gas film; and then submerged in a liquid to determine the attachment of a gas when generated, or on the point of generation, from a solution. It follows that angular hysteresis reaches its highest values for minerals which are most susceptible of flotation and *vice versa*. Hundreds of observations have been made which place this relationship beyond doubt. Besides roughly quantifying the gas condensing power due to the surface energy of solids, it brings us somewhat closer to an explanation of the efficiency of acidification. Thus, whereas the angular hysteresis of silica in plain water may exceed 30°, thus indicating that substance to have a definite power to occlude gas and to float, it drops from 4° to 0° in water acidified with sulphuric acid. Galena, on the other hand, retains its full measure of angular variation or is but slightly affected.

While certain difficulties remain to be solved, especially in regard to blende and galena slime, the Broken Hill problem may be regarded as solved by the flotation processes. It must be remembered, however, that the methods referred to work effectively only on particles of sensible size.

The slime problem has been studied by Horwood, who operates by superficially attacking one particle while leaving the other unaltered, and thereafter submits the product to oil flotation. He usually employs heat to effect this differentiation, a low temperature oxidising the surface of the galena while the blende remains unaltered. The Minerals Separation process is preferred for this work as it deals better with slime.

Magnetic separators employing fields of high intensity still have a wide range of usefulness, although on dry ore have given trouble on account of dust. Wet types have been extensively experimented with, and lately Mr. Ullrich has introduced a multipolar ring machine which overcomes the difficulty of removing small particles from a liquid. Several of these machines are already working, and their possibilities regarding slime mineral are receiving close attention.

Electrostatic methods are revived in the Huff machine. To L. I. Blake is due the credit for first introducing a commercial separator on the electrostatic principle. He was followed by Sutton and Steele, in whose machine the electrode was charged inductively by a brush discharge, and was furnished with an ingenious interruptor whereby any desired degree of saturation of the separating roller could be obtained to suit the varying electrical capacities of the minerals to be separated. An electrostatic generator also was employed in this machine. The main advance in the Huff machine is the substitution of a dynamo current of suitable intensity for the electrostatic energy of a Wimshurst machine. The limiting factor of electrostatic separators is their inability to handle fine powders, and their liability to dust troubles unless the dust be first removed.

In the mechanical treatment of slime we reach the limit of proximate methods of handling zinc ores, and must take up ultimate methods such as smelting and hydrometallurgical processes. Dealing with smelting, the author mentions H. Pape's process in Germany. Low grade zinc-lead ores are smelted with sufficient silicious ore to slag off the zinc and reduce the lead. The slag of zinc silicate is then crushed and briquetted with sufficient carbon and a binding agent, and charged into a blowing furnace, fitted with a special grate on which the briquets are charged to a depth of 3 ft. or 4 ft. Upon ignition, the zinc is first reduced and then blown off as oxide, while the depleted slag flows through the grate. The separation goes on almost to completeness, and the zinc oxide is collected in chambers and bags.

Referring to leaching methods, the bisulphite process is described as follows: Roasted ore is subjected to the action of water and dissolved sulphur dioxide gas, the latter being derived from the roasting operation. The zinc is dissolved as bisulphite and separated from the residue which contains lead, silver and other metals. From the solution the zinc is thrown down as insoluble monosulphite by removing one of the equivalents of sulphurous acid gas. The precipitate is then calcined, resulting in the production of a dense zinc oxide suitable for reduction to spelter. Sulphate of zinc accumulates gradually in the liquors until they are strong enough for recovery of the salt by crystallisation.

A means for the reduction of zinc sulphate is provided by electrolysis, the deposited zinc being stripped from the cathode and remelted to spelter, while the acid regenerated at the anode is used for the leaching of fresh oxidised zinc from roasted ore. In practice it is found that about 5,500 electrical horse power, as a mean figure, is required to produce a ton of deposited zinc, or roughly two-thirds of an electrical horse power year. Wherefore, localities favoured with cheap water power will hold an advantage in the production of electrolytic zinc, for the process could hardly supersede the present distillation processes if located at existing reduction centres. The author concludes with a review of the electric processes for zinc."—H. L. SULMAN.—*Metallurgical and Chemical Engineering*, June, 1911, p. 323. (J. A. W.)

ROTARY FILTER.—"A modification of the type of revolving filter commonly used in the United States has been patented by Mr. Alexander J. Arbuckle, of Johannesburg, Transvaal, South Africa. In the Oliver and Portland revolving filters used in this country the filtering medium is on the outside of the drum only, but in the Arbuckle filter it is proposed to make use of both the outside and inside surfaces of the drum for filtering purposes. The drum is divided into sections which, by means of a three-port valve located in the hub of the drum, are brought successively into connection with a vacuum for forming the cake, a wash and vacuum for removing the retained values and compressed air for discharging the cake. For this purpose three non-communicating spaces are provided in the hub, one for each process mentioned. Consequently, as the drum revolves in a vat of ore pulp, each section successively forms a cake on both its surfaces, which are later washed and discharged. (994,235, June 6, 1911.)"—A. J. ARBUCKLE.—*Metallurgical and Chemical Engineering*, Aug., 1911, p. 423. (J. A. W.)

RECOVERY OF GOLD-BEARING CHARCOAL ON RE-TREATING FILTER-PRESSED SLIME.—"Mr. Victor T. Edquist has furnished some interesting notes on the re-treatment of a quantity of slime at Lakeside. The slime is from the Great Boulder Perseverance mine; it was fine-crushed (dry) on the mine and subsequently railed to Lakeside, some three miles distant, where it was cyanided and filter-pressed. In the process of trucking the slime from the presses to the dump, it was found that, owing to its moist and sticky nature, the slime did not discharge readily when the trucks were tipped; much of it adhered to the bottoms and sides of the trucks. To overcome this difficulty the insides of the trucks were sprinkled with wood ash before loading, and this practice accounts for the large accumulation of charcoal in the dump. Owing to the moisture in the residue, the charcoal has re-precipitated much of the soluble gold that remained in the dump, and, in the course of re-treatment, it has been found desirable to separate the charcoal from the slime before agitation. Mr. Edquist describes the process as follows:—

During the re-treatment of the large dump of filter-press residue at Lakeside, quantities of charcoal and various refuse are met with and removed.

The material to be treated is carted to a bin, from which it is elevated to the mixers by a conveyor belt. After passing through the mixers the pulped slime gravitates through a revolving screen or trommel of punched plate with holes of $\frac{3}{16}$ in. diameter. Thus, any hard nodules of slime or any charcoal or other refuse are removed. When sufficient of this reject has accumulated, it is again passed slowly through the mixers with an excess of water. The second mixing eliminates the slime, leaving a clean product consisting of charcoal, chips, jute, waste, etc., which is rich in gold, ranging from 20 oz. to 40 oz. per ton. Fine particles of charcoal that escape through the trommel screening and flow to the vats, are removed by hand-skimming the charge during agitation.

After rough burning-off to reduce the wood and fibre contents, the product is sampled and bagged for shipment to the smelters.

The total gold recovered from this source amounts to 5 gr. (tenpence) per ton of slime treated.

Charcoal is the chief gold-bearing agent, its value running up to 100 oz. per ton; other organic matter varies in value from 1 oz. to 7 oz. per ton.

Ash charcoal, and other plant refuse were originally dumped with the filter-press residue, and are to be found throughout the dump. The richness of this débris is due to contact with contained gold-bearing moisture in the material dumped; to subsequent action of residual cyanide on gold in the residue; and to the movement of moisture through the mass by natural weather conditions.

The gold contents of the slime average in value 10s. 6d. per ton; the slime contains 23% of moisture and carries gold in solution to the value of 1s. 8d. per ton of slime.

When removed, the charcoal is slime-laden and water-logged, and considering these conditions and the relatively low value of the surrounding slime, its value is surprising.

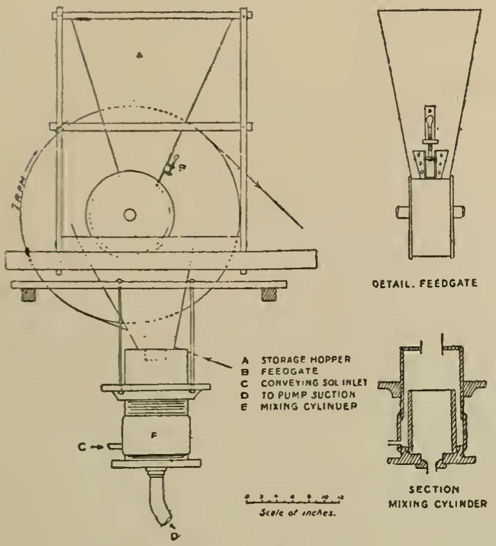
Under the most favourable conditions, practically the limit of enrichment has been reached, this being probably due to the presence of gold-bearing moisture after all cyanide or other gold solvent had been decomposed.

Another interesting feature is the selective action of the charcoal. The bullion obtained from the solution contains 18% of silver, whereas that yielded by the charcoal carries only 3% of silver and a quantity of mercury.

No attempt has been made to determine the relative percentages of mercury and gold, for the reason that a considerable amount of the former is driven off during burning operations."—V. T. EDQUIST.—*Monthly Journal of Chamber of Mines of Western Australia* June, 1911, p. 152. (H. A. W.)

EXTRACTION OF GOLD AND SILVER FROM SLAG.—"The question as to what to do with poor slag carrying gold and silver is constantly coming up, and the following method of dealing with it may be of interest. The slag was taken from a poorly-fluxed melt and assayed 112·5 oz. Ag and 2·41 oz. Au per ton. A charge of 20 kg. was ground in a small tube-mill for 12 hours, when only 5½% was retained on a 200-mesh screen; this part containing 48% of the silver and 53% of the gold, so that a good concentration of the precious metals was obtained in the tube-mill. The slimed portion assayed 60 oz. Ag and 1·12 oz. Au per ton; this was agitated for 120 hours with 0·5% KCN solution at a dilution of 2 to 1. The residue then assayed 3·2 oz. Ag and 0·06 oz. Au per ton. The extraction by agitation was 91% of the silver and 92% of the gold; the combined extraction by concentration and cyaniding being 96·6% of the Ag and 97·5% of the Au. A further improvement might be found in crushing originally in strong cyanide solution, cleaning out the tube-mill, and melting the +200-mesh material, the slimed product going on to the ordinary slime treatment."—E. R. Haggin.—*Mining and Scientific Press*, June 3, 1911, p. 746. (H. A. W.)

Zinc Dust Precipitation.—"The formation of calcium carbonate during precipitation by zinc dust and the detrimental effects of this salt on precipitation are set forth by Mr. Allan J. Clark in the April, 1911, issue of the *Mining Magazine* (London). The formation of calcium carbonate may result from agitating, with compressed air, the solution before the precipitation of the previous metals, or by agitating with compressed air to form the zinc emulsion used for precipitation.



Zinc Dust Precipitation.

At the time the author's attention was first directed to the matter in hand, the method of preparing the zinc emulsion was that of agitating by compressed air in an inverted cone with a continuous overflow to the suction of the pump, where it was mixed with the solution flowing to the presses. The first instance of the effect of calcium carbonate was noticed while precipitating a very weak and low grade solution of high alkalinity. This solution had been collected in shallow vats and agitated with compressed air, and was in prime condition for the formation of the objectionable salt. The precipitation offering some difficulty, the rate of flow was reduced without improvement in precipitation. The press was then opened, and a heavy deposit of calcium carbonate was found, caked on the press cloths or mixed with the precipitate. By eliminating the preliminary agitation of the solution the difficulty was remedied.

The author suggests that the formation of calcium carbonate diminishes the efficiency of the zinc by coating the zinc particles and preventing further action.

By eliminating the mixing cones and substituting the method of allowing the zinc dust to fall from an endless belt into a trough where it was washed by a stream of solution direct to the precipitating line, there was an increase in concentration of 63%; a decrease of consumption per ton of solution of 35%; and a decrease of zinc consumption per ounce of gold precipitated of 38%.

Later, on account of the irregular action of the belt feeder, a new design of zinc dust feeder was installed. It is designed by W. H. Todd, foreman at the Homestake mill. It consists of a hopper with feed gate, and a mixing cylinder where a small stream of solution emulsifies the dust. No air agitation is used. Results of the first month's operation of the new feeder compared with the former method of emulsifying in a cone with air agitation show: Increase in concentration, 80%; decrease in consumption of zinc per ton of solution, 33%; decrease in consumption of zinc per ounce of gold precipitated, 31%. The results warrant the statement that a reasonably regular supply of zinc is sufficient for all practical purposes when, as is usually the case, the zinc and solution travel some distance before entering the press."—A. J. Clark.—*Metallurgical and Chemical Engineering*, July, 1911, p. 380. (J. A. W.)

Slime Agitation at Kalgoorlie.—"We have the choice of the ordinary, A. Z., Kalgurli, inverted cone, Brown and its modifications, silica sponge, air-lift, centrifugal pump, and numerous other styles of agitators, each with its advantages and disadvantages. Agitation as practised at Kalgoorlie, has never been described in a general way, so these notes may be of interest to mill-men.

The following table shows how the ore is crushed and ground in our mills:

Name.	Manner of reduction to slime.
Associated	Ball-mills and pans.
Associated Northern	Ball-mills and pans.
Gt. Boulder Perseverance..	Ball-mills, pans, and tubes.
Gt. Boulder Proprietary	Ball-mills, Griffin mills, and pans.
Hainault	Stamps and pans.
Ivanhoe	Stamps and pans.
Kalgurli	Ball-mills and pans.
Lake View & Star	Stamps, pans, and tubes.
Oroya Links	Stamps, pans, and tubes.
South Kalgurli	Ball-mills and pans.

The product in the dry-crushing mills will average about 87% through 150 mesh, while in the wet-crushing plants it will be over 90 through the same screen.

Each system in use calls for some little comment by way of explanation: The A. Z. agitators were introduced at the Associated mill after a long test with the ordinary type. The roasting used to be poor with some of the furnaces, and as the A. Z. agitator showed as much as 4% better results than the latter they were erected. As regards time of agitation, power consumed, and loss of cyanide, it is a case of 1 to 1½ hours, 12 hp., and 1 lb. KCN per ton, against 2 to 12 hours, 2½ hp., and about 1½ lb. KCN respectively, to which must be added the increased extraction. These vats are small, but of course the gear may be made for any size tank.

The three propeller blades are fixed on a vertical, gear-driven spindle, revolving at 60 r.p.m. The spindle is arranged for upward and downward thrust by having ball-bearings in the head frame, from which it is suspended. The blades are set at an angle of about 25°, and work in a 6 ft. ring, made of plate about 12 × ½ in., bolted to the bottom of the vat, and standing about 6 in. above the bottom in order to allow the pulp to circulate to the sides. Directly under the propeller, a false-bottom is bolted to the vat to prevent the sandy particles scouring the bottom. The gear and head-frame stand on 10 × 6 in. H-iron bolted across the top of the vat, and from this are suspended baffle-plates which extend nearly across. The propeller is not started until the vat is half full of slime, and it may be stopped for any time and re-started, the bottom being scoured clean. This is one good point in the A. Z.; a very sandy pulp rarely prevents its starting. When full of slime, the flow from the periphery to the centre is at the rate of about 3 ft. per second, the pulp thereby being in violent agitation, and all parts thoroughly exposed to the air. This is where the main advantage of the A. Z. lies, and other ordinary types fail. It may be thought that the consumption of cyanide would be high on account of such violent agitation and exposure to the air; but this is not so, and, providing the solution is in good order, even though the roasting is poor, it will keep up its strength well, as the following tests show:

	Per cent KCN			
At start of agitation	0·072	0·062	0·066	0·064
After ½ to 3 hr. agitation	0·062	0·056	0·060	0·052

A peculiar point worth notice is that slime which has been agitated for four hours or so, shows at times a higher residue than that which had only one to two hours. Of course, being a gold-bearing ore, solution is effected much quicker than if it contained silver: and it was pointed out to me by an American mill-man that the A. Z. would not pay in agitating silver ore, this requiring up to 90 hours in a Pachuca tank. This is no doubt due to long contact with the air used. Would not a fast-running machine of the A. Z. type expose the pulp to the air sufficiently in a shorter time? In adding cyanide to the A. Z., it is not broken up at all, but lumps of as much as 30 lb. weight are thrown into the slime without damage to blades or gear. Agitation at the Associated Northern calls for little note. If the roasting and subsequent grinding is good, the agitators do good work, and an ordinary circulating solution of 0·04% KCN is sufficient for solution of the gold, this strength being made up to 0.08% before passing through the extractor boxes. At times extraction falls off, and blowing compressed air in different points of the pulp has been tried, but somehow it is not of much benefit.

The ordinary agitator acts like a buddle, in that coarse particles settle in the bottom, due to concentration. This concentrate is shoveled out occasionally, and assays as high as $25 per ton, part of the value being in the form of fine amalgam washed over from the pans. This is dried, sent to the mills, and re-treated. The Perseverance has a very large agitation plant doing good work. The ordinary vat calls for no comment; but Mr. Wright, the metallurgist, has devised a new scheme for more rapid circulation of the slime. At an angle of 45°, from the central spindle toward the side of the vat, fixed to the stays of the agitator arms are several pipes up to 10 in. diam., with their tops almost submerged below the surface of the pulp. By the centrifugal force imparted, the pulp is forced from the bottom of the pipes to the top, and flows out in a good stream, thus ensuring that the slime at the bottom of the vat is well mixed with the top section. Mr. Wright believes that it only takes a few minutes with large pipes to circulate all the pulp in a vat.

The ordinary practice is followed at the Great Boulder. The Hainault has devised a collecting and thickening vat, the slime from this is circulated by a Frenier special pump, and then flows over a sort of cascade for aeration to an agitator of the ordinary style, prior to filter pressing. With the ordinary type, the Ivanhoe agitates its slime with bromo-cyanide. Air agitation is in use at the Kalgurli in tanks somewhat similar to the Brown system, only that they are smaller, and there is no central lift. This system has been in use for nearly ten years, a considerable time before the latter came into use. The tanks are 12½ ft. deep. The first 7 ft. is 6 ft. diam., and from that point the remaining 5 ft. tapers off to 18 in. at the bottom. For the air, there are two 1 in. pipes fitted near the bottom of the tanks, and a little air at 35 lb. pressure is admitted. The pulp first boils quickly. It takes about 20 minutes to fill each from the settlers, and about 20 minutes to empty by means of a slime-pump which fills the presses, there being a discharge-pipe and valve at the bottom. Each agitator will deal with 24 tons daily." —M. W. VON BERNEWITZ.—*Mining and Scientific Press*, June 3, 1911, p. 749. (H. A. W.)

MINING.

FROZEN DYNAMITE.—"Frozen dynamite is the subject of grave suspicion among most engineers, says the *Engineering Record*, yet experts hold that it is far less sensitive than unfrozen dynamite. In order to prove this, Dr. Walter G. Hudson and E. J. Riedered, the latter superintendent of the Du Pont works at Lake Hopatcong, recently conducted some experiments that are decidedly interesting. They were made with a particularly sensitive grade of 60% gelatine dynamite, and also with some straight dynamite. A number of these sticks were used as targets for bullets from a Krag-Jorgensen Government rifle, loaded for a velocity of 2,150 ft., and fired at a distance of 50 ft. from the dynamite. The bullets discharged in this way failed in every case to explode the sticks of frozen, straight, and gelatine dynamite. Other frozen sticks were then thawed, and used as targets in precisely the same way. In every case the bullets detonated them, thus showing the decreased sensitiveness due to freezing."—*Engineering Record.—South African Mining Review*, July, 1911, p. 14. (M. H. C.)

[I would like to add that frozen dynamite or gelatine is very susceptible to fire and easily burnt, and it no doubt often happens that the detonator, though failing to explode the frozen sticks of gelatine in the hole, will set them alight, thus causing a "burnt out hole, with the accompanying gassing cases, bad heads, and general discomfort to the workmen. A frozen stick of gelatine may also be the cause of a misfire accident, by remaining in the bottom of the hole unexploded in its frozen state; but thawing out by the aid of the natural rock temperatures, intersected by a hole in the course of being drilled, an explosion takes place, and "no one knows where the gelatine came from!" —M. H. C.]

ROUGH TEST FOR TIN IN CASSITERITE.* "We owe to the kindness of Mr. U. P. Swinburne, Inspector of Mines, Natal, and formerly Inspector of Mines of the Pretoria District, Transvaal, the following suggested rough tests for tin which we recommend to prospectors.

1. If the presence of tin be suspected in a sample of rock, crush, but not too fine and pan off the concentrate.

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* See this *Journal*, Vol. viii., p. 72.

2. Place a few particles of the concentrate in a small porcelain dish or in a cup or saucer and place a few small pieces or strips of zinc in contact with the mineral. Add a little diluted hydrochloric acid and allow to stand till all action of zinc with the acid has ceased.

3. If tin is present the particles tested will become covered or partly covered with a grey coating.

Before using the above test, however, the following tests should be applied:

1. If the particles are black apply the magnet and if magnetic they are probably magnetite and it is useless to test further for tin.

2. Crush between two silver coins and should the powder be black it is useless to apply the zinc test, powdered cassiterite is a pale grey colour.".—U. P. SWINBURNE.—*Mining Journal of the Mozambique Company*, April, 1911. p. 19. (H. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

PHYSICAL CHEMISTRY: ITS BEARING ON BIOLOGY AND MEDICINE. By J. C. PHILIP, D.Sc., Ph.D. (Edward Arnold, 1910. Pp. vi+312. Price 7s. 6d. net.)

"The methods of physical chemistry have found ever-increasing application in recent years for the investigation of biological problems, and the present book is intended to give a systematic account of the principles and methods of physical chemistry which are of importance in biological and physiological work. The more important subjects, which are dealt with in considerable detail, are: General properties of gases, osmotic pressure, permeability of membranes, the properties of dilute solutions, electrolytic dissociation, colloidal solutions, adsorption, chemical equilibrium and the law of mass action, velocity of chemical reaction, including enzyme action. Dr. Philip is heartily to be congratulated on the success with which he has discharged his task. The exposition is so clear and logical that the book can be read and enjoyed by a student with a comparatively elementary knowledge of chemistry and physics. The advanced student, on the other hand, is supplied with an accurate and thoroughly up-to-date account of the subjects dealt with, and numerous references are given, so that further information on a particular subject can readily be obtained.

In the course of careful examination of the book we have found practically nothing that calls for criticism. In the section on "Some Peculiarities of Enzyme Action," p. 294, reference should have been made to the work of C. S. Hudson (from 1908 onwards), who has shown that the work of Henri and others on the hydrolysis of cane sugar by invertase is largely vitiated by the mutarotation of the glucose and fructose. This doubtless accounts for the deviation from the law of mass action illustrated by the example on p. 294, but does not affect the general conclusion that at least some enzyme actions are abnormal in the sense indicated, since Slator (*Trans. Chem. Soc*, 1906, 89, 128) has found by a non-polarimetric method that above a certain concentration of glucose the rate of reaction is independent of the glucose concentration."—*Transactions of the Faraday Society*, Feb., 1911, p. 229. (H. A. W.)

THE RELATION BETWEEN CHEMICAL CONSTITUTION AND SOME PHYSICAL PROPERTIES. By Dr. SAMUEL SMILES. London: Longmans, Green & Co., 1910, pp. xiv+583, price 14s.

"The long-announced volume by Dr. Smiles on the relationship between physical properties and chemical constitution has now appeared, and forms a valuable addition to the well-known series of text-books edited by Sir William Ramsay. Certain branches of the subject, such as the relation between chemical constitution and optical rotation, have been dealt with in previous volumes of the series, and the present volume treats of the following subjects: Capillarity, viscosity, volume relations, specific heat, fusibility, boiling-point, refractive and dispersive power, absorption of light, fluorescence, magnetic rotatory power, anomalous electric absorption. In accordance with the size and price of the book, the subjects are dealt with in considerable detail, and numerous long tables of experimental data are quoted; in fact, the latter appear to contain nearly all the available trustworthy data. As might be anticipated from the author's own field of activity, the subject is treated almost exclusively from the organic side. A valuable feature of the book is the indication of many problems suitable for research.

In most respects the book is very good. The arrangement is logical and the language clear and well expressed, so that the author's expressed wish of making it accessible to students of an elementary knowledge of chemistry and general physics appears destined to be realised. The information is thoroughly up to date, and on the whole the book appears to represent the present position of each subject fairly well, although not equally well in each case. A rather more critical attitude towards views which have not yet met with fairly general acceptance would have added to the value of the book.

In a book covering such a wide field there are naturally many statements on which opinions may differ, but space will only admit of reference to one or two points. As lead and certain other metals are associated in the liquid condition but are monatomic in solution, Dr. Smiles considers (p. 27) that mercury exercises a dissociating effect on these metals. It is at least equally probable that the observed effect is due to the formation of a compound between the metal and mercury containing one atom of the metal in question. The method of detecting the formation of compounds in solution by viscosity measurements (pp. 77, 529) is by no means so generally accepted as the author's statements would lead one to suppose. Further, no indication is given that the Ramsay-Shields method of determining the extent of the molecular complexity of liquids from capillarity measurements is in many respects unsatisfactory, although this was pointed out by Nernst and by van der Waals long before the recent discussion at the Faraday Society.

The policy of quoting so many tables of experimental data and thus adding so greatly to the size of the book is a somewhat questionable one. Sufficient data must, of course, be given to make the subject intelligible, but it might be contended that the proper place for extended tables is Landolt-Bornstein or other tables covering a similar field. In this respect, and indeed with regard to the apportionment of space to subjects in general, there appears to be a want of uniformity in the different books of the series. The comparatively high price of the book, due partly to the causes just referred to, must to some extent militate against that wide circulation among students which the book thoroughly deserves on account of its many merits."—Transactions of the Faraday Society. Feb., 1911, p. 229. (H. A. W.)

Calvert, G. T. The Manufacture of Sulphate of Ammonia. 8vo, pp. 154. *J. Allan & Co.* Net 7s. 6d.

Humphrys, N. H. The Construction and Management of Small Gasworks. With a Section on Actual Costs and Capacity of Recently Erected Works by J. H. Brearley. Cr 8vo, pp. 268. *J. Allan & Co.* Net 7s. 6d.

Jamieson, A. A Text Book of Applied Mechanics and Mechanical Engineering. Vol. 3. 8th Edition. Thoroughly Revised. Cr 8vo, pp. 278. *C. Griffin.* 5s.

Park, J. A Text Book of Mining Geology. 3rd Edition. Revised and Enlarged. Cr 8vo, pp. 322. *C. Griffin.* Net 6s.

Abstracts of Patent Applications.

(C.) 358/11. Peter Burd Jagger. Improvements in collapsible cores for making hollow concrete posts and the like. 26.6.11.

This invention, which is intended for use in moulding or casting articles of concrete or the like, relates to collapsible cores of the kind wherein multiple rigid sections are employed in contra-distinction to cores comprising a section or sections which, in the distension or contraction of the core, must be more or less distorted. The object of the invention is the provision of a collapsible core comprising a number of rigid sections, which are so arranged that when expanded they present a surface that is exteriorly entirely complete; the sections being arranged to be moved positively and directly outwards and inwards through means operated by the lengthwise movement of a rod or rods actuating the sections, these means being such that the outside lap sections are distended and contracted with a slower movement than the inside lap sections, so as to cause the inside lap sections to press against the outside lap sections and produce a good joint.

(C.) 362/11. William Campbell Paterson. Improved agitation apparatus especially applicable for the treatment of pulp containing precious metals. 30.6.11.

This patent refers to a pulp agitating apparatus, the main features of which are a conical tank having an annular compartment in which the clear solution accumulates and whence it is withdrawn by a pump and used for agitating the contents of the tank by being injected through the cone of the tank and impinging in a pipe placed in the centre of the tank, thus causing the circulation of the charge.

(C.) 357/11. Peter Burd Jagger. Improvements in means for attaching rails to concrete sleepers. 26.6.11.

This application relates to a method of securing dog-spikes in concrete sleepers for rail tracks.

The method proposed is to insert a spiral coil, similar to a spring, in the hole cast in the concrete, and to drive into this a wooden ferule into which the dogspike is driven.

(C.) 367/11. Kurt William Otto Schweder. Improvements in means for dividing and distributing a stream of pulp or other mixtures of solids and liquid. 1.7.11.

This application refers to apparatus for dividing and distributing a stream of pulp or other mixtures of solids and liquid.

The claims describe a rotating circular receiver placed beneath the discharge of a feed launder; this receiver is divided into equal divisions by means of movable plates or sectors and is rotated by positive means applied from outside sources or by the reaction of the stream flow. The outlet from this receiver is by means of pipes connecting with the bottom of each division and discharging into annular and concentric launders.

(C.) 384/11. Graflich Schaffgotschsche Werke Gesellschaft mit beschraenkter Haftung (1), Valentin Stasch (2). An improved detent for checking run-away trolleys and the like. 8.7.11.

The invention has for its object the provision of a catch for preventing trucks, etc., from running away, and is intended to effect this for either direction of travel, but only comes into operation when the speed exceeds normal speeds.

The device comprises a detent lever operated by a rocking lever actuated by the passing truck. At normal speeds the rocking lever does not move far enough to operate the detent lever, but at higher speeds this operation takes place and the trucks are caught by the detent lever which is pivoted on a pin, this pin being either rigidly fixed or brought gradually to rest relatively to the rails by the action of an eccentric cam or some such device.

(C.) 395/11. Robert Okell. Improvements in combined gates and nozzle holders for classifying cones and the like. 13.7.11.

This invention relates to a combined gate and nozzle holder for use with classifying cones or other vessels in which it is necessary to provide means for rapidly changing a nozzle, and for closing the outlet of the vessel, either during the operation of changing the nozzle or for any other purpose.

The inventor provides a metal plate pivoted on one side of the outlet at the bottom of the cone, so that the plate lies in a horizontal plane and is capable of revolving beneath the outlet so as to open or close it as desired. The plate carries one or more nozzles and is made long enough to cover entirely the outlet and to bring the discharge nozzle outside the line of the outlet when the outlet is closed. The plate may be made large enough in the direction of motion to contain one or more spare nozzles of different sizes if desired. The upper surface of the plate is covered with leather or some other material so that when the plate is screwed up tightly, either in the open or closed nozzle position, a liquid tight joint is made.

(C.) 329/11. Valentin Jancee. Improvements in apparatus for transporting, manipulating, and fixing props, timbers, or the like in stopes or other workings of mines and the like. 10.6.11.

This invention has reference to an apparatus or means especially devised and intended for transporting, manipulating or fixing props or other heavy timber in stopes or other underground workings of mines and the like. The apparatus comprises a carriage or wheeled structure fitted with means for adjusting the carriage in a horizontal position, and provided with a winch by means of which the carriage can be dragged into position in the stope or any required position, and with a drum and pair of uprights for carrying pulleys round which passes the rope which serves as a sling for manipulating the prop of timber to be placed into position.

(C.) 346/11. William James Dunne. An improved liner or foundation for abrasive or corrosive treatment appliances. 19.6.11.

This invention relates to an improvement in liners or foundations for resisting wear and tear, either mechanical or chemical in abrasive or corrosive treatment appliances, and is especially intended for use in those parts of ore dressing mills, etc., where the abrasive or corrosive action is severe, such as in launders, conveyors, agitators, mortar boxes, ore passes or chutes, crushing mills, etc. The improved liner or foundation consists essentially of an admixture with cement of a silicious hard stone, such as crushed flint, chalcedony rhodonite or garnet.

(C.) 363/11. Walter Mackesy. Improvements in cams for operating the stamps of gravity stamp mill. 30.6.11.

This specification relates to improvements in cams for operating stamps of gravity stamps mills by which the shock when the cam strikes the tappet is reduced.

Instead of using a cam of which the working face is an ordinary involute, by which the stamp is lifted with a velocity equal to the velocity of the generating circle of the involute, the inventor describes a cam of which the working face is made with the radius of contact of a continuously varying quantity, being small at the beginning and large at the end of the lift.

Instead of using an ordinary circle for the generator of the involute of the face of the cam, Mr. Mackesy uses either a plain spiral as the generating curve or the arc of a circle which is eccentric to the cam shaft.

(C.) 487/10. Samuel Raybould. Improvements in guides for the stems of the stamps of stamp mills. 7.10.10.

This application relates to guides for stamp stems, and the claims include the following :—

(1) Metal guides in halves, the back portion of guide having recesses formed in the front portion of the bolt holes into which fit shoulders on the bolts for securing the back half of the guide to the guide girt independently of the front half. The front half of the guide is held in position by the ends of the bolts which project above the shoulders.

(2) The formation of a grease containing ring in the above described guide.

(3) In the above described guide the thinning of the back portion of the guide in the middle to reduce the weight.

(C.) 571/10. Sir Joseph Wilson Swan (1), James Alfred Kendall (2). Improvements in and apparatus for the manufacture of alkaline cyanides. 22.11.10.

This application refers to improvements in an apparatus for the manufacture of alkaline cyanides, the main features of which are the feeding of nitrogen and alkaline matter into the lower part of a reaction chamber or vessel containing carbon and arranged in connection with a condensing chamber or vessel so that the vapours produced during the process pass into it and are condensed therein,

Parts of the apparatus are placed in a furnace and must be made of metal which will withstand the action of the materials used for the production of the chemical, as well as the high temperature developed, which is necessary to obtain the reactions. Nickel with a small percentage of manganese or carbon is recommended for the construction of this particular part of the apparatus, which is placed in a furnace made of refractory material.

The necessary heat is obtained by the combustion of gas or oil fuel injected by air.

(C.) 340/11. Martin Deacon (1), William Gore (2), Palatine Engineering Co., Ltd. (3). Improvements in or relating to filtering apparatus. 17.6.11.

This invention relates to filtering apparatus and has special reference to the treatment of water, although it may be used for other liquids.

The applicants aim at automatically washing the sand or similar filtering medium without in any way delaying the filtering process. The filtering medium is transferred little by little from the bottom to the top by the water itself, which is caused to act ejector-fashion upon the medium at the lower end of the containing vessel carrying the sand, etc., up through a pipe to a box or receiver above the main body of the filtering medium, whence it is distributed over the top of the mass.

(C.) 488/10. John Fraser Price (1), Robert Craib (2). Improvements in means for separating liquids from crushed ore products. 7.10.10.

This application relates to improvements in means for separating liquids from crushed ore products or other solids.

A centrifugal separator is described, comprising an inverted conical vessel, with filtering devices attached to the inclined sides, and an annular launder arranged round the top to receive the thickened pulp. The machine is intended to work continuously and the degree of separation is regulated according to the speed at which the machine is rotated.

(C.) 326/11. Alexander Trifonoff (1), Daniel Gardner (2). Method of treating antimony and arsenic ores. 9.6.11.

This application describes a method of treating ores containing antimony and arsenic by melting with sodium sulphate, iron sulphide and carbon, with recuperation of the iron sulphide from the slag by leaching with water.

(C.) 300/11. Heinrich Ostwald. Improvements in ball mills. 26.5.11.

This invention provides for a mill drum with absolutely plain inner surface, in which balls are raised by magnetic force excited from the exterior ; when these have reached any desired height, the attractive force is automatically cut out and the balls are released and may fall down unhindered on the material they are intended to crush.

(C.) 400/10. Francis Allen (1), Gordon Travers Euston (2), Hugh McMenamin (3). Improvements in means for feeding ore to the stamps of stamp mills. 13.8.10.

This application refers to apparatus for eliminating the fine from the feed ore going to a stamp battery

The claims describe an inclined sieve which ac also as a feeder, the sieve being pivoted and vibrated by the action of the stamps so that the oversize is delivered to the mortar box and the undersize caught in a tray or chute and delivered to a launder. A spray of water assists the screening action and serves to carry the fine down the launder.

(C) 383/11. Milon James Trumble. Process and apparatus for the evaporation of liquid. 8.7.11.

This application describes a method of evaporating liquids by spreading them in a thin film over a substantially vertical wall and heating said wall.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 463/11. Peter Norman Nissen. Improvements in battery posts for stamp mills. 16.8.11.

(P.) 465/11. Edward Cuthbert Singer. A new arrangement for automatically raising, securing and closing windows. 17.8.11.

(P.) 466/11. William Arthur Caldecott. Improvements in means for introducing a gas below the surface of a liquid, applicable for use in the treatment of ore pulp and for other purposes. 17.8.11.

(P.) 467/11. Charles Friederich Eduard Oscar Schmitt. Improvements in feeding means for tubular and like grinding mills. 17.8.11.

(C.) 468/11. Theodore Fitzroy Fitzsimons. Venting and discharging valve for vessels containing liquids. 17.8.11.

(C.) 469/11. Dr. Werner Siebert. Process for the production of solutions of nitric pentoxide in monohydrate of nitric acid or concentrated nitric acid. 18.8.11.

(P.) 471/11. William Henry Hocking (1), Richard Thomas Paull (2). Improvements in bearing frames for rollers. 19.8.11.

(P.) 472/11. Frederick Victor William Swanton. An apparatus to produce power by means of weighted cantilevers with eccentric connections. 21.8.11.

(P.) 473/11. Joseph Barcham. Improvements relating to the manufacture of bricks, slabs, pipes, tiles and the like. 23.8.11.

(P.) 474/11. George Hubert Cox. Improvements in cam shaft pulleys and the like. 23.8.11.

(P.) 475/11. Donald McLachlan. Improvements in means for allaying the dust produced in rock drilling. 23.8.11.

(P.) 476/11. Ernst Heinrich Thiel. Improvements in lubricating means for rock drills. 24.8.11.

(C.) 477/11. Blaugas-Patent-Gesellschaft m.b.H. (1), Hermann Blau (2). Improvements in and relating to the manufacture of illuminating gas and apparatus therefor. 25.8.11.

(C.) 478/11. Charles Christopher Hausen. Improvements in rock drills. 25.8.11.

(C.) 479/11. Louis Beermann (1), Heinrich Balzar (2). Combined time and speed indicator and records for vehicles. 25.8.11.

(C.) 480/11. William Alfred Phillips. Method for obtaining oxides of nitrogen. 25.8.11.

(P.) 481/11. George Henry Seldon. Improvements in and relating to feed devices for hammer drills and means for regulating pressure on same. 25.8.11.

(P.) 482/11. Charles Jack David Symons. Principle and device for catching dust caused by drilling operations. 25.8.11.

(P.) 483/11. Kenneth Kemp. Sand filling improvement. 28.8.11.

(C.) 484/11. Paul Girod. Method of supplying electric furnaces with triphase currents. 29.8.11.

(C.) 485/11. Nicholas Henry Berrill. Divided cam for battery stamps and means for securing same on cam shaft. 31.8.11.

Changes of Address.

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Adamson, R., *l/o* Scotland; Susanna Mine, Gwanda, Rhodesia.

Baskett, E. G., *l/o* Benoni; Modderfontein "B" G. M. Co., Ltd., P. O. Box 14, Brakpan.

Bagshaw, J. F., *l/o* Warmbaths; Worcester Mine, P. O. Box 85, Barberton.

Bawden, E. R., *l/o* Cornwall: Ncheu, Boma, Nyasaland.

Blewett, E., *l/o* Selukwe: Northallerton Mine, *via* Queque, Rhodesia.

Candy, H. S., *l/o* Johannesburg; Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights.

Coll, D., *l/o* Cleveland; New West Bonanza G. M. Co., Ltd., Klerksdorp.

Cook, James, *l/o* Battlefields; Manager, Eldorado Mine, Rhodesia.

Cotter, A. J., *l/o* Johannesburg; P. O. Box 541, Salisbury, Rhodesia.

Dodd, Michael, *to* P. O. Box 3928, Johannesburg.

Durham, B. W., *l/o* Rhodesia; Lonely Reef G. M. Co., Ltd., *via* Inyati, Rhodesia.

Gray, J. Wensley, *l/o* West Africa: 7, Ashbrook Terrace, Leeson Park, Dublin, Ireland.

Hamill, J. C., *l/o* Germiston: Mabels Luck Mines, Syndicate, Gatooma, Rhodesia.

Harrison, M. J., *l/o* Germiston: Modderfontein "B" G. M. Co., Ltd., P. O. Box 14, Brakpan.

Haugh, *l/o* Pipeclay Creek; "Rhodesia," Francis Street, Bondi, New South Wales.

Heeley, A. W., *l/o* Macequece, P.E. Africa; Manager, Eagle "A" Mine, Shangani, Rhodesia.

Holman, B. W., *l/o* Johannesburg; Van Ryn G. M. Estate, Ltd., P. O. Box 22, Benoni.

Jameson, J. J. I., *l/o* London: Manager, Leeuwpoort Tin Mine, P. O. Rooiberg, *via* Warmbaths.

Johnson, J. H., *l/o* Germiston; French Bobs Mines, Ltd., P. O. Louws Creek, near Barberton.

McConnell, Thos., *to* Manager, Lydlinch Tribute, Moodies, Barberton.

Milligan, J. T., *l/o* Johannesburg; Rand Klip, Ltd., Benoni.

Nicholls, J. C., *l/o* Cal., U.S.A.; Mills Building, San Francisco, Cal., U.S.A.

Prout, W. M., *l/o* Knights; c/o S. C. Black, Esq., 1, Abel Road, Berea, Johannesburg.

Rusden, A. G., *to* Meyer & Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg.

Sale, G. G., *l/o* Johannesburg; Shagari P. O., *via* Gatooma, Rhodesia.

Summers, Ed., *l/o* Knights; Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg.

Thomas, Arthur, *l/o* Redruth; P. O. Box 67, Driefontein, East Rand.

Twining, D. H., *l/o* Penhalonga; East Rand Proprietary Mines, Ltd. (Angelo), East Rand.

White, S. R., *l/o* Mount Magnet; Bullfinch Proprietary Gold Mines, Bullfinch, West Australia.

Wilson, H. L., *l/o* Chihuahua; Las Dos Estrellas, El Oro, Mexico.

Yates, John, *l/o* Salisbury; New Club, Johannesburg.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

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Vol. XII. OCTOBER, 1911. No. 4.

Proceedings
AT
Ordinary General Meeting, October 21, 1911.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, October 21st, Mr. H. A. White (Vice-President) in the chair. There were also present :—

48 Members : Messrs. E. J. Laschinger, F. L. Bosqui, M. H. Coombe, K. L. Graham, J. Gray, T. Johnson, J. E. Thomas, F. W. Watson, Prof. J. A. Wilkinson, A. F. Crosse, A. McA. Johnston, Dr. J. Moir (Members of the Council), W. J. Abel, A. Avent, J. Q. Braidwood, J. M. Campbell, J. Chilton, F. W. Cindel, L. L. Ellis, R. Gascoyne, J. H. D. James, A. M. Kratz, G. A. Lawson, H. Lea, J. Lea, R. Lindsay, L. Marks, H. H. Morrell, M. T. Murray, P. N. Nissen, E. A. Osterloh, H. W. Pidsley, J. H. Rider, E. Roberts, G. A. Robertson, A. M. Sim, G. H. Smith, J. A. Smith, A. L. Spoor, C. T. Stephens, A. N. Stuart, W. A. Tester, A. Thomas, Prof. G. R. Thompson, K. Tonnesen, O. Tonnesen and H. Warren.

15 Associates and Students : Dr. J. L. Aymard, Messrs. G. Bulman, W, E. Burrows, J. Davidson, I. B. Dicks, W. Human, A. King, W. H. Pickburn, F. J. Pooler, R. A. Porter, H. Rusden, H. Stadler, E. H. Tamplin, G. W. Williams and S. A. Woolf.

22 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as published in the September *Journal*, were confirmed.

NEW MEMBERS.

Messrs. M. T. Murray and E. A. Osterloh were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been duly elected as follows :—

Evans, Richard Charles, New Kleinfontein Ltd., P. O. Box 2, Benoni. Chief Sampler, Kleinfontein Group Central Administration.

Horwood, C. D., Princess Estate & G. M. Co. Ltd., P. O. Box 114, Roodepoort. Cyanider.

Nissen, Peter Norman, Messrs. Head, Wrightson & Co. (S.A.) Ltd., 7, Standard Bank Chambers, Johannesburg. Mining Engineer.

Pieper, Gustav Adolph, Messrs. Erdmann & Sielcken, Batavia, Java. Mining Director.

Young, E. B., Mijnbouw Maatschappij Aequator, 11, Park St., Untermhaus, Gera-Reuss, Germany Engineering Director.

The Secretary notified that since the last meeting of the Society the following Associates and Students had been admitted by the Council:

As Associates—

Agnew, John Alexander, Messrs. Bewick, Moreing & Co., 55 MacDonald St., Kalgoorlie, W. Australia. Mining Engineer.

Arnot, Nigel Stanley, Simmer Deep Ltd., P. O. Box 108, Germiston. Metallurgical Student. (*Transfer from Student Roll.*)

Boonzaier, Lewis, Simmer Deep Ltd., 110 President St. Germiston. Cyanider.

Dicey, Edward Charles, Consort Mine, per private bag, Barberton. Mining Engineer.

Gilbert, Thomas Woodford, Rand Collieries Ltd., P. O. Box 13, Brakpan. (*Transfer from Student Roll.*)

Julius, Matthias Willem, Moera Aman, Benkoelen, Sumatra, Dutch East Indies. Government Mine Engineer.

McEwin, John Andrew, Waiuta, Grey Valley, New Zealand. Mill Superintendent.

Rees, Emrys Francis, Simmer Deep Ltd., P. O. Box 178, Germiston. Cyanider.

Robinson, Victor Edward, Simmer Deep Ltd., P. O. Box 76, Jeppestown. Cyanider.

West, George, Simmer Deep Ltd., P. O. Box 178, Germiston. Smelter.

As Student.—

Lorentz, Dirk Frederik Karel, Knight Central, Ltd., P. O. Box 91, Germiston. Reduction Works Learner.

General Business.

ZINC DUST AS A PRECIPITANT.

Mr. M. Thornton Murray (*Member*) : In view of experiments now being conducted with zinc dust as a precipitant for gold in cyanide solutions, the following note may be of interest.

Surface Exposed by Zinc Dust to Precipitating Action.—The sample of zinc dust examined was obtained by the courtesy of Mr. E. E. Johnson, and was representative of the material being used in experiments which he is carrying out.

The sample was first graded through screens having the following apertures :—

0·0197 in., 0·0100 in., 0·0060 in., and 0·0030 in.

Particles passing one screen (say, 0·0197 in.), but remaining on the next (say, 0·0100 in.) were considered to have a diameter equal to the mean of these two (say, 0·0149 in.), while those passing the final screen were examined under the microscope, and an average diameter of particle ascertained as accurately as possible. This last operation might, of course, lead to error, since the personal element is liable to intrude, but by taking (separately) the opinions of a number of different observers, this error was, as far as possible, eliminated.

The results were then tabulated, as follows :—

Particles between 0·0197 in. and 0·0100 in. (say, 0·0149 in. diameter) equal (by weight) 0·16% of the whole.

Particles between 0·0100 in. and 0·0060 in. (say, 0·0080 in. diameter) equal (by weight) 2·42% of the whole.

Particles between 0·0060 in. and 0·0030 in. (say, 0·0045 in. diameter) equal (by weight) 2·88% of the whole.

Particles less than 0·0030 in. (microscopically examined, and shown to have a mean diameter of 0·00085 in.) 94·54% of the whole. From these data the average diameter of particle was calculated as 0·0011506 in.

It may be noted here that none of the dust remained on the 0·0197 in. aperture screen, while the microscope showed that the zinc itself was in the form of almost perfect spheres.

A sphere of the diameter given would expose a superficial area of $4·161 \times 10^{-6}$ sq. in., while its volume would be equal to $7·979 \times 10^{-10}$ cub. in. Whence it can be shewn that 1 lb. of the material exposes 144·8 sq. ft. of precipitating surface (taking the specific gravity of condensed zinc at 6·92). The zinc dust used was guaranteed 90% metallic zinc, so that the available precipitating surface of one pound of this material would be about 130·32 sq. ft. This compares favourably with the filiform zinc at present in use, giving a little more than three times the surface, weight for weight.

The Nature of the Surface Exposed.—It has been contended, by critics of the zinc dust precipitation process, that the particles are likely to be covered with a film of zinc oxide. In that the condensation of the zinc, in the smelting works, takes place in an atmosphere composed largely of carbon monoxide, this criticism appears somewhat baseless. Indeed an examination of the particles under the microscope shows that each particle of zinc has an exceedingly bright metallic lustre, even after several weeks exposure to the air.

Zinc oxide and other impurities can be plainly discerned, and discriminated, under the microscope. They occur quite separate from the metal, as irregular aggregations of very fine particles. During screening, most of these aggregations remains on the 0·0100 in. aperture screen.

The Chairman : This exceedingly fine powder only exposes a surface three times that of the zinc shaving generally in use. Those who are familiar with the zinc dust precipitation, know that the time of contact given is very short compared with contact in the extractor boxes but most excellent results are obtained, so evidently there is another factor involved which has not yet been ascertained.

The Secretary read the following communication from the Chief Inspector of Explosives :—

EXAMPLE OF DANGERS IN MINING.

Major A. B. Denne (*Chief Inspector of Explosives*) *:* The piece of rock shown in the photograph was brought by the Inspector of Mines from the Meyer and Charlton Gold Mine where it had been found on the sorting belt. It weighed 2½ lbs.

The explosives in the hole were about ¾ of a plug of gelignite ⅞in., one No. 6 detonator, and a piece of fuse which was burned through.

There is no clue to the date of the explosives, beyond that the wrapper of the cartridge had on it marks showing that the gelignite must have been made at some time since 1909.

The gelignite cartridge was freshly broken. The capsule of the detonator was eaten through by acid and broke in my hand. The top of the fulminate of mercury was coated with green.

There was nothing to show why the fuse had failed to ignite the fulminate, but this may have been due to

(*a*). Failure to cut the end of the fuse before insertion ;
(*b*). Damp detonator ;
(*c*). Sawdust in detonator.

The miner was evidently a careful man, and had tied the wrapper of his primer cartridge round the fuse as prescribed !

The broken bore-hole (also $\frac{7}{8}$in.) would no where near intersect the misfired hole, as the photograph would lead one to imagine.

NOTES ON HIGH DUTY GRAVITY STAMP MILLS.

By Peter N. Nissen.

Although the gravity stamp mill has been in use for generations, the writer ventures to state that it is not as generally understood and consequently not as thoroughly appreciated as its merits warrant. Possibly this is due to the great simplicity and seeming crudeness of its construction and the tendency to regard anything simple as unimportant and unworthy of serious study. However, the most interesting facts are often concealed in the simplest forms, and it is safe to say that nothing is of too little importance to repay investigation. This is especially so with the gravity stamp mill, for in spite of its simplicity and seeming crudeness, it is susceptible of the most delicate adjustment, its efficiency depending in a great measure upon the skilled attention devoted to it.

It is interesting at this stage to find as a result of reliable tests that the well designed gravity stamp has proved to be the best machine for crushing, within the prescribed limits, as will be shewn in these notes.

It has been demonstrated in many fields that in the items of power, labour, wear and tear, the cost of running stamp mills is lower than for any of the other known crushing machines. In addition, the stamp mill covers a wider field in its work than competitive machines, since the size of feed for stamp mills ranges from over 3 in. to a fine sand ; therefore its position as a crushing machine is unique. Every practical man knows that it is almost impossible to maintain the product from ore breakers at the required size for more than a short period. This fact adversely effects work by rolls and Chilian mills, the feed for which must remain within the size governed by the diameter of the rolls or edge runners if they are to perform efficient work. Tube mills are limited in their feed to a size not exceeding $\frac{3}{8}$ in., while ball mills although not so limited to size of feed, cannot possibly compete with stamp mills in view of the enormous wear and tear on balls and liners in crushing the ore of the Witwatersrand.

The history and development of this excellent machine has been ably recorded by many others ; it is therefore not the intention of the writer to attempt any further elucidation of the subject, so far as the development of the multiple stamp mortar battery, having units of five stamps, is concerned.

It is generally agreed that, in spite of the length of time this machine has been in use and while many improvements have been made in an effort to get the best results, it still remains a very imperfect machine.

Much thought has been given to the proper design for the rectangular mortar for five stamps for different conditions of service, until we have passed in review many types, all of which adhere to one general principle and differ only in detail.

The principle of having five stamps in one mortar has rarely been assailed, but it rather seems to have been taken for granted that it could not be improved upon. That this has been the case with such an important and useful machine, must always remain a surprising fact.

Since the function of the stamp is one of crushing the greatest quantity of ore for a given expenditure of power, it follows that to get the highest efficiency from each blow of every stamp, the conditions of feed are most important. Regardless of the order of drop and in spite of the different theories advanced, it is evident that all the stamps in a multiple stamp mortar are not doing the same amount of useful work; it must be recognised that the uncrushed ore in the mortar is not distributed evenly over the dies, so that the coarser pieces, and therefore the pieces first struck are not always arranged near the centre axis of the falling body. This lack of control of the feed is, in the writer's opinion, largely responsible for broken shoe necks, broken stems, worn guides and uneven wear of dies. The uncertainty of having the uncrushed ore, particularly the large pieces, in the centre of the die face renders it impossible to expect really efficient work from the stamp. Of equal importance with a controlled feed is the positive discharge of the crushed ore, when reduced to the desired size. The rectangular mortar having a screen on one of its longer sides, does not suggest a logical method of dealing with the problem. By the order of drop necessary to force the ore from one die to another, waves of motion are set up from end to end of the mortar, and only that material adjacent to the screen can be discharged. Double discharge mortars might have improved these conditions, were they not still more difficult to feed, owing to increased width and the necessity of having to introduce the ore higher in the box, besides other faults of design.

It is not within the limits of commercial possibility to make the multiple stamp mortar of sufficient weight in itself to secure the necessary inertia for effective crushing. It is therefore important that it should be securely fastened to a foundation having great weight and solidity. In a mortar with the blows of the several stamps acting on different sections of the casting with great rapidity, the greater the length the more difficult it is to hold securely on the foundation. Regardless of the skill exercised in fitting the mortar to the foundation, a certain rocking tendency exists, due to its being struck at alternate ends by the falling stamp. It is difficult to keep the foundation bolts always tight; in any case they are liable to stretch and finally break, due to the strain. Certain internal stresses exist in the mortar casting, due to its design, and this eventually breaks by the repeated blows.

Coming to stamps weighing above 1,250 lb. and up to 2,000 lb. each, we find the same diameter shoes and dies are used in all cases, so that the mortar need be no longer. Attention to this fact has been drawn by several writers, and rightly so, for there must be a correct relation between the area of the crushing members and the weight of stamp, on rock of a certain hardness and friability, to get the highest efficiency from the blow. Given the same length of mortar, to increase the weight of the stamp, it becomes necessary to use longer heads and heavier tappets. It is apparent that this is not the most desirable way of obtaining the object in view, because it raises the centre of gravity and increases the distance between the face of the die and the lower guide. A more rational way of accomplishing this object would be to increase the diameter of the head. This however would demand a longer and heavier mortar to accommodate the larger stamps and thus exaggerate the rocking tendency, besides increasing the distance between cam shaft bearings, which is undesirable. It is well-known that there are limitations to the diameter of cam shafts, since the larger the shaft, the greater the velocity of the cam at the point of engaging the tappet, due to the greater distance between centre of cam shaft and centre of stem.

It has been proved that heavy stamps are better crushing machines than the lighter stamps, within certain limits, therefore since the heavy stamp is desirable, and as ordinary five-stamp battery construction does not lend itself to use with the very heavy stamp, a new design had to be evolved. To render the use of the heavy stamp possible, different principles of design must be adopted, so as to harmonize the desired greater crushing capacity with sound mechanical construction. The Nissen stamp mill has been designed with this object in view.

This stamp mill has been developed on the unit principle, since it offers a remedy for many of the disadvantages found in the multiple stamp mortar system.

The Nissen mortar, if viewed in a vertical cross section, is very similar to the modern five-

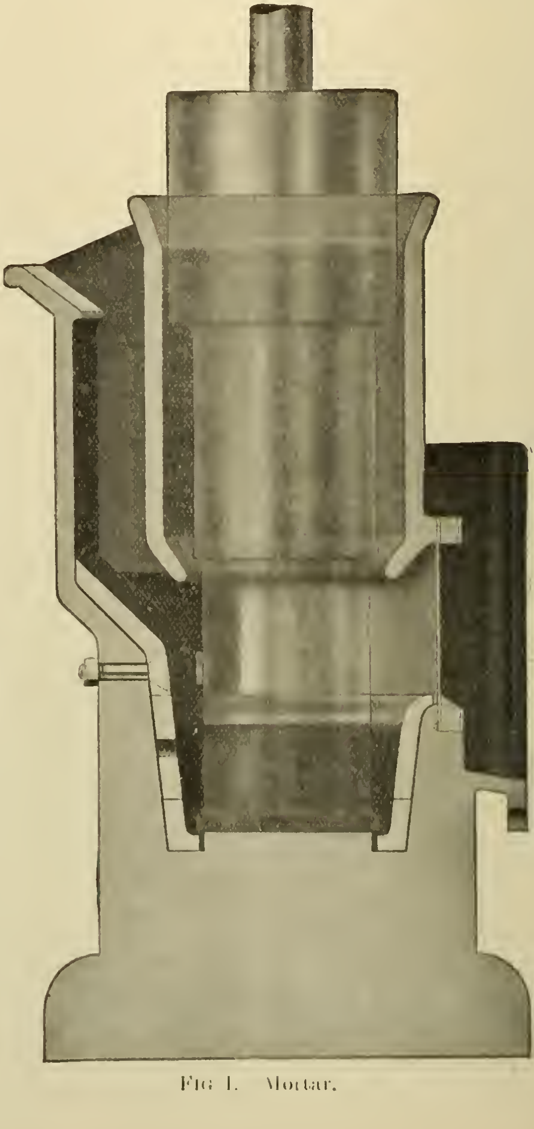

FIG. 1. Mortar.

stamp mortar, the difference being that it is circular and one mortar is provided for each stamp.

The working height of discharge is much the same as for five stamp mortars and the screen is practically the same distance from the stamp, but owing to the mortar being circular, this screen can be extended around the stamp for the greater part of the circumference of the mortar, so that it is of equal distance from the stamp throughout its full length. A screen of much larger area per stamp is thus provided, which is always at right angles to the splash of the pulp and is therefore fully effective.

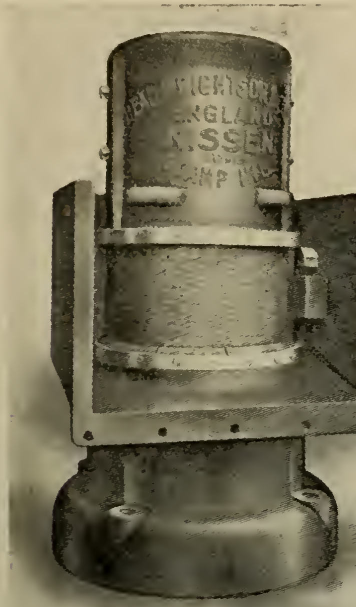

Fig. II.

At each blow of the stamp the pulp is forced radially against the screen, so that all particles sufficiently reduced are discharged. The stamp is therefore dropping on uncrushed ore, thereby doing useful work at every blow. Owing to the mortar being circular, each time the stamp is raised, all the material in the mortar flushes back to the centre to be struck by the falling stamp.

It therefore follows that the uncrushed ore is automatically returned to the crushing zone, so that the best conditions of feed are maintained. Each stamp is supplied with a specially designed feeder, which, as it is controlled by the one stamp it supplies, yields, as already pointed out, the closely adjustable feed which is so desirable.

By the use of a circular mortar, many mechanical advantages appear in the entire machine, all of which are of great importance. Any machine to be of commercial value must operate economically, when considered from every point of view.

To facilitate access to the inside of the mortar and to allow for easy removal of shoes, dies and heads, also to reduce the height of the battery structure, the mortars are designed with easily removable fronts, which are secured in a simple manner to the main casting.

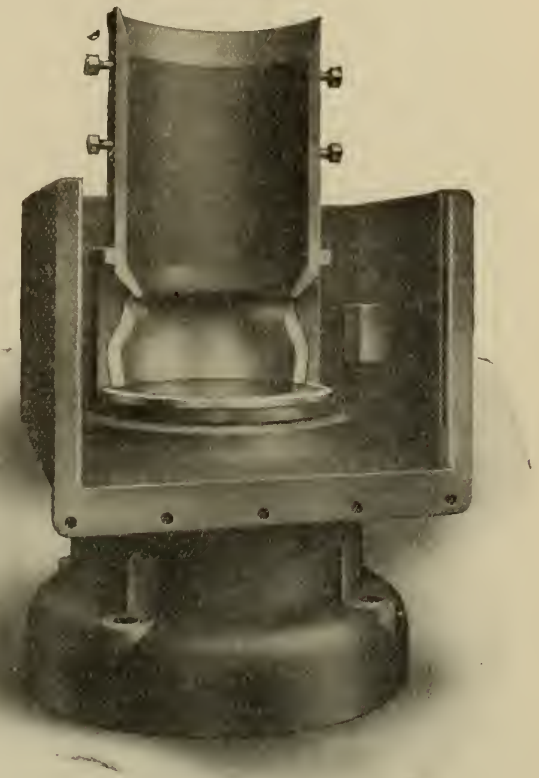

Fig. III.

The entire crushing zone of the mortar is protected with steel liners that effectively protect the casting from wear. With the Nissen mortars the screens are not tacked to wooden frames, but are secured to the casting by circular bands drawn up by two keys whereby the destruction of screen frames is avoided.

Although the mortar is comparatively light, it is yet very rigid on its foundation, owing to the blow being always received in its vertical axis. One of the special features of the Nissen mortar is that it can be cast with less fear of shrinkage strains, which are so destructive to ordinary five-stamp mortars. The foundation bolts are not subjected to undue strains, and therefore give no trouble. Furthermore, should a serious explosion take place in the mortar, caused by dynamite, which accidentally may get into the mortar with the ore, so that the casting is destroyed, the expense of replacing one mortar is naturally less than would be the case for a large five-stamp mortar casting. Another advantage of the unit principle is the more continuous operation of the entire plant, as each stamp can be put out of commission independently, so that a much greater percentage of total time can be employed in useful work with a correspondingly greater total duty, than is the case with the multiple stamp system. An incidental advantage is the comparatively smaller loss in duty due to less interruption for classifiers and tube mills since only individual stamps need be stopped. It has been found possible in large installations of Nissen stamps to obtain an actual operation of over 95% of the total possible time. The same number of parts per stamp make up the falling weight as are in common use in stamp batteries, consisting of shoe, head, stem and tappet. The design of all these members is somewhat different to those used in ordinary stamp batteries, owing to the greater amount of room available by using the unit principle of construction. All these parts are made larger in diameter, therefore shorter, thus greater strength is secured and the centre of gravity is lowered. The shoe is 10 in. in diameter for a stamp of 2,000 lb. falling weight, this size having been adopted as a result of experience. A larger shank to the shoe is rendered possible by the larger diameter head, and this ensures greater strength and less breakage.

The dies are of the round base type and are provided with a suitable recess in the bottom, which fits over a projecting boss cast in the mortar; thus the dies are always kept central. A special feature of the Nissen stamp is the even and flat wear of these dies, which results from the return wash of the ore to the centre of the mortar with each stroke of the stamp, and from the increased number of rotations of the stamp due to wider cam and tappet faces. The advantage of even wear of the die can readily be appreciated; not only is the crushing capacity of each blow greater, but a less amount of metal need be discarded for the die can be practically worn out.

Special attention has been given to the head, as in the writer's opinion, this is the most important member of the gravity stamp. It is, of course, desirable to have the centre of gravity of the stamp as low as possible, and this can best be accomplished by using a heavy head. In the five-stamp battery construction, owing to the centres of the stamps being closely defined, this increased weight can only be secured by long heads which necessitate the lower guides being too far removed from the crushing point. With Nissen stamps the heads can be made very large in diameter, consequently short. By the large diameter more strength is secured in the lower section where the shoe joins the head, to prevent splitting, whilst the upper section is much larger to give weight, and the resulting shoulder acting in conjunction with the special design of the mortar forms an efficient baffle to prevent splashing, yet allowing the head to pass out of the mortar, so that compensating weights can be easily applied and the greatest capacity maintained.

As a result of the design of the mortar, wear and tear on the the lower section of the head has been practically eliminated. The combined weight of the shoe and head is 57½% of the total weight of a 2,000 lb. stamp, therefore the greatest part of the dropping weight is concentrated in the head and not in the stem, as is the case in many of the ordinary heavy stamps. A stem 5 in. diameter by 10 ft. long of double fagotted iron is used and can be made of greater diameter, if necessary, as there are no limitations, as is the case with five-stamp mortars, the increased size of the head allowing of this very desirable feature of design. With the increased diameter of stem, low centre of gravity, less distance between face of die and centre of lower guides, light tappets and even wear of dies, stem breakages are practically eliminated.

The tappet is made as light as possible, consistent with the weight of the stamp, and owing to there being no practical limit to the diameter it is short and provides a broad lifting face, consequently a more rapid rotation of the stamp is obtained. With this larger diameter, stiffer keys, that will securely hold the tappet and stand driving without damage can be used. The writer is of the opinion that a heavy tappet is wrong in principle, since it is more difficult to set fast and too much weight is applied in the wrong place. In five-stamp battery construction the diameter of the tappet is regulated by the distance between the stamp centres, and as the stems must be as large as possible, the keys are necessarily light and have too much spring to stand driving, therefore are easily destroyed by the constant hammering they are subjected to.

Self-tightening chrome steel cams having broad faces and large hubs are used, and have given gratifying results.

The importance of good guides is thoroughly understood and appreciated, as is evidenced by the great variety of both wood and iron ones that have been produced. Iron guides are used with Nissen stamps, and are made similar to an ordinary shaft bearing, each one being securely fastened to the girths by four large bolts. The caps for these guides are made adjustable so as to prevent excessive wear due to slackness. Each guide is 14 in. long and they have proved satisfactory in actual use.

The cam shaft used with Nissen stamps is 6½ in. diameter, and is made of double fagotted iron. This shaft is of equal size for four stamps as is commonly used with ten stamps of about 1,500 lb. weight each. By using right and left hand cams with each two stamps, they are arranged close to the bearings, which must be recognised as a decided advantage. Since the shaft in lifting four Nissen stamps transmits about half the power required for ten stamps in multiple mortar construction, and is subjected to 40% of the shocks, it has much longer life, thereby reducing the maintenance charges to a considerable extent.

The cam shaft bearings are of the cast iron bored type, made with very heavy bases; they are also provided with massive caps of a special type. These caps have no lugs to break off, but are held in position by a substantial device, as shewn in Fig. IV.

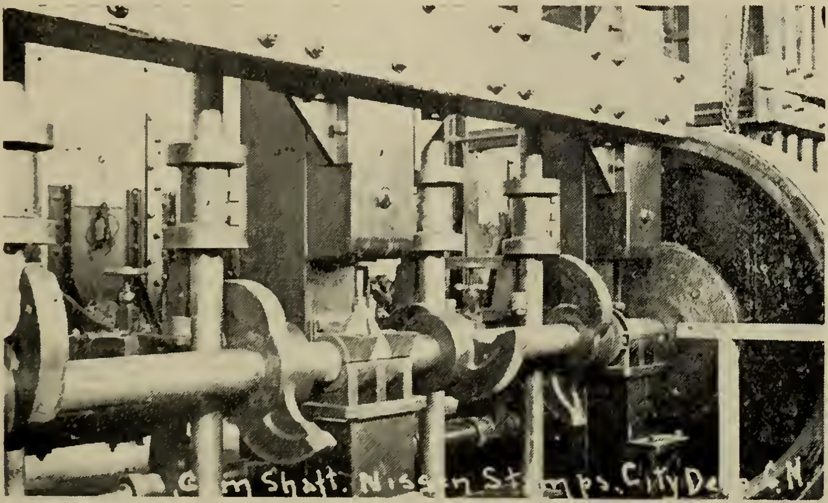

Fig. IV.

A special type composite battery post is used with Nissen stamp batteries as illustrated in Fig. V. This consists of a 14 in. × 6 in. × 42 lb. steel joist bolted between wood members. All bolts used in securing girths and cam shaft boxes to posts pass through the wood and steel, thus securely binding the members together; the advantage of this steel core being that the post is not weakened where the front section is cut away to receive the cam shaft box, since this steel joist extends the full length. By using this system, battery posts can be built up of deal where heavy timbers are not available and in all cases the steel insertion is sufficiently strong and rigid, and with the wood portion to absorb the vibration, a very serviceable and strong post is obtained.

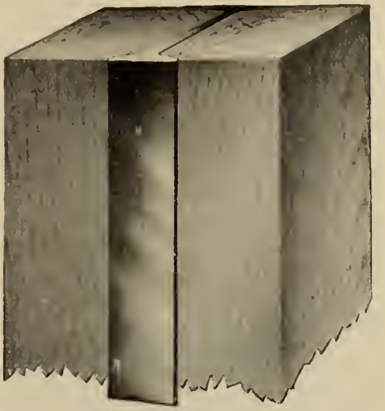

Fig. V.

A feeder of a Challenge type is used with each stamp and is rope-driven from the tappet. The driving gear is self adjusting and can be depended upon, so that it requires very little attention. The feeder is usually attached direct to the bins and a chute is provided to carry the ore to the mortar. This chute catches all particles that drop from the feeder, so that no shovelling or sweeping is necessary. The water feed to the mortars is introduced at the bottom of this chute, so that the ore and water enter the mortar together. An important feature resulting from the unit principle as applied in the Nissen stamp, is the advantage that a much lower and therefore more solid battery frame construction can be used. By increasing the diameter of the different members and decreasing their length, a battery frame having a total height of about 13 ft. above the foundation is possible.

Figures VI. and VII. illustrate the latest arrangement of the Nissen stamp battery construction showing 5 in. stem 10 ft. long and king posts 12 ft. 6 in. long. It will be noted that the centre of the cam shaft is but 9 ft. 6 in. above top of foundation. By this construction a very rigid battery frame is obtained, the centre of gravity of which is very low, and a reduction in the height of the bin and the building is obtained, which decreases the cost of installation. The bin structure can be made far lighter with the same amount of storage, while the retaining walls and underframing are also much less expensive. Where locomotives and heavy trains pass over the bins, this is an item meriting serious consideration.

The Nissen stamp for the same tonnage requires about 30% less power than the ordinary stamp, therefore smaller motor, shafting, pulleys and belting are needed. The greater capacity of the Nissen stamp reduces the total weight of iron and steel in the battery for a given tonnage and this may amount to as much as 40%.

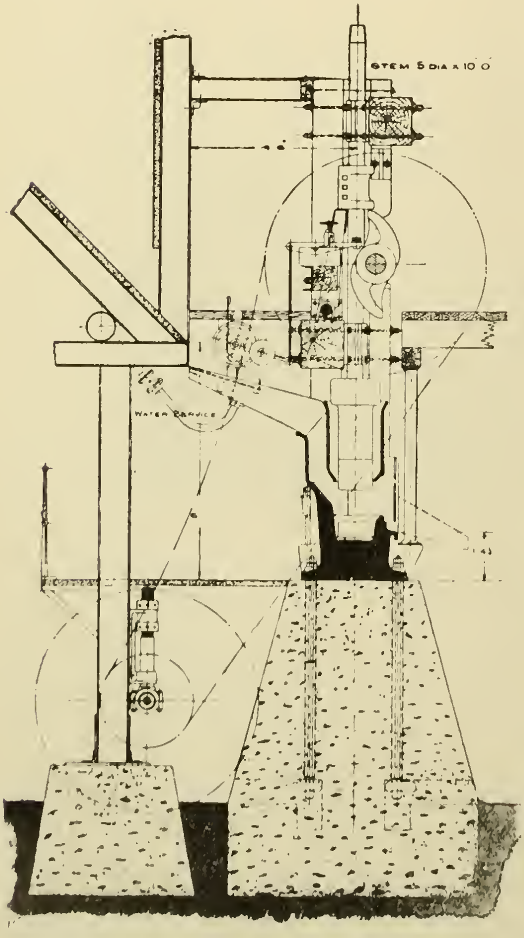

Fig. VI.

Not unlike other machines, the Nissen stamp has had a gradual development. Starting with a dropping weight of 1,100 lb., the weight has been gradually increased until a stamp weighing 2,250 lb. has been successfully operated, with a possibility of a still heavier stamp being used. What the maximum weight will be, the writer is not prepared to state at present.

Exhaustive tests have been carried out with four Nissen stamps of 2,000 lb. dropping weight at the reduction works of the City Deep, Limited, in connection with tests made simultaneously with a ten stamp battery of City Deep type, and the result of these tests are embodied in table No. I. It will be seen that five competitive tests were made, while with test No. 4 the City Deep stamps were not operated. The running weights of the Nissen stamps varied from 1,927 lb. in test No. 2 to 2,245 lb. in test No. 3. The number of drops for the Nissen stamps was 103 per minute, and the height of drop in all tests but No. 3 was 8½ in.; in the latter it was 8¾ in. The height of discharge varied from 2 in. to 2⅝ in. The screen used was equivalent to nine holes per square inch, with the exception of tests No. 3 and No. 6 when screening with ⅜ in. apertures was used.

The stamp duty for the nine mesh screen varied from 24·47 tons to 30·85 tons for Nissen stamps and from 18·26 tons to 20·95 tons for City Deep stamps; the weight of the latter ranging from 1,775 lb. each to 1,863 lb. each. The duty per stamp per day when using ⅜ in. screen varied from 36·69 to 37·74 tons for Nissen stamps and for City Deep stamps 22·72 to 24·34 tons.

The power consumption per ton of ore crushed varied from 2·7 H.P. hours to 4 H.P. hours for Nissen stamps and from 4·45 H.P. hours to 5·5 H.P. hours for City Deep stamps. The quantity of ore crushed per pound of falling weight varied from 25·35 lb. to 37·91 lb. for Nissen stamps and from 19·6 to 27·45 lb. for City Deep stamps.

The wear and tear of shoes and dies is slightly in favour of the Nissen stamp.

Analysing the grade of battery feed and discharge it will be seen that although the Nissen stamp shows correspondingly greater duty per pound of falling weight than the City Deep stamp and although the grade of the discharge was approximately the same, yet the feed was in all cases, excepting test No. 6 very much coarser in the case of the Nissen stamps.

Further analysis of the battery discharge of test No. 2 for the two different types of stamp, by Stadler's method, proved that the relative

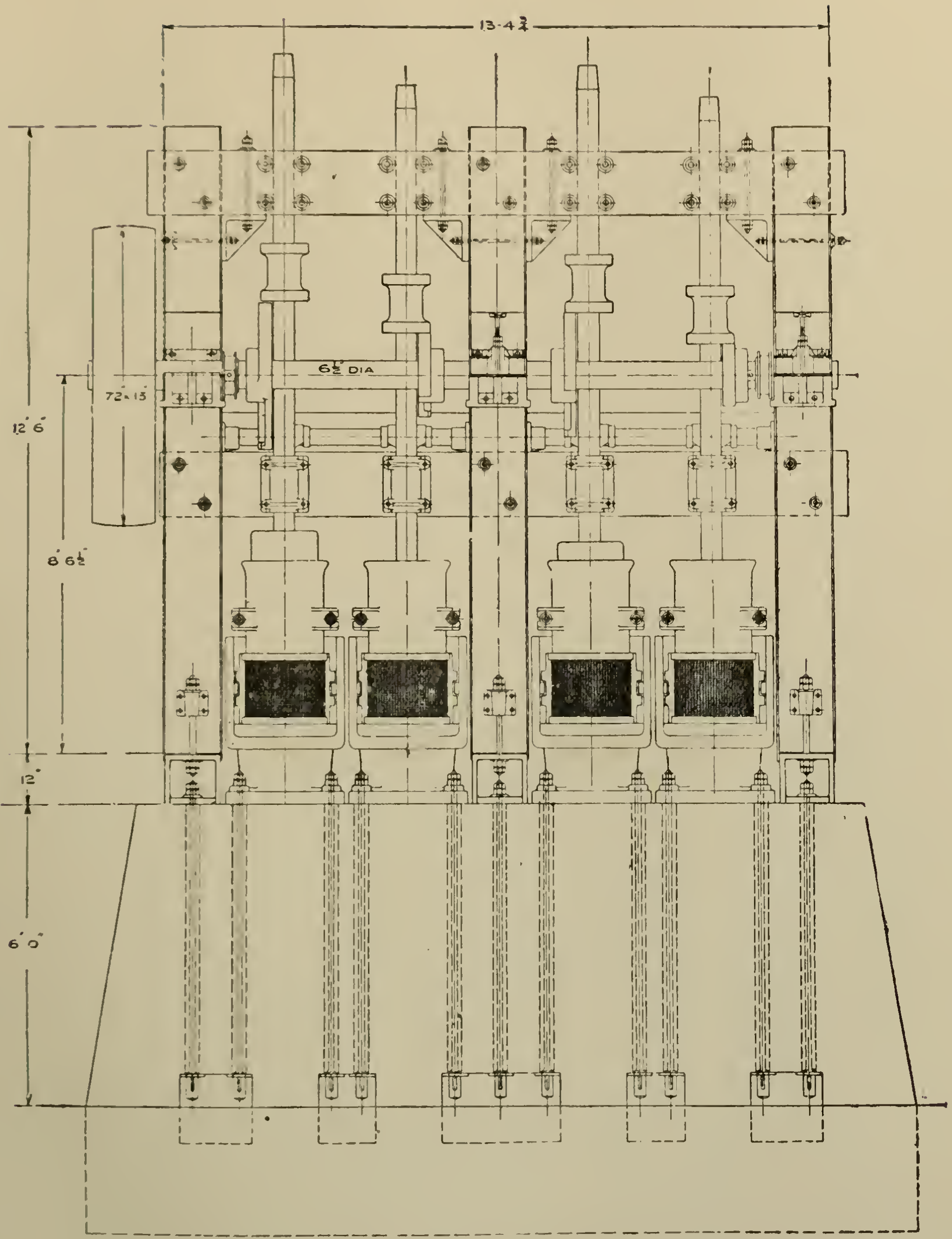

Fig. VII.

TABLE No. I.—SUMMARY OF COMPARATIVE TESTS BETWEEN NISSEN AND "CITY DEEP" STAMPS.

Test Number and Date -	1 June 23, 1911		2 July 5, 1911		3 July 17, 1911		4 July 26, 1911		5 Sept. 11, 1911		6 Sept. 16, 1911	
No. of Stamps and kind -	Four Nissen.	Ten City Deep.	Four Nissen.	Ten City Deep.	Four Nissen.	Ten City Deep.	Four Nissen.	Ten City Deep.	Four Nissen.	Ten City Deep.	Four Nissen.	Ten City Deep.
Weight of Stamps in lb.	1932	1863	1927	1859	2245	1855	2237		1993	1775	1991	1773
No. of Drops per min. -	103	100	103	100	103	100	103		103	100	103	100
Height of Drop in inches	8½	8½	8½	8½	8¾	8¾	8½		8½	8½	8½	8½
Height of Discharge -	2⅝	2¼	2	2⅜	2	2½	2	NO COMPETITIVE TEST	2¼	2¼	2¼	2¼
Class of Screen - -	Tyler	T.C.	Tyler	T.C.	Sq. m.	14(·08)	T.T.C		Sq. m.	wire	Sq. m.	wire
Mesh of Screen - -	9	9	9	9	⅜″	⅜″	9		9	9	⅜″	⅜″
Width Aperture in inches	·205	·205	·205	·205	·375	·375	·23		·277	·277	·375	·375
Length Aperture ,,	·536	·536	·536	·536	·375	·375	·85		·277	·277	·375	·375
Dry Ore crushed tons -	180·86	192·75	180·25	192·12	193·88	206·66	191·7		185·32	194·53	185·7	191·2
Total Run in hours -	44·33	25·33	39·0	23·166	31·70	21·83	37·28		37·29	22·28	29·52	18·85
Tons per stamp per 24 hr.	24·47	18·26	27·73	19 90	36·69	22·72	30·85		29·81	20·95	37·74	24·34
H.P. per stamp* - -	4·09	4·21	4·09	4·21	4 95	4·21	4·95		4·23	4·65	4·23	4·65
H.P. hours per ton of ore crushed* - - -	4·0	5·5	3·56	5·06	3·24	4·45	3·8		3·41	5·32	2·71	4·56
Lbs. ore crushed per lb. falling wt. per day -	25·33	19·6	28·78	21·412	32·68	24·49	27·58		29·91	23·6	37·91	27·45
Ratio water, ore wt. -	5·4 : 1	2·7 : 1	4·2 : 1	2·4 : 1	3·1 : 1	2·15 : 1	3·2 : 1		3·3 : 1	3·3 : 1	3·03 : 1	3·26 : 1
Cons. metal in shoes in lb. per ton crushed -	·129	·145	·139	·24	·198	·187	·122		·147	—	·112	·08

*Excluding Belt, Shafting and Motor losses.

GRADING ANALYSIS.

FEED	1		2		3		4		5		6	
+2 %	36·9	12·1	14·96	12·4	26·9	18·2	52·3		39·0	14·1	23·7	27·1
+1½	18·6	14·4	14·50	11·35	18·75	16·0	14·5		19·7	18·0	20·9	17·1
+½	15·1	20·2	20·40	19·35	19·3	18·15	10·4		15·0	21·3	18·8	19·1
+¼	11·7	12·7	16·95	17·70	13·05	16·25	9·4		9·3	16·0	12·7	12·6
−¼	27·5	37·9	33·20	39·20	22·0	31·4	13·4		17·0	30·6	23·9	24·1
DISCHARGE												
+60	65·2	63·4	64·9	63·4	67·24	69·33	67·3		66·6	66·5	70·7	70·7
+90	8·3	9·1	8·46	8·70	8·62	9 09	8·75		9·9	10·2	10·1	10·3
−90	9·7	12·0	8·46	9·10	10 84	9·82	8·55		8·8	10·9	10·2	9·2
−200	16·7	10·7	18·10	19·5	13·3	11·76	15·4		14·7	12·4	9·0	9·8

CENTRAL MINING & INVESTMENT CORPORATION, LTD.,

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Consulting Metallurgist.

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mechanical efficiency for Nissen stamps is 0·25 and 0·19 for the City Deep stamp, or a difference of 35% in favour of the Nissen stamp. This may be seen on reference to Table II.

TABLE II.

Mesh.		Nissen Stamps. Grading per cent.		Nissen Stamps. Mech. Val. E.U.	City Deep Stamps. Grading per cent.		City Deep Stamps. Mech. Val. E.U.
+5	9	19·80	66·54	1·782	14·8	64·78	1·332
8	11	10·75		1·183	11·28		1·241
12	13	9·30		1·209	10·59		1·377
20	15	10·04		1·506	10·57		1·586
30	17	7·92		1·346	8·50		1·445
50	19	8·73		1·659	8·86		1·683
80	21	10·17	10·17	2·136	11·48	11·48	2·411
120 200	24	4·71	23·29	1·130	5·38	23·74	1·291
−200	28	18·58		5·202	18·36		5·141
		100%		17·153	100%		17·507

Duty in 24 hr.	27·72 tons	19 9 tons
Wt. of stamp	1,927 lb.	1,860 lb.

Relative Mechanical Efficiency.

$$\frac{7{\cdot}153 \text{ E.U.} \times 27{\cdot}72}{1{\cdot}927} = 0{\cdot}25 \qquad \frac{17{\cdot}507 \text{ E.U.} \times 19{\cdot}9}{1{\cdot}860} = 0{\cdot}19$$

If wt. assumed to represent h.p. consumed.

To better illustrate the capabilitity of the Nissen stamp and the advantages to be derived by its installation as compared with the ordinary type of stamp, a comparison is made between two batteries having a capacity of 40,000 tons per month or 1,330 tons per day. Assuming that the ordinary battery is capable of running 90% of the total time the gross capacity would then be 1,480 tons per day of 24 hours. It is safe to assume a nett running time of 95% for Nissen stamps, which, in this case, would give a gross capacity 1,400 tons per day of 24 hours. On the basis of crushing through a 9 mesh screen to yield a product as follows :—

65% + 60
8·5% + 90
8·5% + 200
18% − 200

and using a stamp of 2,000 lb. falling weight, a duty of over 29 tons can be obtained with the Nissen stamp, so that 48 of these stamps would be ample for the work. To crush the 1,480 tons per day of 24 hours to this grading with the ordinary stamp of about 1,400 falling weight, a duty of under 15 tons may be expected through a 9 mesh screen thus 100 of these stamps would be required for this capacity. It therefore remains to compare a battery of 48 Nissen stamps of 2,000 lb. each with a battery of 100 ordinary stamps of 1,400 lb. each, a comparative statement of which will be found in Table No. III.

Capital Cost.—Assuming that these batteries will probably be erected in a single row, a construction which hereafter will be the standard in these fields, the total length of the mill building will be 196 ft. for Nissen stamps and 194 ft. for ordinary stamps. It will be seen therefore that the buildings and ore bins are of approximately the same length in both cases, and as the ore crushed is identical, both types of batteries being run under the same conditions, as regards the supply of ore to the mill, the cost of the mill ore bins will be approximately the same, however, owing to the greater height needed for the ordinary type of battery there will be a difference in favour of the Nissen stamp installation. The same remarks also apply to the building. This has been allowed for in making the comparative estimates shown separately in appendix No. I and embodied in table No. III. It appears that while the 100 stamp battery of the ordinary type will cost £23,000, the Nissen battery of equivalent capacity will cost £20,000, showing a saving

Table No. III,
Comparative Statement of Capacity, Power Demand, Capital Cost and Wear of Shoes and Dies for Nissen and Ordinary Stamps.

		Nissen Stamp.	Ordinary Stamp.
1.	Net capacity per month (tons)	40,000	40,000
2.	Net capacity per day (tons)	1,330	1,330
3.	Percentage of net working time to total time ...	95%	90%
4.	Gross capacity per day (tons)	1,400	1,480
5.	Size of screen. (9 mesh per square inch)	0·277	0·277
6.	Falling weight of stamp (pounds)...	2,000	1,480
7.	Duty per stamp per day of 24 hours (tons)	29·16	14·8
8.	Number of stamps required	48	100
9.	Power consumption per stamp H.P.	4·3	3·02
10.	Power consumption of entire mill H.P.	206·4	300
11.	Loss of power in motors, shafting and belting ...		
	(a) Nissen mill, 5 drives at 7·8 H.P., 2 drives at 4·8 H.P.	48·6	
	(b) Ordinary mill, 10 drives at 7·8 H.P.		78
12.	Total power demand H.P.	255	370
13.	Total power demand K.W.	190	276
14.	Cost of power per K.W. hour (pence)	0·6	0·6
15.	Cost of power per day	£11 7s. 10d.	£16 11s. 2d.
16.	Saving in favour of Nissen stamps per day	£5 3s. 4d.	
17.	Saving in favour of Nissen stamps per ton milled (pence)	0·931	
18.	Capital cost of battery (as per Appendix I.) ...	£20,000	£23,000
19.	Capital cost of battery (per ton milled per day) ...	£15·04	£17·29
20.	Saving in favour of Nissen mill	£3,000	
21.	Interest on capital saved at 12% per annum, per month	£30	
22.	Interest on capital saved at 12% per day	£1	
23.	Interest on capital saved per ton milled (pence) ...	0·180	
24.	Wear on shoes and dies per ton milled (lb.)	0·3	0·4
25.	Cost of shoes and dies at 3d. per lb. per ton milled (pence)	0·6	0·8
26.	Difference in favour of Nissen mill (pence)	0·2	
	Total saving (18 + 20 + 931) pence per ton milled ...	1·311d.	

of £3,000 in favour of Nissen stamps. Allowing for interest, depreciation and redemption of capital at the rate of 12% per annum, a saving of £1 per day or ·1804d. per ton milled is obtained.

Power.—As can be seen from table No. III. the power consumption for Nissen stamps is only 255 H.P. 190 K.W. as against 370 H.P. or 276 K.W. for ordinary batteries, representing a saving of 94 K.W. amounting to £5 3s. 4d. per day or 0·931d. per ton milled.

Wear and Tear.—The wear and tear of shoes and dies during the tests at the City Deep was not obtained by weighing the worn parts, nor was any account taken of the metal which must be thrown away in both shoes and dies, as it is impossible to wear them out completely. Figure VIII. shows a set of shoes and dies that have been in use in the Nissen stamp from May until September 1911, during which time about 1,600 tons of ore were milled. The total loss of metal was 480 lb., equalling 0·299 lb. per

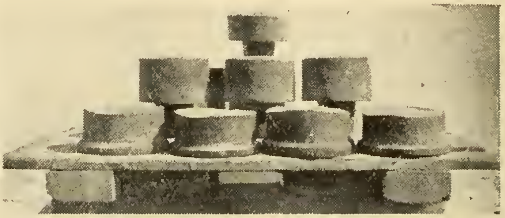

FIG. VIII.

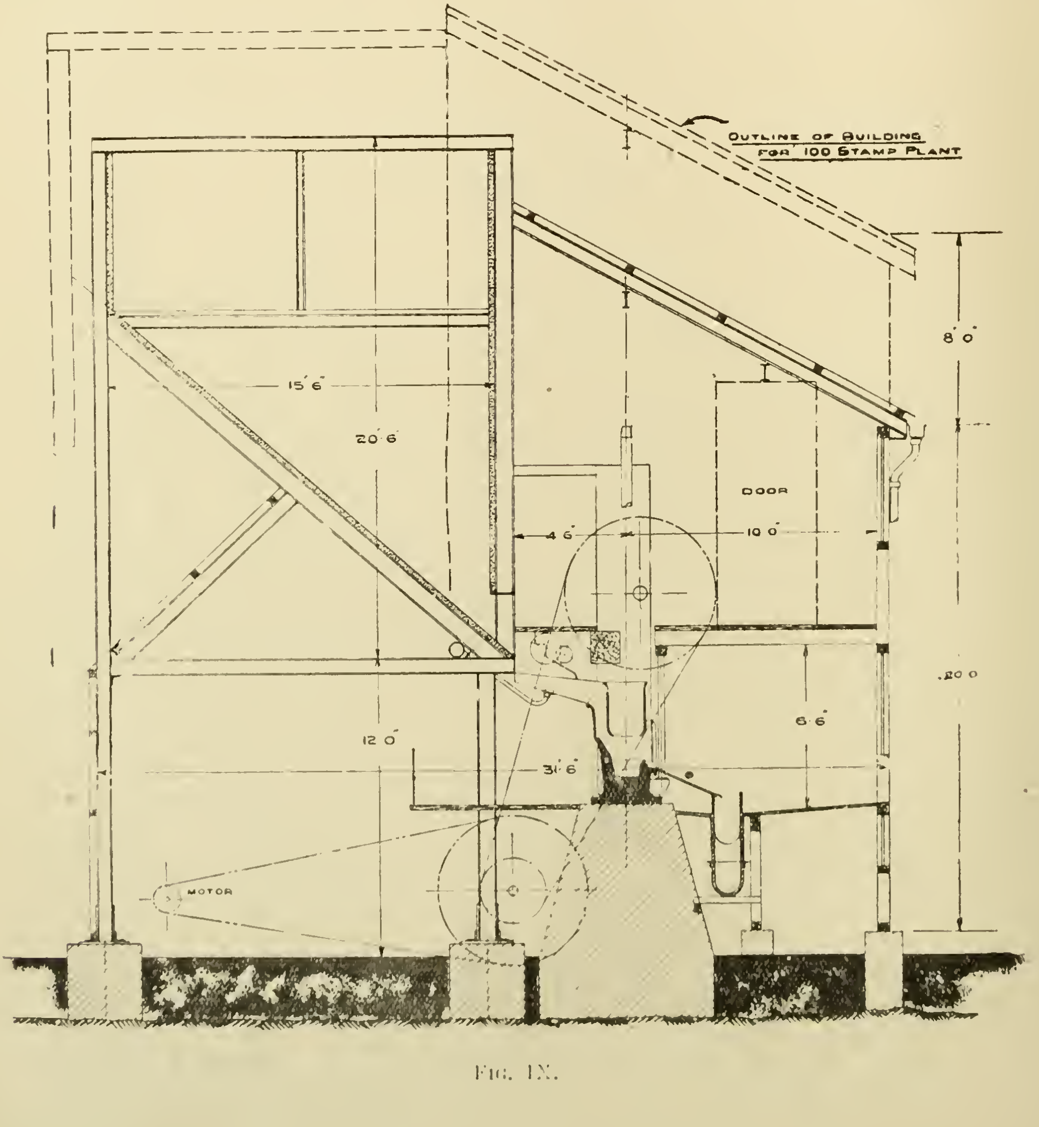

FIG. IX.

ton of ore crushed. The top shoe of the pyramid shows much greater wear than the others, which is explained by the fact that it was of inferior material and was an experimental one. It is however safe to say that in view of the remarkably even wear in the Nissen shoes and dies, and considering the tonnage passed through, the amount of wear per ton for these stamps will not exceed 0·3lb. of metal per ton crushed, as against 0·4lb. per ton for the ordinary battery.

These figures for the ordinary battery must be considered low, when taking into consideration the average wear of the Rand. The difference in favour of the Nissen stamps in wear of shoes and dies is 0·1 lb. equivalent to 0·2d. per ton milled.

The saving under different headings per ton of ore crushed is:—

(1) Saving in capital charges ...	0·1804d.
(2) ,, ,, power ,, ...	0·931d.
(3) ,, ,, metal (wear and tear) ...	0·2d.
Total	1·31d.

or a saving of £7 5s. 3d. per day. This represents a saving of £2,500 per annum.

Fig. IX. is a cross-section of the 48 stamp battery forming the basis for estimates herein mentioned. It will be particularly noticed that the building required is very small. This figure also shows in dotted lines the approximate height of bin and building that would be required for the ordinary type of battery. Fig. X. is a cross section of a stamp battery building, showing a design whereby circular steel tanks are utilised as ore bins. In ordinary warm climates it would not be necessary to cover these tanks by a roof, for by the use of Nissen stamp feeders, all water will pass through the mortars without leakage.

In the preceding statement no account has been taken of the obviously reduced cost of renewals in the Nissen mill as compared with the ordinary mill. This need only be pointed out to be appreciated. It is evident that given the same conditions, the breakage of stems and cam-shafts must be less, and the splitting of heads is rendered impossible by the greater diameter, as well as the breakage of tappets due to bad design. In addition the cost of mortar box liners for the Nissen stamps is less per ton milled; apart from the actual cost of the new parts for renewals the labour required and time lost in changing is much less than for the ordinary stamp mortars, hence the estimate of the saving to be expected by using Nissen stamps cannot be thought too sanguine.

Apart from equipping new mines with stamp batteries, the question of reconditioning existing batteries having timber foundations, presents a continually recurring problem. A considerable amount of money has been spent in this work during the past few years, a case being that of the Simmer and Jack Proprietary Mines described by Mr. C. O. Schmitt and published in the *Journal.**

It can be taken that the cost of doing this work would average £13,800 for 200 stamps of 1,350 lb. falling weight as detailed in Appendix II. If instead of reconditioning the battery with light stamps, the Nissen stamps were used a substantial saving, chiefly in power, would be effected. Assuming the case of an existing battery of 200 stamps of 1,350 lb. falling weight being remodelled by the introduction of Nissen stamps, the following alternative proposals are possible:

"A."—80 Nissen stamps of 2,500 lb. falling weight would take the place of 200 existing stamps at a cost of £16,500
Extra capital outlay ... £3,000
Saving in power ... ... 168 h.p.

"B."—96 Nissen stamps of 2,000 lb. falling weight of which 80 would take the place of the existing 200 stamp, whilst 16 would constitute an addition to the mill, with building ore bins and ore transport, could be installed at a cost of £19,500
Extra capital outlay ... £6,000
Saving in power ... ... 150 h.p.

"C."—80 Nissen stamps of 2,000 lb. falling weight take the place of the existing 200 stamps with one tube mill to maintain the same crushing capacity possessed by the former plant, at a cost of £15,500
Extra capital outlay ... £2,000
Saving in power ... ... 234 h.p.

Table IV. is a comparative statement of capital and power costs for the various methods of doing the work. Considering these three alternatives it will be seen that from all points of view "C" method will probably be the most favourable because the extra expenditure due to the installation of Nissen stamps and one tube mill will only amount to £2,000, whilst the saving in power amounts to 134 h.p. The saving in combined capital and power charges as compared with reconditioning the existing 200 light stamps, will be over £5 per day or £1,700 per annum. All the estimates are based on doing the reconditioning of the 200 light stamps at a cost of £13,500, and this figure, whilst it may be excessive in one case, will probably be low in many instances. Even assuming that the work of reconditioning 200 stamps could be done at one-half the estimated amount, the saving in capital and power charges will still be £850 per annum. In the foregoing no account has been taken of the obvious saving in labour and maintenance charges. It is apparent that the cost af operating 80 well

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* See this *Journal*, Vol. X., Jan., 1910, pp. 242—252.

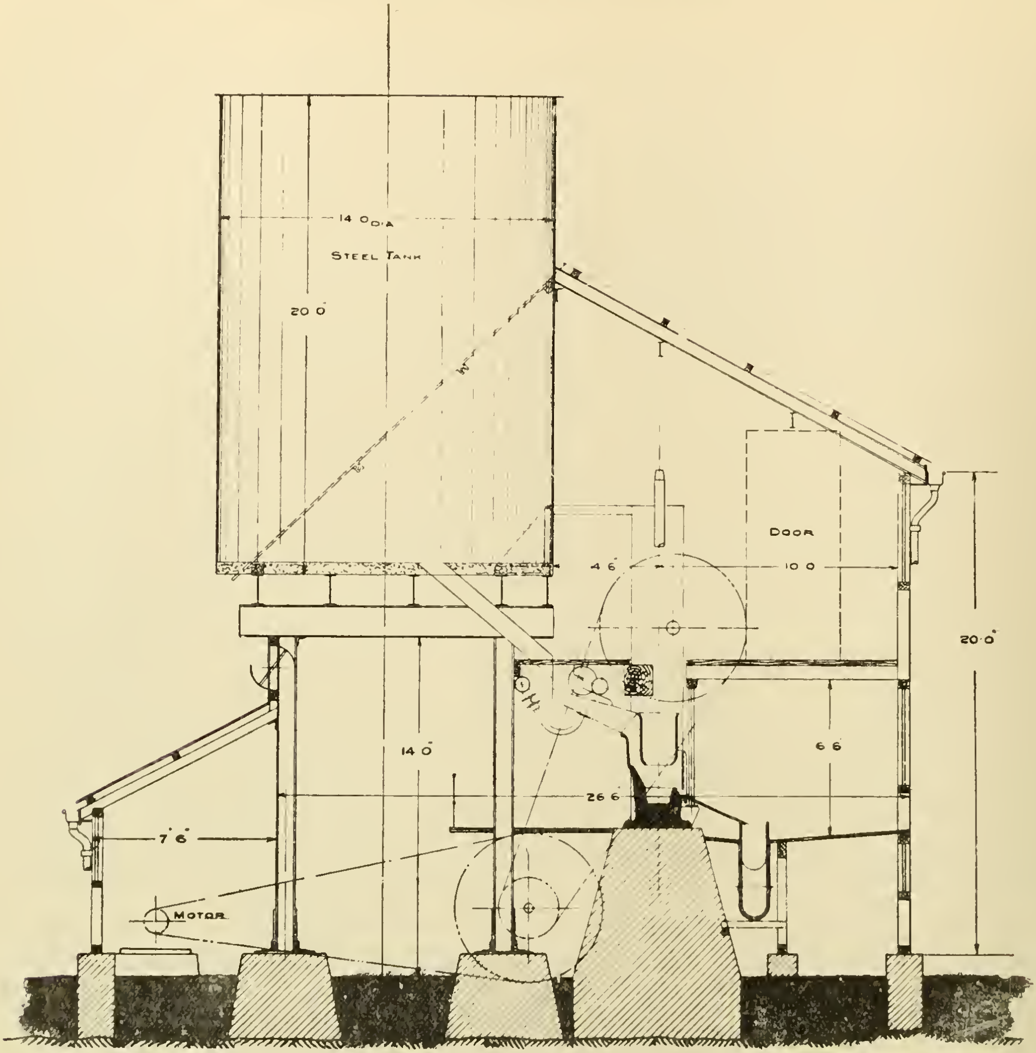

FIG. X.

designed stamps plus one tube mill would be considerably less than would be the case for 200 stamps.

The number of Nissen stamps required has been calculated on a basis of the experience gained during the City Deep trials where the efficiency of the Nissen stamp was 30% higher than that of the heavy type ordinary stamp. It is reasonable to expect that a comparative trial between the Nissen stamp and the ordinary stamp of light weight will show a greater percentage of saving for the former so that the foregoing assumptions must be considered conservative.

In view of the foregoing the writer hopes he has been able to show that no apprehension need be felt regarding the future of the gravity stamp. Although the ordinary type of stamp may have its limitations the improvements effected in gravity stamp mill design as expressed by the Nissen stamp mill have undoubtedly increased its range of usefulness. Whether the ultimate limit has now been reached may fairly be doubted for it seems reasonable to expect that further improvements can be effected, thus benefiting the industry both here and in other parts of the world.

The writer has no doubt that with such a large number of experienced mill-men as members of this society, new ideas will be brought forward during the discussion, which he will be the first to welcome.

TABLE IV.

Comparative Statement of Capital and Power Costs for Reconditioning a 200 Stamp Mill of 1,350 lb. Falling Weight or its equivalent in Nissen Stamps.

	Re-conditioning of 200 Stamps of 1,350 lb.	A 80 Nissen Stamps of 2,500 lb.	B 80 Nissen Stamps of 2,000 lb. in Old Mill 16 in New Mill.	C 80 Nissen Stamps of 2,000 lb. in Old Mill and One Tube Mill.
Capital cost	£13,500	16,500	19,500	15,500
Interest, depreciation and redemption at 12% per annum per month...	135	165	195	155
per day ...	£4 10s. 0d.	£5 10s. 0d.	£6 10s. 0d.	£5 3s. 4d.
Total number of stamps	200	80	96	80
Power demand per stamp H.P. ...	2·7	5·0	4·3	4·3
Power demand for entire mill H.P....	540	400	412	344
Power losses H.P.	120	92	98	82
Power demand of tube mill ...	—	—	—	100
Total power demand H.P. ...	660	492	510	526
Total power demand K.W. ...	490	366	380	392
Cost of power per K.W. hour ...	0·6d.	0·6d.	0·6d.	0·6d.
Cost of power per day	£28 8s. 0d.	£22 7s. 6d.	£22 8s. 0d.	£23 10s. 4d.
Total of capital and power costs per day	£33 18s. 0d.	£27 17s. 6d.	£28 18s. 0d.	£28 13s. 8d.
Saving by using Nissen stamps, per day		£6 0s. 6d.	£5 0s. 0d.	£5 4s. 4d.
Saving per annum (approximate) ...		£2,040	£1,700	£1,700

NOTE.—The cost of power at 0·6d. per unit is very low for the average plant on the Rand so that if a higher figure were taken the result would be still more favourable for Nissen plants.

The writer wishes to acknowledge his appreciation of service rendered to science and to the mining industry by the Central Mining and Investment Corporation for making the tests at the City Deep, and for permission to publish the results incorporated in these notes.

APPENDIX I.

Comparative Estimates for a Stamp Battery of the Nissen and Ordinary Type, Capable of Crushing 40,000 tons per month.

	Nissen.	Ordinary.
Foundations	£2,220	£2,800
Mill machinery ...	8,280	9,180
Ore bins	3,730	3,990
Building	1,050	1,250
Battery and cam shaft floor	715	715
Battery framing ...	720	800
Battery drive (motor shafting, belting, etc.) ...	1,900	2,700
Water service	260	300
Lighting	60	60
Mill crawls	20	20
Painting, etc.	60	60
Mill launders	250	250
Contingencies	735	875
	£20,000	£23,000

APPENDIX II.

Approximate estimate for re-conditioning a battery of 200 stamps of 1,350 lb.

Foundations and bolts (erected) ...	2,500
New machinery (erected)	8,000
Re-erecting old machinery	1,000
Repairs to mill framing, floors, launders, water service, etc.	2,000
Total	£13,500

Alternate estimate for re-conditioning above battery with Nissen stamps.

A.—80 Nissen stamps of 2,500 lb.—

Foundations and bolts (erected) ...	2,000
Mill machinery (erected)	12,500
Repairs to mill framing, floors, etc. ...	2,000
Total	£16,500

B.—96 Nissen stamps of 2,000 lb. (80 stamps in existing mill. 16 stamps with bin and building)

Foundations and bolts for 80 stamps (erected)	£2,000
Mill machinery for 80 stamps (erected)	9,250
Repairs to mill framing, etc. (erected)	2,000
New battery of 16 stamps (erected) ...	6,250
Total	£19,500

C.—80 Nissen stamps of 2,000 lb. and one tube mill (22 ft. × 5 ft. 6 in.).

Foundations and bolts (erected) ...	£2,000
Mill machinery (erected)	9,250
Repairs to mill framing, etc. ...	2,000
Tube mill (erected)	2,250
Total	£15,500

The Chairman: I must move a hearty vote of thanks to Mr. Nissen for this valuable and interesting paper, which I now declare open for discussion.

Mr. E. J. Laschinger *(Vice-President)*: I heartily endorse the remarks of the Chairman that Mr. Nissen deserves the cordial thanks of the members for his able and interesting paper. We know that the author has devoted a great deal of time and thought and attention to the improvement in design of the gravity stamp, and the author is, I think I may say without any reflections, an authority on the subject. In his final remarks he has expressed the view which has been the result of the experience of a great many engineers and other men who have studied crushing machinery, and that is that the stamp has still a great future before it, as far as we can see at the present time.

In a paper like this one cannot make many hurried remarks that will carry much weight but it seems to me that it is desirable to make *some* impromptu remarks on a subject of this kind because such remarks, though very often embodying first thoughts which are not always the best, tend to open up a field of useful discussion. There are quite a number of remarks which have been made by the author and put forward without proof but no doubt he will be able by the time the discussion has terminated, to bring forward his proofs for the statements which he has made.

He says "since the function of the stamp is one of crushing the greatest quantity of ore for a given expenditure of power it follows that to get the highest efficiency from each blow of the stamp the conditions of feed are most important." I think the truth of that remark has been appreciated by everyone who has studied stamps. The conditions of the feed have probably more to do with the efficiency of the stamp than any other factor connected with it. As one who has made some study of stamps I think that it is one of the hardest things to get the old-time mill-man to alter his feed in any way whatever in spite of the fact that one of the most important factors in obtaining high efficiency of stamps is this question of feed.

He also says "coming to stamps weighing above 1,250 lb. and up to 2,000 lb. each we find the same diameter shoes and dies are used in all cases so that the mortar need not be longer." This point of shoe area was very well brought out by Dr. Caldecott in his well-known paper read simultaneously before the Institution of Mining and Metallurgy and this Society,* and on thinking over the reason why the size of shoes and dies is not varied in proportion to the increase in the falling weight I think one comes to the natural conclusion that actually within very wide limits the area of the shoe and die has very little to do with the efficiency of the gravity stamp. In the cases of crushers, rolls and steam stamps, the unit pressure in those kinds of machines is very much greater than the unit pressure in gravity stamps even of the heaviest type. The author goes on to say that he has made up his ten inch diameter shoe and die as against the smaller diameter at the City Deep, as the result of experience. It would be interesting to know what that experience is.

Another feature that the author claims for his machine is its large screen area. The question of screen area has been thrashed out thoroughly before this Society, and in the case of coarse crushing there is sufficient screen area with ordinary mortars. The point he makes about the screen being in the position in which it can discharge the crushed ore most readily is however a very good one. His screen is placed radially round the stamp and this should be more effective in getting rid of the ore which is sufficiently crushed in the least possible time. With regard to the stamp being open fronted, as we know, that idea is not new, but is an improvement which he has incorporated in his stamp design.

The author goes on to say "one of the special features of the Nissen mortar is that it can be cast without fear of shrinkage strains which are so destructive to ordinary five stamp mortars." I am prepared to agree that in the design he has shown, the shrinkage strain in the mortar castings will be less than in the ordinary castings but whether he has discovered the secret of casting mortars without shrinkage strains is doubtful.

With regard to the running time of Nissen's stamps and of other stamps the author claims here that he gets over 95% running time for the mill and also says in another place that it is very difficult to get 90% of the running time in the ordinary battery. No battery on the Rand shows less than 90% effective running time. Unless there has been a shortage of rock coming to the mill I do not think any battery manager would allow his mill to show even 90% of the running time; it would be nearer 95% and sometimes even higher. It is always a very good thing to have the battery larger than is actually needed on a high running time basis not only for mechanical maintenance but for better supervision. If the mill is kept crushing all the rock the mine can send up there will be an incentive to the mines to keep the mills supplied with ore. It is a sound principle to design any stamp battery on a 90 running time basis.

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* See this *Journal*, Vol. X., Sept. 1909, pp. 108–115.

One of the greatest troubles and one which anyone who has been responsible for the design of stamp mills meets, is bad and irregular wear of dies. An inspection of the Nissen dies and shoes shows that they have been very evenly worn, in fact the regularity of the wear of the dies is just as good as the wear of the shoes and that is saying a good deal.

The author makes a point of being able to keep the ore bins lower for his type of stamp than for the ordinary stamp. For myself I should like to see proof of that. It is quite possible to make the height of the feed ore chute in a battery mill bin just as low with the ordinary type as with the Nissen type

Then the author claims that owing to the smaller number of shocks to the cam shafts his cam shaft would have a much longer life. That statement should be backed up with some actual figures on the life of cam shafts. There is no means of predicting the life of a cam shaft. One may last not more than a few weeks, days or hours; another may last as long as three years, and if the author claims that his cam shafts last longer I think we ought to have some definite proofs.

With regard to the construction of the kingposts we see that he has adopted a composite construction, consisting of a steel joist bolted between two pieces of wood. I have not had any experience of that kind of construction, but I am rather sceptical as to its superiority. I should have thought it would have been better to have had a well built up wooden kingpost because there must be some vibration between the steel and the wood, and I should imagine that in the course of time the bolts holding the two together would give.

He further says, "The greater capacity of the Nissen stamp reduces the total weight of iron and steel in the battery for a given tonnage, and this may amount to as much as 40%." I think it requires some figures to justify the statement that it is 40% less.

In making his comparative estimate the author is of course justified in making that estimate as favourable for his stamp construction as possible, but still we ought to know the reasons why he should compare a heavy Nissen stamp with a light ordinary stamp for the purpose of making comparative estimates. From experience we know that it is quite possible to effectively run a stamp up to 2,000lbs weight in the ordinary way, and there is no need to go back to the 1,300 or 1,400lb. stamp, but the author says, "It therefore remains to compare a battery of 48 Nissen stamps of 2,000lb. each with a battery of 100 ordinary stamps of 1,400lb. each." With regard to the figures of the estimate I should like to take an opportunity of looking through these very carefully. I think the cost of both battery constructions is too cheap, and that the costs should be rather higher. The comparison of power costs is very important. One should have an opportunity of looking into them rather carefully before criticising the author's figures.

There is no doubt that this is a subject which is at the present time of the very greatest interest to metallurgists, engineers and mining men generally all over the world. The stamp battery has within the last few years been assailed in a most thorough manner. Many devices have been tried to supersede the stamp. In regard to the whole practical problem of crushing, although it is possible to put down in a scheme a combination of three or four machines each of which has its field of usefulness, and to make what one might consider an ideal plant—a series of machines which will give the highest possible paper efficiency—it is true that in any process which involves a series of operations there is the great practical difficulty of making that series work effectively, cheaply and with true economy. In practice the battery stamp has always this to say for itself, that is it does not make much difference whether the ore is one inch or two inches or even sometimes six inches if anything has gone wrong with the breakers, the stamps will deal with it and that with very slight variation in the cost of crushing. A series of elaborate machines each of which must have a feed of definite size causes trouble and has to be shut down should anything go wrong at any stage in the series.

Mr. Jas. E. Thomas *(Member of Council)*: In Mr. Nissen's comparative statement of capital cost for a battery of the ordinary type of 100 stamps and a Nissen battery of equivalent capacity, the figure given for interest, depreciation and redemption is 12%. Is this the amount usually allowed on these fields for these items?

Mr. G. A. Robertson (*Member*): I should like to ask the author how old this invention is, because it appears to me that this is another of those inventions developed years ago which has at last been given a trial on the Rand.

Mr. A. M. Kratz *(Member)*: Mr. Nissen has said that his stamp has a very low centre of gravity and that this gives him a very low construction, thus saving bin area and materially reducing capital costs. I do not think that the centre of gravity of his stamp is much lower than the City Deep stamp, which has a head of 9½in. dia., where his is 10in. He then goes on and compares his 2,000lb. stamp battery with a 1,350lb. stamp battery for a given tonnage, and finds the total lengths almost equal. I think he has compared the length of a City Deep battery, substituting 1,350lb. stamps. City Deep stamps are 2,000lb., and it is manifestly unfair to compare this. I think that his point of saving in bin capacity and length, thus effecting a saving in capital cost, is not a good one. If there is so great an advantage in a low centre of gravity, I am pretty sure that by widening the mortar box slightly, stamps with a lower centre of gravity can be used, thus obtaining all the advantages he claims for low battery construction. Further, I am convinced that if he compares his 2,000lb. stamp battery length for a given tonnage, with a similar length using the usual 2,000 lb. stamp construction that the saving in capital expenditure will be much in favour of the ten stamp battery.

Mr. M. H. Coombe (*Member of Council*): On this paper I would only wish to refer to the gradual coming back on the part of mill architects to the old methods of construction. From the pitch pine mortar box piles to the concrete, then the additions of a heavy anvil block, then the substitution of a heavy anvil block by pitch pine cushions laid in concrete, and finally the laying of the concrete in segments, each segment carryng two mortar boxes or 10 stamps. It is another proof that the mechanical engineers work moves in circles as does human thought and effort in all the professions.

Mr. E. Roberts (*Member*): I think the author has given us a really good paper, and there are several points in it with which I agree. Increasing the diameter of the shoe and die seems especially to be a sound policy, as up to the present the majority of mill designers on these fields have only increased the weight of the stamp. That in my opinion is the cause of the small increase in the work performed against the power expended. I remember some years ago working in a mill of 850 lb. stamps when we were running at 96 drops a minute and crushing through a 900 mesh screen, the duty per stamp being 4·5 tons per day. We increased the weight of the stamps to 1,250 lbs., and the speed to 100 revolutions per minute, still keeping the 900 mesh screen, but strange to relate our crushing was only increased to 5·5 tons per day. To better our crushing we put on a 500 mesh screen, when our stamp duty went up to over six tons per day. This set me thinking there was something wrong in the system, and I came to the conclusion that the fault lay in keeping to the same diameter head and die, and in the increasing of the thickness of the feed in the mortar box. I have not seen any reason to alter that opinion up to the present, and I think if the author increased his diameter of head to 12 in. he would have still better results. I do not care very much for the right and left hand cam on the one shaft. It is not good in practice, and before I close I should like to ask if there has ever been a 900 mesh screen tried with the heavy stamps, and what was the duty per stamp?

SOME ASPECTS OF MINE VENTILATION AND TEMPERATURE ON THE WITWATERSRAND.

(*Read at May Meeting, 1911.*)

By J. Whitehouse, B.Sc., M.I.M.E. (*Associate*) and W. L. Wotherspoon.

DISCUSSION.

Mr. M. H. Coombe (*Member of Council*): Before the reply to the discussion is read, I would like to make a few remarks on Mr. Rowland Gascoyne's contribution as given at the last monthly meeting of this Society and also published in our *Journal*.

Mr. Gascoyne says that an 18 ft. diameter circular shaft is equal in area to the largest rectangular shaft sunk on these fields. I ask him to revise his figures. An 18 ft. diameter circular shaft has an area of 254·4696 sq. ft.

A 7-compartment rectangular shaft has an area in the clear of 284·375 sq. ft., a gain of 30 ft. over the circular shaft, but, as the ground is cut, the area of a rectangular shaft is 450 sq. ft.

As skeleton setts only are used in these shafts, it will be seen that a very large proportion of this cutting is available as air spaces and for air circulation.

It is simply theorising to say that "the advantages of a circular shaft over a rectangular shaft of exactly the same area are most pronounced," as far as ventilation is concerned. As far as I know there are no comparative data available, and until the two types of shafts can be placed under similar conditions to do the same work, no reliable data can possibly be obtained. One thing is certain, that metalliferous engineers and miners the world over prefer the rectangular shaft for all purposes of their work, including ventilation, and that even the Scottish coal fields are equipped with the rectangular shafts, and I have not heard that their ventilation is inferior to that of the English, Welsh or Continental coal fields. As Mr. Gascoyne remarks, it would be considered improper to go into other questions concerning the merits or demerits of these two types of shafts for general mining work in connection with Messrs. Whitehouse and Wotherspoon's paper, but I trust such a paper will be brought before this Society and the whole question laid open for debate and discussion. I do not agree with Mr. Gascoyne's remark that "the round shaft offers immense advantages over the rectangular shaft in case it becomes necessary to sink through heavily watered ground." There has never been a round shaft sunk yet where a rectangular shaft could not have been sunk, and probably at less cost. We have an instance on the Western Rand Estates of failure to pierce the ground with a round shaft on account of water troubles. I maintain that a rectangular shaft, 7 compartments, could be sunk alongside of the round shaft without any extraordinary difficulty.

Then I must refer to Mr. Gascoyne's quoted cost of the New Modderfontein circular shaft, viz, £20 per foot. Can Mr. Gascoyne substantiate this figure? I for one accept it with the proverbial grain of salt.

Mr. Rowland Gascoyne (*Member*): May I ask for your indulgence to enable me to reply to the remarks of Mr. Coombe? I wish to take this opportunity of confirming all I stated with regard to circular and rectangular shafts. An 18 ft. diameter circular shaft has an area of 254 sq. ft., but granted that Mr. Coombe has discovered a seven compartment rectangular shaft with an area over all of 284 sq. ft. there still remains 30 sq. ft. to be deducted, thus bringing as I stated the rectangular shaft net area down to 254 sq. ft. which is the same area as an 18 ft. diameter circular shaft as already stated.

I took the West Rand Consolidated seven compartment shaft as one of the most representative of its kind and did not refer to any shaft not being completed or in actual use. The West Rand Consolidated shaft measures in the clear 41 ft. in length by a width of 6 ft. giving a gross area of 246 sq. ft., but if the timbering is allowed for the net area is only 219 sq. ft. This shows a gross area for this particular shaft of 246 sq. ft. as compared with 254 sq. ft. a difference scarcely worth mentioning.

Mr. Coombe's suggestion to include a very large proportion of the whole of the excavated area for air circulation is so unreasonable that I do not consider it necessary to take any serious notice of the proposal.

When Mr. Coombe states that it is simply theorising for me to say "that the advantages of a circular shaft over a rectangular shaft of exactly the same area are most pronounced as far as ventilation is concerned," he shows that he has not fully gone into the matter and his suggestion to have the two types of shafts placed side by side under similar conditions is as I will show quite unnecessary.

If Mr. Coombe will refer to my remarks again he will find what I stated was that where all other conditions are equal the quantity of air passing through a shaft will vary in the same proportion as the rubbing surface increases or decreases. Now this is an axiom in mine ventiltion, and with the application of simple arithmetical problems the advantages of a circular over a rectangular shape as regards ventilation are easily proved. Now if we take a circular shaft 18ft. in diameter and 1,000 ft. deep we arrive at the following: $18 \times 3{\cdot}1416 = 56{\cdot}5488$ circumference which multiplied by the depth 1,000ft. = 56,548 sq. ft. of rubbing surface. Now take the West Rand Consolidated seven compartment shaft and calculate the rubbing surface $41 + 41 + 6 + 6 = 94$ft. as the inside dimensions, which multiplied by the depth of 1,000ft. gives 94,000 sq. ft. area of rubbing surface for a rectangular shaft as compared with 56,548 sq. ft. rubbing surface for a circular shaft of approximately the same area. Therefore if this rectangular shaft will pass 100,000 cub. ft. of air per minute a circular shaft of the same area and under exactly the same conditions will pass 166,000 cub. ft. of air per minute, whilst the division of the rectangular shaft into compartments adds still further to its air throttling capabilities. I do not know whether it is necessary for me to go further into this aspect of mine ventilation, but I find that in discussing Mr. Penlerick's paper Mr. Stuart Martin said (vol. xi., p. 215) when explaining the difficulties encountered at the Village Deep that "The small free air space and the enormous frictional resistances in the main rectangular vertical shafts prohibited the placing of the fan on the surface," and what this meant to the mines involved in this ventilation scheme is too self-evident to need any further explanation.

Mr. Coombe goes on to say that "one thing is certain, that metalliferous engineers and miners the world over prefer the rectangular shaft for all purposes of their work, including ventilation, and that even the Scottish coal fields are equipped with rectangular shafts, and I have not heard that their ventilation is inferior to that of the English, Welsh or Continental coalfields." In this statement about the Scottish coalfields, Mr. Coombe is particularly unfortunate, because if there is one mining field in the United Kingdom with a bad reputation for indifferent ventilation, it is those portions of Scotland where the rectangular shafts are used. Always, of course, excepting Cornwall and the lead mining districts of

England and Wales, whose faulty ventilation has long been a byeword in all the mining districts of England. Even in Cornwall, however, they have introduced circular shafts to improve the ventilation and attempt to bring the mining practice of that field up to date. With regard to metalliferous engineers and miners the world over preferring the rectangular shaft for all purposes of their work, I have shown that that remark can scarcely be said to apply to Cornwall. My experience in different parts of the world rather tends to show that mining engineers and mine managers frequently adapt their practice more to ruling local conditions than to rules of thumb and prejudice, perhaps previously acquired in other districts where conditions are possibly totally different.

Mr. Coombe says that he does not agree with my remarks as to the advantages offered by a round shaft over a rectangular shaft in heavily watered strata, and further says that a rectangular shaft could have been sunk in all cases where a round shaft has been sunk, and probably at less cost. No doubt rectangular shafts can be sunk in heavily watered strata, but that particular shape of shaft naturally does not lend itself so readily to tubbing back the water as a circular shaft would do, and without tubbing or similar contrivances the water has a tendency to increase with the depth, whereas by the use of tubbing the water is kept out of the way, and pumping operations reduced to a minimum. In my own experience, by the use of tubbing in a circular shaft, a flow of ten million gallons per day has been kept back in the early sinking stages, and the shaft proceeded with subsequently as a dry sinking. At Welgedacht when the water reached less than two million gallons a day, and at Grootvlei when the water reached three millions with a five and seven compartment rectangular shaft, in each case sinking operations were suspended. Mr. Coombe may say that it is all a question of adequate pumping appliances, but in a circular shaft water troubles are dealt with as they arise, and by the use of tubbing, pumping operations may be promptly dispensed with. The Western Rand Estates I am informed suspended sinking operations principally for financial reasons before the sinking operations got properly started.

If Mr. Coombe will refer to the last published report of the New Modderfontein Co., he will ascertain for himself that including the total cost of sinking and lining, the circular shaft had so far cost on the average £24 a foot and for my own part I do not see as suggested by Mr. Coombe any use for the proverbial grain of salt. At Welgedacht the cost of sinking a rectangular shaped shaft is officially quoted at £57 a foot but then the conditions at Welgedacht and Modderfontein are quite different.

I have used rectangular shafts somewhat extensively and am therefore not prejudiced against their use especially where as in the diverse and varying conditions of metalliferous mining they can be beneficially applied. For small shafts where the ground is fairly good, value and life of property uncertain, timber plentiful and cheap, no better and handier shape can perhaps be adopted. When however it comes to working an extensive deep level area on the Rand, where the conditions so closely approach those of coal mining and good ventilation also becomes a *sine qua non*, it is necessary at all events to consider the question of the shape of the shafts as well as other questions with an open and unprejudiced mind, and to adopt whatever seems desirable and best from both coal and metal mining practice.

REPLY TO DISCUSSION.

Mr. W. L. Wotherspoon: Mr. Rowland Gascoyne has raised one point of interest as the result of his experiments; namely, that a steeper heat gradient is observed in flat measures as compared with the steeper measures. We quite agree that in all probability the heat noticed in coal seams was probably due to oxidation, and therefore the holes used for these determinations should be deeper than those necessary for observations in rock. The figures given by Mr. Phillips are based on some of the earlier results included in our paper. The difficulty of guiding air currents in mines is undoubtedly an important point, and unless good stoppings are put in, difficulty in this direction is sure to arise. We remember a very interesting case in Derbyshire where an adit about a mile long through the lower carboniferous measures had a splendid current of air going into it although there was no return, the air passing through the rock, which was fissured, and in that way reaching the surface. The ventilation of the lower unconnected workings becomes a problem in the mines of the Rand owing to the heavy blasting operations which make air pipes or brattice very difficult to maintain. Mr. Gascoyne has raised a point with regard to circular shafts as compared with rectangular ones. With regard to this question, it must be understood, that, although there are, perhaps, seven useful compartments in a rectangular shaft, they are not all drawing compartments, one at least being used for pipes, etc. The open spaces included in a circular shaft, of course, take the place of the 7th compartment for this purpose, and comparing say a 21 ft. circular shaft with a rectangular shaft having six drawing compartments with a circular one of the size given, we think the capacity of the one is not greater than the other. With regard to Fig. VII., Mr. Gascoyne is quite correct in saying that this scheme for ventilation does not comply with the principle which should, where possible, be followed, and points out that only in an old mine is this justified. The authors would point out that this is actually the case, the scheme in question having been drawn out for a mine which is fully developed and more or less worked out and cannot therefore justify anything more elaborate.

Mr. Johnson asks so many questions that he will, no doubt, not mind if a few of them are passed over, however, we will endeavour to answer the more important ones.

He asks why the percentage of CO_2 is greater later in the morning than when the shift goes down. The explanation of this would appear to be that when the men first descend, the mine is full of fresh air, and any vitiation owing to the presence of human beings, is not felt for some time.

With reference to the temperature taken as an example for natural ventilation, and the influence which surface conditions may have upon this, the authors will point out that the temperature of 84 F. referred to by Mr. Johnson was a maximum figure extending only over a few hours, and not an average one which should be used, the daily variation on the Rand being as much as 20° to 30°F.

The costs referred to in the paper are based on the tonnage of the Village Deep and the Village Main Reef, the fan ventilating both these mines. With regard to the placing of the Village Deep fan, for practical reasons the authors consider its position by far the best, since it lends itself very well to the natural conditions of the mine, and had it been placed as Mr. Johnson suggests, endless trouble would have been encountered in bratticing and keeping things tight, and the W G. required to circulate the quantity of air would have been much greater owing to such a large quantity having to be drawn through one small shaft. Under the present conditions there are 3 downcast shafts and many outlets to act as upcast.

Surely Mr. Johnson can understand that in any case the currents of air would be the same. We should have been glad to hear what good purpose it would serve if we were to give Mr. Johnson the proportion of the W.G. required for each mine.

Dr. Aymard asks whether it is true that Mr. Whitehouse paid a visit to the Nourse Mines and saw the system of dust prevention in use there ; the answer is that he has.

Dr. Aymard also asks why this system has not been adopted on the Village Deep. In answer to this the authors would point out that managers are always prepared to exchange views, and anything which was considered worth adopting has been tried on the Village Deep, and the system which has been found to work best is based on the experiments made by Mr. Jourdan at the Nourse Mines, it being less troublesome for the men than any other bag arrangement which has being tried, and further the time of drilling holes is much reduced by it. Experiments which have been made on the Village Deep show that back holes can be drilled in less than half the time required if they are drilled dry.

The authors would like to state that we are indebted to Mr. Stuart Martin for the excellent system of ventilation which exists at the Village Deep, and which provides fresh currents of air throughout the mine, thus making the deepest workings cooler and more comfortable to work in than most outcrop mines.

The Chairman: Before we pass on to the other papers for discussion, I should like to draw your attention to a speech delivered by Mr. Lionel Phillips, before the South African Institute of Electrical Engineers recently. In that speech he referred to the paper which had been read before this Society on the question of Mine Ventilation and Earth Temperatures, and he said how surprised he had been to find so little discussion taking place on that important question. He went on to say that however subordinate a position a man held he could not possibly endanger that position by freely taking part in the discussion of any scientific paper, and that he personally was anxious to see a much greater interest taken in these discussions. The Council of the Society has been somewhat disappointed with the small amount of discussion which takes place on many important papers, and I trust that Mr. Phillips' remarks in this connection will be taken to heart, and that members will understand that they will not in any way prejudice their position by giving the freest vent to their opinions on the papers that are read before us.

SINKING OPERATIONS AT THE NORMA SHAFT, KATTOWITZ, UPPER SILESIA.

By B. C. Gullachsen, M.I.M.E., (*Member*).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): There are several questions I wish to ask about this work, and I also wish to mention several different methods that could have been used in this and similar cases.

Why did the engineers in charge of this shaft decide to sink a six sided shaft? If anyone will go to the trouble to sketch out a six sided figure, and in imagination, picture timber framed accordingly being put in to hold loose ground, it will be at once seen, that it is a very bad form for the purpose, the angle of the joints being much too sharp; a little extra pressure on one side, or, a soft pocket opposite an angle and there is all that is necessary to start trouble.

If a twelve or fifteen sided shaft had been started, with the backs of the timbers built up so that the outer line of the timbering would have formed a circle, there is no doubt but that the shaft would have been carried through the sand without any special difficulty.

Then having come to the conclusion to use the sinking drum method after the timber failed, why did they not fill the shaft with ash whilst waiting? If this had been done, it would have helped to preserve the shaft and formed a bottom for the men whilst doing the new work, thus saving a lot of trouble. Then whatever possessed them to start a sinking drum inside of crushed timbering and railway rails? I think that was looking for trouble with a vengeance.

Another thing—what did they want to fill up that bottom 5 metres for? To form the cement bed? Cutting shoes can be started off planking as was done at Astley Green Shafts, Astley, England. There the tubbing was 3 in. thick, much heavier than at the Norma shaft, the same at Newbiggen Colliery, Northumberland, England, in both these cases, four shafts in all, the cutting shoes and tubbing were started off planks laid on the sand in the bottom of the shafts. Why was the pressure ring made in two portions called the pressure ring and pressure shoe? Why were the anchor bolts run up into the brickwork above the pressure ring? Surely they were not afraid of the brickwork and anchor ring sinking, although it appears so by their shoving 16 rails into the side to help carry them. If they were so afraid of these sinking, why did they not start the brickwork on a cutting shoe and send it down as far as they could as was done at the Hansa-Silberbery Company of Empelde, Nr. Hanover, where they sent the drum down 41 ft. At the Hugo No. 1 shaft a masonry drum was sent down 65 ft., under water. At the new shaft, Thiederhall Potash Salt Works, Thiede, Brunswick, a brickwork drum was sent down as far as needed, that is 73 ft. At the Rhein-Preussen Colliery, Hamburg, on the Rhine, in the No. 4 and 5 shafts, brick drums were sent down 56 ft. and 65 ft. At the Smith shaft in the Swanny district, near Princeton, Michigan, a sinking drum was sent down by its own weight for 40 ft., then weighted with wet sand and sent down to the solid at 60 ft. This shaft was rectangular $21\frac{1}{3}$ ft. × $17\frac{1}{3}$ ft. outside diameter and of reinforced concrete. The Kidder shaft was sunk about half a mile from the Smith shaft, circular in section, 24 ft. outside diameter, a reinforced concrete drum was used which went down 87 ft. by its own weight, it was eventually sent to 113 ft. by weighting with pig iron to the amount of 375 tons.

The cost, £39 10s. per foot, was heavy considering there was no pumping of water or hoisting of debris.

Alternative methods for this work.

To have started with a shaft of twelve or more sides as mentioned before and have taken it down full size for say the first 75 ft., then quartered it down through the other 75 ft. It is justifiable for me to assume that quartering would have answered, seeing that quartering enabled them to free the cutting shoe when it got stuck, judging also from the section and my experience of similar work. Of course timber would have been used as a lining, the joints of the horizontal timbers being well strapped. This shaft would have been circular on the outside but twelve or more sided on the inside, this shape of timbering would, I think, be stronger than curbs as used in English colliery shafts, seeing that in none of the sections of the frame are the fibres cut through.

2nd. To have started with a circular shaft using a brick lining and have quartered it down, as has been done many times in similar ground.

3rd. To have gone down with underhanging tubbing as at Hamsterley Colliery, about 10 miles south-west of Newcastle-on-Tyne, at the Zollern No. 1 shaft, Westphalia, and other places in Germany. Allowing the first timbering to break caused all the trouble, as it allowed the ground to get moving; in all sinking work whether in sand, soft dikes, faulty ground, etc., it should be a first principle to keep all ground outside the shaft in its place, for as soon as holes are allowed to form in the sides of the shaft, the difficulties are greatly increased.

Other methods of doing this kind of work are by wood piling as done at Bowburn Colliery, County Durham and elsewhere. Steel piling as that of The Interlocking Channel Bar Co., of Chicago, consisting of channels and Z (zed) bars rivetted together.

Forepoling or spilling through, a system many of us must have practiced.

Corrugated Iron with strengthening rings inside as used by Kohler in Galicia, shaft 8 ft. clear diameter 65 ft. of sand, sheets of iron 6 ft. 6 in. × 1 ft. 8 in. × ·06 in.

Wrought Iron cylinders strengthened with channel iron rings used by Honingman, two shafts 320 ft., one 425 ft., one 250 ft.

Pipe Piling as used by Hasse at Soessen and Guerini. At Guerini sand was 72 ft. thick, shaft 9 ft. 6 in. × 7 ft. 2 in., 6 lengths of piles used each 13 ft. long screwed together making 78 ft.

The Pattberg Compound Drum, this is inside flanged tubbing with wide flat rings inserted between the horizontal joints, at various distances apart, according to pressure expected, these inserted rings are used to build the masonry on and are bolted one to the other as we use hanging bolts. It was used in the Rhein-Preussen Shafts No. 4 and 5.

Reinforced Concrete Drums on the same principle as that used at the Norma shaft, these drums have been used in America with wood cutting shoes and steel cutting shoes, also rectangular and circular in shape.

The Pneumatic System, this is an application of the principle of the diving bell, used mostly in Belgium and several places in England, not good for great depths owing to pressure of air necessary, say 100 to 120 feet. It can be used either with cast iron tubbing, wrought iron or steel plates with masonry or reinforced concrete lining. Has been used successfully where the tide rose above the level of the surface at site of shaft, 120 ft. is about a limit.

Shield Method, this is a system of working something like tunnel driving with shields and not suitable for depth with very wet ground, as the shield is short and must be open, for it cannot be forced down if closed.

Freezing Systems. Poetsch circulates brine at low temperature through double pipes placed in boreholes concentric with the shaft and about 3 ft. apart, the troubles with this system are deviation of boreholes which sometimes leaves too great a distance between some two holes, so preventing the ground between being frozen solid ; escape of the brine, water running through the ground to be frozen, but has been used in many plans. This method was used at Dawdon Colliery, near Seaham Harbour, in 1903-04, successfully, although the water was brackish and kept in motion by the tides.

Gobert proposed forcing liquefied ammonia or carbonic acid gas instead of brine down the freezing tubes. There are several other systems or combinations of systems, but I think those mentioned are enough.

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THE TESTING OF TRANSVAAL COALS.

(*Read at June Meeting, 1911.*)

By M. Thornton Murray, M.Sc. (*Member*).

DISCUSSION

Mr. Jas. Gray, F.I.C. (*Member of Council*) : Mr. Murray's paper is opportune, and it is therefore surprising to find that it has given rise to comparatively little discussion, especially in view of the fact that the determination of calorific value by the Mahler bomb is assuming such importance in the Transvaal.

The author states that his experiments have been carried out in an endeavour to determine how far approximation may go in the technical determination of the calorific value of coal, and I am sure that his efforts in this direction have been appreciated by his confrères, even though "their voices have been heard but faintly in the land." To my mind the most important part of the paper is that in connection with radiation, and to this part alone I will practically confine my discussion. As pointed out by the author, the neglect to take into consideration the correction due to radiation (which is a variable error) is entirely unjustifiable ; and the assumption that this correction is counterbalanced by the iron wire and acid corrections, is unpardonable. The author adopts the Regnault Pfaundler formula to determine this correction, yet I submit that this formula should have been considered with the others in general use.

For example, Poole* says : "It is not necessary to use the corrections of Regnault and Pfaundler with this apparatus. Newton's law of cooling gives sufficiently accurate results, even in rigorous investigations." Brame and Cowan† remark : "The correction of cooling used was the simple one which Mahler recommended, based on Newton's law, this being sufficiently accurate for almost all calorimetric work where the time between the ignition and the maximum is small." Gray and Robertson‡ state : "The accurate methods of correcting for loss of heat proposed by Berthelot, and by Regnault and Pfaundler, as modified by Stohmann, are much too complicated for technical work."

I would therefore ask the author, in his reply, to institute a comparision of these formulae with the Regnault-Pfaundler. In the second place, I think that the author has been unfortunate in his example of working out the correction by the Regnault-Pfaundler formulæ, and I would suggest revision.

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* Calorific Power of Fuels, p. 60.

† *Journal of the Society of Chemical Industry*, 1903, pp. 1230-3.

‡ *Journal of the Society of Chemical Industry*, 1904, pp. 704-7.

For instance in the initial period $N = 5$, not 6, and therefore $V = -0{\cdot}0026$ and similarly in the final period $N = 5$ and $V_1 = 0{\cdot}0076$ and $V_1 + V = 0{\cdot}0102$, not $0{\cdot}0085$ as stated. Possibly $\Sigma^{n-1}\theta$ should also include 0_0 in which case it would $= 24{\cdot}034$ not $21{\cdot}779$. However, by taking $V_1 + V = 0{\cdot}0102$, the correction for radiation is $0{\cdot}0394$, and the final result 6986 calories, or 25 calories higher than calculated by Mr. Murray.

The latter part of the paper, dealing with the determination of the calorific value from the ultimate analysis, is interesting, but unsatisfactory ; as only one sample of coal is dealt with, no conclusion of any great value can be drawn from the results obtained. The first formula given "Dulong's modified" gives much more satisfactory results than the second, viz : 7,166 calories, and I have also to point out that the author has made an arithmetical blunder in the second case, the correct result being 7,388 calories as against 7439·6 calculated by the author. I shall look forward to the author's reply with interest, in view of the promise to bring forward further data on the subject matter of this paper.

In conclusion, I would again thank the author for bringing this matter before the Society, and giving those interested an opportunity to discuss this somewhat vexed question.

Mr. J. Q. Braidwood (*Member*) : I wish to congratulate the author on his paper which has come at an opportune time, but perhaps a little late. I do not wish to go over old ground which has been already dealt with by other contributors to the discussion, but more as a mark of appreciation than anything else, I wish to say a few words as friendly criticism, as there is no doubt that the paper deserves to be well criticised so as to enable the author to establish his facts. It is to be regretted from the point of view of the discussion that so few members of this Society have had practical experience with the bomb calorimeter.

Grinding.—With reference to grinding through small screens of about 90 mesh, it is in my opinion preferable that the coal should not be ground finer than say 60 mesh. 60 mesh gives ample reliability of obtaining true samples. It is handier to deal with and of course reduces the time occupied by the operation. Whereas a 90 mesh screen might give equally true results for some coals, there is no doubt that it is not so reliable for all coals. The irregularities may be due to the following causes :—

Loss of included gases.
Oxidation.
Loss of some volatile constituents, due to greater heat generated by increased grinder friction, and lengthened period of grinding,

and further the very fine grinding may render the coal more sensitive to surroundings involving the absorption or giving up of moisture.

Formulae.—The formulae used by the author are of a complicated nature, and I do not think such elaboration is necessary for commercial purposes. I would rather see a desire for simplification and the elimination as far as possible of any chance of serious error, than use an elaborate calculation for radiation correction ; I would prefer to spend the time making the conditions such as to reduce that correction to a minimum.

Calorimeters.—For the same reason I would give preference to a type of Bomb Calorimeter on the ground that it gives the least radiation, is capable of rapid handling, and has moving parts such as valves protected against corrosion, and regular stirring by mechanical power, as it is desirable to have the same conditions and method of working at the time of checking the water equivalent as will be used when doing the ordinary test. The Bomb Calorimeter cannot be looked upon as a rule-of-thumb instrument, far from it. It requires constant and undivided attention when in use as it is a most delicate instrument requiring practice and a thoroughly intelligent understanding of its principles in order to avoid errors. I would like to mention while on this subject that it has been suggested by some operators that radiation is neutralised by the formation of acids, but as the quantity of acid formed by any particular substance tested under the same conditions will remain the same, and the radiation will be governed by external conditions which may not remain the same, so that one is a fixed quantity and the other a variable one, therefore one cannot be relied upon to neutralise the other even if by accident it might so happen occasionally. It will be noticed that in his paper the author gives radiation its proper consideration, which is second only in importance to the original rise in temperature.

REPLY TO DISCUSSION.

Mr. M. T. Murray (*Member*) : I have to thank those members who have discussed my paper, as much for their adverse as for their favourable criticism. At the same time a glance at the names of my critics reveals the fact that the majority of those chiefly interested in the subject of coal testing have not contributed to the discussion.

Prof. Wilkinson has very kindly supported the point I raised concerning the different spelling of large and small calories: it is, perhaps, more a chemical question than an engineering one (at least in this country) but I think it certainly merits consideration.

Prof. Wilkinson has also drawn attention to the question of sampling, which I purposely dealt with only superficially in my paper, in the hope that those who were practically concerned in coal-sampling on the Reef, would bring forward their opinions, and describe their methods: that they have not done so, is not for want of an invitation, but argues a bashfulness too rare to be credible.

I should like, however, to reiterate the remarks made at the commencement of the paper, i.e.: "Any inaccurracy in sampling the bins, bunkers, etc., falls on the shoulders of the engineer and not on those of the analyst."

A question so all-important, however, as sampling deserves a paper to itself, and I would suggest that one of our members who is interested in the matter, and whose voice (as Mr. Gray puts it) has been heard but faintly in the land, should supply the deficiency. There is a most interesting theoretical side to sampling, which might throw a light, perhaps all too bright, on the rule of thumb methods at present employed in many cases.

In C. J. and J. Beringer's "Text book of Assaying" (1908 edition), an outline of the theory, as applied to ores, is given, and further data may be obtained from books on such branches of mathematics as "The Law of Error," and the "Law of Probability."

In sampling coal (of an average diameter of particle of $1\frac{1}{2}$ in., say) I think it would be found, from mathematical considerations, that the size of sample necessary to reduce the limit of error below, (say) 1%, would be prohibitive, and that the limit under practical conditions would more likely be in the region of 2% or even 3%.

As long as this is so, it is useless to endeavour to reduce subsequent errors (in calorific determinations, for instance), below this margin, as Dr. Moir has pointed out. The point at issue was, however, concerned with the analyst's fine sample, and it is from the analyst's point of view that the paper was written. It is the analyst's business to test the sample which is submitted to him as accurately as he knows how, not to inquire (beyond the limits of necessity for his testing) into the character and record of the sample. In my paper, be it noted, I have never laid down that such accuracy is practically advisable. Indeed, I am of the opinion that it is not, but so long as analyst's results are to agree within certain narrow limits, great accuracy is necessary.

The question of moisture has raised quite a storm—which will probably clear the air a little: in spite of the evidences of its comparative inaccuracy, I am still convinced that drying at either 95° C. or 105° C. is the best method for ordinary technical work.

So far as testing for heat of combustion goes, the suggestion put forward by Dr. Moir of drying the coal at 60° to 70° C. before testing is scarcely commendable and I feel sure that had he been accustomed to testing coals for heat of combustion, in technical practice, he would modify his opinion considerably.

Mr. McArthur Johnston's contribution to this point is most valuable, as showing the hygroscopic nature of dried, fine coal, but in one instance his figures are unreliable. In his test on the moisture-loss of coal, marked "B," he dried the samples at 94·5° C., and (to quote his words) "at once placed in weighing tubes," obviously the coal should have been *dried in a weighing tube*, in order to avoid risk of moisture absorption during the transference of the hot, dry fine coal (which is admittedly very hygroscopic) to the weighing bottle.

With Dr. Moir's formula for the estimation of heat of combustion from ash and moisture determination, I confess I am not impressed. The ratio of fixed carbon to volatile constituents, alone, in Transvaal coals varies far too widely to make, in my opinion, any such formula reliable. Indeed one case at once occurs to me of two coals *from the same colliery* in which the ash and moisture percentages are almost identical, but which give, respectively, some 13 and 11 evaporative units as heat of combustion.

Dr. Moir recommends the attainment of thermal equilibrium between the calorimeter and its surroundings before experimentation, and suggests that the initial difference in temperature was due to the bomb having been too long in contact with the operator's hands. I can assure him that this was not responsible for the difference: it was merely due to the heat generated by a previous combustion.

With regard to the sulphur allowance—a small matter, and perhaps in many cases, as Dr. Moir suggests, safely negligible—I think that his *reasons* for omitting it would scarcely bear investigation. He says, "The combustion of the iron in pyrite probably only goes to FeO (namely, $FeSiO_3$) in the bomb, against Fe_2O_3 in the furnace, and this loss of heat (to say nothing of the fusion of the ash) will mostly cancel the heat due to burning the sulphur to SO_3." I have made a rough calculation (taking into consideration that the heat of formation of $FeSiO_3$ is some 305 calories per gram—a point which he seems to have overlooked) and find that the total heat generated in the bomb, by the combustion of sulphur and its concomitant iron, would be some 392 calories against 168 calories generated under the furnace—a difference of 224 calories per gram, of which the fusion of the slag might absorb 85 calories, leaving 139 unaccounted for.

Exception is taken to my statement of Mahler's formula, on p. 607. I must confess to a certain ambiguity of diction, which seems to have worried my critic. His strictures in this respect would be perfectly just had I intended the formula to be calculated from coal *plus* ash: the phrase "calculated on an ash and moisture-free basis" would probably have been better than my "disregarding ash and moisture."

Dr. Moir suggests that if iron-wire, sulphur and nitrogen be allowed for, during calibration, subsequent allowances must, to a certain extent, be redundant. The criticism is put in the form of a question, to which I reply that the iron-wire and nitrogen were certainly allowed for, during calibration, but that they were allowed for in such a manner that that subsequent allowances could not lead to duplication.

Dr. Moir deserves the thanks of all who are interested in the question of coal-testing, for his effort to draw up a scheme of operations for technical practice in bomb calorimetry. Since he asks for criticism, however, I venture to offer my quota.

(*a*) *Moisture.*—The method of drying at 60° to 70° C., previous to testing for heat of combustion, is open to some inaccuracy, as pointed out by Mr. McArthur Johnston, while the testing of dried coal does not give practical results, or compels the engineer to dry all his coal at from 60° to 70° C. before using—a lengthy and laborious process—to obtain concordant results.

(*b*) I think I am voicing the opinion of analysts generally, when I say that it is impossible, in technical practice to afford the time for bomb and water to attain thermal equilibrium, when a number of samples are to be tested in a short time.

(*c*) I would point out that my paper did not lay down a method of procedure to be followed; it merely aimed at discovering some datum line from which subsequent approximations might be made.

(*d*) This method of calculation of the radiation loss has been proved to be inaccurate in the paper and elsewhere, and must especially be so, where the thermal equilbrium cannot be obtained before the commencement of the experiment.

(*e*) The subtraction of a constant figure for nitric and sulphuric acids, two of which are variable corrections, appears to me inadvisable, while the figure is, in my opinion (based on a number of determinations on Transvaal coals) much too low. Again, the different nature of the materials to be tested—"fat," or "lean" coals, low-grade boiler-slack, and even ashes would give very widely varying results. Mr. Gray objects, naturally enough, to the cumbersomeness of the Regnault-Pfaundler formula. I have already stated that I consider it unsuitable, in practice, as it stands, and only useful as a basis for subsequent approximations. Mr. Gray has kindly given me references to other formulæ; I will return the compliment by saying that in *Metallurgical and Chemical Engineering*, Vol. IX., No. 9, p. 449-452, the question of cooling correction is fully discussed, and a method of approximation proposed which seems feasible.

I have compared the formula used by Gray and Robertson, with the Regnault-Pfaundler, on their temperature readings, and my own. In the former case I find the correction obtained by it was ·005, while the Regnault-Pfaundler gave ·0104, while in my own test this formula would have given 0·0181, as against 0·039 by the Regnault-Pfaundler—in each case only about one-half of the more correct figure. I think this formula, at least, must be ruled out of court.

Mr. Gray has discovered two arithmetical errors in my paper, for which I apologise. They were, in each case, clerical errors during transcription, but the pressure of time under which the paper was written only partly excuses them.

Dr. Moir, in discussing the Regnault-Pfaundler formula, says: "This formula cannot, as the author says, take account of evaporation, else it would contain a factor involving the latent heat of steam. If Dr. Moir had read my paper a little more carefully before criticising this point, he would have discovered that I said: "take into consideration *the cooling effect due to evaporation*, and any other constant heating or cooling effect,"—which is a very different thing.

I have to thank Prof. Wilkinson for a useful elaboration of many of the points which I was unable to touch upon at length, and to him, as well as to the rest of my critics I again offer my thanks for their contributions to the discussion.

NOTES ON THE TREATMENT OF MILL CONCENTRATE.

By R. Lindsay (Member).

DISCUSSION.

Mr. Jas. E. Thomas (*Member of Council*): It is to be regretted that Mr. Lindsay's interesting paper on the treatment of mill concentrate has not drawn forth more discussion, more particularly as it offered an opening for contributions from our younger members who are actually engaged in the operation of treating this product.

At the Simmer Deep-Jupiter joint reduction works the preliminary operations are much as described by Mr. Lindsay with the exception that lime is added to any material ground in the amalgam barrels, as advocated by Mr. Dowling,[*] to prevent the formation of gases which might cause an explosion on opening the barrel. Each charge is ground for 18 hours and washed over a Batea, the pulp being allowed to settle in a vat placed at the foot of the ordinary amalgam cleaning table. The product thus collected is pumped to a small tube mill plant consisting of a cone 30 in. diameter at the overflow and 42 in. deep, a tube mill 4 ft. 6 in. diameter 6 ft. long together with two small stationary amalgamating tables having a grade of 18%. The underflow from the cone containing about 42% moisture passes through the tube mill, which runs at a speed of 30 r.p.m., and over the plates, and is re-elevated to the cone classifier, the overflow of which enters two vats 10 ft. diameter 8 ft. 6 in. deep with conical bottoms 18 in. deep connected to a 4 in. centrifugal sludge pump. The whole operation is therefore similar to ordinary tube mill practice on a large scale, the coarser particles being reground until fine enough to pass away with the overflow. This arrangement dispenses with the Batea mentioned by Mr. Lindsay, which seems a somewhat cumbersome and costly arrangement to instal and run, any mercury or amalgam not caught at once on the tables being returned through the cone to the tube mill circuit.

As no compressed air is available, treatment by cyanide is carried out by circulating, transferring, settling and decanting as in an ordinary ore-slime treatment plant. Each charge contains about 8 tons of concentrate. This is given 8 washes of 0·07% KCN solution of 15 tons each, followed by two water washes of the same amount, sufficient lime being added to ensure settlement. The residue is discharged to a small dam and the water allowed to evaporate. It is then transferred by native labour to the circuit of the continuous sand collecting plant in which it finishes treatment in the ordinary manner.

The total time occupied by treatment in the black sand plant is 8 days, the decanted solution gravitating to the extractor boxes where it mingles with the richest solution from the usual sand treatment. The tube mill and 2 in. centrifugal elevating pump consume about 8 h.p. and run for 48 hours on an 8 ton charge. The 4 in. circulating and transferring pump is run for 9 hours each day during treatment and requires 4·25 h.p. As the whole of the operations are carried out by the men ordinarily employed in the mill clean-up room no special charge is made against the plant for attendance.

The following are the average results obtained during the last seven months :—

Before treatment by cyaniding ...	90·0 dwt.
After " " " ...	2·60 "
Theoretical extraction	97·1%

Grading.	+60	−60+90	−90+200	−200
	0·2%	0·2%	1·0%	98·6%

No assay is made of the concentrate before tube milling as it is considered impossible to obtain a correct sample owing to the presence of amalgam. The average amount of amalgam recovered from each 8 ton charge is 110 oz.

In conclusion I desire to thank Mr. T. A. Malpage, Acting Manager, Simmer Deep, Ltd., for permission to publish these few remarks.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Assay of Tin Ores.—" It is a curious fact that for the last 25 years tin-assay methods have made little or no advance, while those now in general use in connection with copper and lead ores can scarcely be compared with those then obtaining, so greatly have they been improved upon. Though it is not altogether expected that the method herein advocated will at once meet with general approval, the writer hopes the resultant discussion may bring out points that have hitherto escaped detection.

The old Cornish method of smelting with anthracite is now obsolete. The cyanide assay, although unreliable, is still much used because of its simplicity.

In the Malaya many buyers of tin ores use the cyanide assay, the smelting being carried out generally in a benzine furnace.

It is essential that the smelting with cyanide be performed at a low temperature. Even then, impurities, such as Sb_2S_3 and Bi_2S_3, if present, are reduced, rendering the button of tin impure—

$$Sb_2S_3 + 3KCN = Sb_2 + 3KCNS$$

Oxides of iron are readily reduced, the iron going in with the tin, and if the button contains impurities the percentage of tin in the button must be determined by wet methods ; therefore it is desirable to use a wet method, and so avoid the doubtful fusion connected with the cyanide process.

In technical practice, volumetric methods are preferable, being quite as accurate and much more rapid than gravimetric methods. Consequently this paper will be confined to discussing volumetric methods. Many schemes are in use, but professional men appear to differ as to which is the most practical and satisfactory volumetric method The two in general use at the present time are :

1. Titration of stannous chloride with standard iodine solution.
2. Titration with a standard solution of ferric chloride.

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[*] See this Journal, Vol. XI., Dec. 1910, p. 238.

Both these titrations give satisfactory results, but the writer prefers No. 1, as the operations of preparing the assay solution are simpler.

The following method, which is a modification of that commonly known as the Pearce-Low scheme, was found to give very satisfactory results. It is rapid, and answers every ordinary requirement:

Place from 6 gm. to 8 gm. of stick sodium hydroxide in a thin-spun iron crucible provided with an iron cover, and gently heat to expel moisture. Weigh out about 5 gm.* of the finely powdered ore and add to the iron crucible after the melted hydroxide has cooled sufficiently to permit of its setting. Cover the crucible, and again heat, gently at first, and afterwards to dull redness, until fusion is complete. Remove the cover and allow the 'melt' to cool, placing the cover in a 5¼ in. casserole, or in an ordinary porcelain evaporating basin. When the crucible has cooled, place it on its side in the casserole, add a little water, and heat to boiling point. Next, loosen the 'melt' with a glass rod, wash the crucible well until all the 'melt' has been removed, and then lift the crucible from the casserole, giving it a final rinse: Rinse the crucible with a few cubic centimetres of weak hydrochloric acid to insure that absolutely all the 'melt' has been removed, adding the acid to the solution in the casserole, which must be covered to prevent loss by effervescence. Remove the crucible cover from the casserole, and if any globules of the 'melt' are still attached, remove them with a dilute acid wash.

It is important to keep the washings in the above operations as small in volume as possible.

Next add strong HCl, a few cubic centimetres at a time, keeping the casserole well covered until effervescence ceases. Then add 20 cc. of strong HCl—in excess, heat to boiling point and transfer the solution to a suitable flask. No undecomposed residue should remain in the casserole, except perhaps a little gelatinous silica. Add three horseshoe nails to the solution, the nails being bent to form the letter U. Boil for 15 minutes, and filter through glass wool, washing the filter with about 50 cc. of distilled water.

Horseshoe nails, being made of a very pure and soft iron, act efficiently in removing Cu, Sb, As, etc., and at the same time minimise the risk of introducing other impurities.

To the filtrate, which has been caught in a 6 oz. conical flask, about 5 gm. of finely powdered antimony is added. The solution is then heated to boiling point, and the boiling is continued for five minutes. The flask is next cooled in an atmosphere of CO_2 and titrated with a standard solution of iodine, using a starch solution as an indicator. It is essential that the solution, after boiling with antimony, should be cooled as rapidly as possible, and to accomplish this the flasks can be placed in a tray and cold water circulated round them. As the stannous chloride oxidises very rapidly to stannic chloride, especially when the solutions are hot, the solution must be cooled in an atmosphere of CO_2. This may be done by placing a pea of marble in the flask. This last instruction cannot, however, be recommended, as the results are generally low probably owing to the introduction of air.

When doing a number of determinations it is better to cork the flasks during the operation of boiling, the corks being fitted with a delivery tube. On removing the flasks from the hot plate, connect the delivery tubes to a constant supply of CO_2.

J. J. Beringer, in an article written to the *London Mining Journal*, advocates the use of zinc fume or clippings for reducing the tin compounds, in place of fusion with sodium hydroxide.

The writer has used this method and found it to be an efficient one, giving results which agreed well with the method already described under the heading of "Method Advocated."

The hydrogen reduction method championed by Parry, and claimed by him to be *the* method, is, in the writer's opinion, cumbersome, and decidedly inferior to the above methods for accuracy, simplicity and speed.

Many methods are in use for reducing the stannic solutions to the stannous state ready for titration.

Iron horseshoe nails are effective if the solution is kept strongly acid and boiled for at least 25 minutes.

Sheet nickel may be used in place of iron, but it is expensive, and the strong colour imparted to the solution tends to mask the appearance of the first tinge of starch iodide, and this is objectionable when testing for small amounts of tin.

Zinc reduces tin chloride to the metallic state, and it can then be filtered off and redissolved in strong HCl. Experiments showed that while being redissolved there was a tendency for the tin to oxidize to stannic chloride, and in this one is obliged to assume that the tin dissolves wholly as stannous chloride. As this assumption is not justified, the method is quite unreliable. The writer considers the best reducing material to be finely powdered antimony, which is effective and rapid, and the excess of Sb that remains does not interfere with the iodine titration, which is performed in the cold. It is essential that the antimony should be extremely fine and free from interfering metals. Kahlbaum's pure pulverulent antimony answers well, the rapidity and completeness of reduction well repaying the cost of the pure metal.

In the ferric chloride method of titration the strong yellow colour of ferric chloride indicates that the conversion of the stannous chloride to the stannic condition is complete. With this method antimony cannot be used as a reducer, metallic zinc being generally employed, and as pointed out above, this is open to error. The titration must be performed above 70° C., and at such a high temperature it is difficult to prevent oxidation.

After a considerable number of experiments, the writer is of the opinion that fusion with NaOH in an iron crucible, followed by reduction with iron nails and antimony and the iodine titration, gives the most accurate and consistent results with the minimum of trouble.„ R. G. MORGAN, *Australian Institute of Mining Engineers.—Mining Science*, August, 1911, p. 150-1. (A.R.)

METALLURGY.

EXPERIMENTS ON MINERAL FLOTATION. "*Flotation in Water.—Cold Water.*—Many minerals when dropped carefully on to the surface of water remain on it and do not sink; some at once sink to the bottom. Some minerals, when ground to pass a 30 mesh sieve, form a film on the surface of the water, and this film will support a considerable weight. A certain sample of zinc-blende concentrate could be heaped up in this manner to a depth of ¼ in. on the surface of water in a 50 cc. beaker. The weight of material was 15 gm., and the top portion was quite dry. The following is a list of the minerals tried that would float on the surface of water: Galena, blende, sulphur, azurite, malachite, rhodonite, garnet, calcite, mica, telluride of gold, pyrite, pyrrhotite, wolframite, cassiterite. All the metals in the form of foil would easily float. Some samples of quartz will float and others will not. Particles of glass and turquoise would not float.

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* Probably 0·5 gm. is meant, Ed. Com.

The weight of each particle on the surface causes a distinct depression or sag round itself, and the apparent attraction of one particle for another is probably due to the depressed surface caused by this sagging. If alcohol in the form of a drop on a glass rod be brought near the particles floating on the surface, the particles are repelled from the rod. If alcohol is added to the water the particles become wetted and sink. Caustic soda acts similarly. This is apparently a surface tension phenomenon. In the case of alcohol on the glass rod, the vapour of the alcohol mixing with the water decreases the surface tension and increases the adhesion of the water for the mineral. The water wets the particle at a higher level on the side nearest the glass rod; an inclined plane is formed on the side furthest from the rod, and the particle is apparently repelled. Water has the greatest surface tension of all liquids, under ordinary atmospheric conditions, except mercury, and thus, if another liquid is added to it, the surface tension is decreased. If pulverised mineral is wetted and covered with water, and then is exposed to the air by causing the water to run to one side of the containing vessel and the water is then brought carefully back, some of the mineral will be seen to float on the surface of the water. This form of flotation takes place to a greater or less extent with most minerals, and is apparently a different phenomenon from the simple floating of particles by dropping them on the surface. In the latter case there is an almost continuous film of air surrounding the under side of the mineral, and by far the greater portion of it is projecting above the surface of the water. That the film is not continuous and that the particle is partly wetted, can be shown by floating some magnetic substance, as pyrrhotite, and attracting it with a magnet, when the surface of the water will be dragged up with the mineral. When the substance is finally lifted out, water is carried up to the magnet. In the former case, the particles will be seen to be almost entirely submerged, only a portion like a pin's point being above the surface, and, although, a distinct depression of the surface is caused, it is not so deep as in the other case.

Hot Water.—In hot water the behaviour of minerals is different. Most of them will float for a few seconds only, becoming wetted and sinking, usually a bubble of gas leaving each particle as it sinks. Fine grained material can be heaped up as with cold water, a large number of bubbles being observed on the under surface of the mass.

Under Water.—When minerals are submerged in water and the water is heated, many of them will float. Sulphur, crystalline azurite and malachite, some galenas, some blendes, some chalcopyrites and stibnites, and some of the metals, copper being the most persistent, will float in water on heating. A sample of zinc concentrate from the Potter process (floated with H_2SO_4) floated when heated in water, and when boiled the 'scum' continued indefinitely. If boiled violently and continuously, the concentrate sank, but if allowed to cool they again floated on heating. This was repeated 12 times, and would apparently continue indefinitely, if after each test the water were allowed to cool for some time. Air will get into boiling water, being either dissolved or carried down mechanically as bubbles. A large beaker of distilled water was boiled continuously for two hours. An inverted funnel was then placed in the beaker and an inverted burette filled with the boiled water was placed over the stem of the funnel. Air then slowly collected in the burette. In some cases minerals appear to evolve a gas, and usually the water begins to boil and water vapour to form first on the minerals introduced into it.

Reduced Pressure.—Many substances when sunk in water will rise to the surface when the pressure above the surface of the water is lowered. Among the substances tried the following floated: All the metals in the form of foil, sulphur, graphite, mica and zinc blende concentrate.. Particles of galena, zinc blende, cerussite, river sand, and calcite came to the surface, but sank again. When a mixture of particles of zinc blende and river sand was put into a vacuum flask under water under reduced pressure, it was noticed that the zinc blende came up more persistently than the sand, which rose to the surface three or four times, but would not do so again. Particles of mineral are caused to float by gas bubbles attached to them. In some cases these bubbles are strongly attached to the minerals. On some minerals bubbles will form, but the minerals do not float, as quartz, garnet, gypsum, hematite, wolframite, cassiterite and glass. Pieces of metal in the form of foil will float even after being repeatedly sunk. If the water is boiled free from air the metals will not float if the first bubbles to attach themselves are removed. A piece of copper foil, immersed in air-free water, which would not float when the pressure was reduced, floated on the surface when brought in contact with the vapour above the surface of the water. Crystalline pieces of the following minerals, under water and with reduced pressure, were all coated with bubbles: Galena, blende, garnet, tourmaline, pyrite, gypsum, andalusite, calcite, cerussite and quartz. These minerals were then placed in water boiled free from air, and again became coated with bubbles on reducing the pressure, thus showing that some gas is carried down with each mineral as it sinks. The metals also had bubbles attached to them, but on freeing these with a glass rod only a few very minute ones formed again. After the bubbles have formed on these minerals, if atmospheric pressure is restored, the bubbles become almost invisible, but expand again in the same position on again lowering the pressure. Tarnished minerals form as many bubbles on their surfaces as the clean ones.

Flotation in Dilute Acid Solution.—Cold.—Similar to water even with strong H_2SO_4.

Hot.—None of the minerals would float when sprinkled on the surface of a hot H_2SO_4 solution, even when very dilute (1%), but became at once wetted and sunk. Copper foil will float on a hot, 3% H_2SO_4 solution. Zinc concentrate forms a film the thickness of one particle only, the remainder falling through the surface, but coming up again in the form of agglomerated masses.

Heated in Acid Solutions.—Most of the sulphides will float if heated in a weak H_2SO_4 solution. Some samples of pure galena and zinc blende would not float, but on the addition of calcite and generation of CO_2 they floated. Hematite, biotite, quartz, wolframite, cassiterite, rhodonite, calcite, siderite, muscovite, serpentine, would not float. With the sulphides, H_2S is given off to a greater or less extent in every case. A sample of copper matte would not float in acid solution. To show the amount of acid required a small test with a sample of the South Mine, Broken Hill, tailing was carried out, taking 10 gm. of the tailing, 50 c.c. water, different amounts of acid, and heating to about 95° C.:—
0·36% H_2SO_4; no float. 0·72% H_2SO_4; small scum formed. 0·90% H_2SO_4; poor float. 1·40% H_2SO_4;

good float. This tailing contains much calcite, and therefore uses up more acid than most of the other companies' tailings.

Reduced Pressure.—The same lot of crystalline particles tried with water alone were tested under reduced pressure in an acid solution ranging from 1 up to 10% H_2SO_4. The bubbles formed as before, but were of a different nature, appearing to be larger and coming away from the minerals more readily. Fewer bubbles formed on the quartz and gypsum than on the other minerals. In the case of quartz and rhodonite and the other silicates, a few bubbles were carried down, but on shaking these off no others formed. This also was the case with copper foil. Bubbles formed on the sulphides, garnet and tourmaline even after those first formed were detached. Some cubical fragments of galena were coated with lead chromate and sulphate, respectively, and immersed in an air-free acid solution, under reduced pressure. Only a very few bubbles formed on these particles. Most of the sulphides will not float in the cold with dilute acid. The zinc blende in a sample of South Mine tailing kept coming up to the surface, but fell back again. The carbonates are attacked by very dilute acid solutions under reduced pressure. It is interesting to note that siderite and magnesite are attacked by cold acid under reduced pressure.

Some Broken Hill Proprietary's zinc concentrate, washed free from soluble salts, was boiled in a 2% H_2SO_4 solution. The following metals went into solution, Fe, Al, Zn and Mn, while H_2S was given off. A sample of the Central mine's concentrate (oil and acid process) was washed with strong NaOH, and then washed free from NaOH. The zinc blende would not then float on boiling air-free water, but when made acid with H_2SO_4, a float was obtained. No float could be obtained by adding ammonium carbonate. Another sample of the Proprietary's zinc concentrate washed free from acid, then washed with strong NaOH and finally washed free from alkali, would not float on hot water, but floated on adding acetic or sulphuric acid.

Flotation of Oils, etc.—None of the minerals tried would float on the surface of oleic acid (commercial), kerosene or alcohol. Copper foil floated on oleic acid, but not so persistently on kerosene. Iron wire would not float on kerosene or oleic ocid, but showed a tendency to float, not sinking immediately. Zinc blende, galena and the other minerals would not float on alcohol, but did not immediately sink.

Hot.—None of the minerals floated when heated in kerosene or oleic acid.

Reduced Pressure.—None of the minerals or metals floated when immersed in oleic acid, alcohol or kerosene under reduced pressure.

Behaviour of Oiled Minerals in Water and Acid Solutions.—Nearly all the minerals and metals will absorb oil if shaken in water to which oil has been added. If previously oiled with oleic acid, vaseline or kerosene, nearly all minerals will float in water. Some float in the cold; others require heating or reduced pressure. The following float well: Calcite, sulphur, galena, garnet, zinc blende, wolframite, cassiterite, siderite, cerussite, magnesite, azurite, malachite, rhodonite, some quartz and the metals in form of foil. The following give no float, or a very poor one: Muscovite, amorphous malachite, turquoise, gypsum, quartz and particles of slate. Calcite and muscovite if crushed together can be separated by oleic acid, the calcite floating and the muscovite not absorbing the oil. Copper matte will float with oleic acid in H_2SO_4 solution. Whether a mineral is caused to float or is merely collected together in a mass or in small rounded balls or is made granular but coherent, is dependent on the quantity of oil used. A series of tests showed that with an excess of oil over the mineral the latter was carried to the surface of the water by the lighter oil. With oil and the mineral in certain ratios, as 5 gm. oil to 20 or 30 gm. of zinc blende, the oil and mineral formed one globular coherent mass. By increasing the proportion of mineral, smaller rounded masses of oil and mineral formed which would not float; finally with 150 gm. of mineral and 5 gm. oil a granular product was formed which would float on violent agitation, being buoyed up by attached air bubbles. On adding acid to the minerals which have absorbed oil in the presence of water, most minerals, other than sulphides and metals, will not float when heated, the acid apparently causing them to separate from the oil. The amount of acid required to make the action selective is very small, but from the results of numerous tests it is apparent that with only minute quantities of acid the action is only partially selective. If more acid is used the freer from gangue the concentrate will be. Calcite and oleic acid will not mix in the presence of oxalic acid, although the calcite is not apparently attacked. Calcite will absorb oleic acid in the presence of acetic and lactic acids. There is no very marked difference between the behaviour of sulphur and oleic acid in an acid solution or in water. Carbon bisulphide acts like an oil, and is apparently absorbed by the minerals.

Reduced Pressure.—Crystalline fragments of tourmaline, garnet, blende, galena, pyrite, quartz, gypsum, andalusite and calcite were taken, two of each, one oiled with oleic acid and the other unoiled. These were immersed in water, and the pressure reduced. Bubbles immediately formed on the oiled minerals, and on further decreasing the pressure, bubbles larger and in greater number formed on the oiled than on the unoiled minerals. The bubbles on the oiled and on the unoiled minerals were different, those on the oiled being of an inverted watch-glass shape when first formed. On adding H_2SO_4 the bubbles changed in character, and most of the bubbles left the unoiled quartz and gypsum, but not the other minerals. At the lower pressure CO_2 was evolved from the calcite, which was not attacked at first, showing that the acid solution gets under the oil film on the mineral. With a lens minute specks could be seen on the faces of the pyrite crystals which gradually grew into larger bubbles on reducing the pressure. The bubbles on the oiled minerals were of a more permanent character and the manner of leaving the minerals was also different. With the oiled minerals the bubbles become distorted and dragged away more than with the unoiled minerals. On warming the solution, the bubbles clung much more tenaciously to the sulphides than to the other minerals, the quartz, calcite and gypsum being almost free from bubbles.

If a layer of oil is floated on water and a particle of mineral, as galena or quartz, is dropped on it, the mineral will sink only as far as the top of the water and remain there. Minerals will sink through alternate layers of different oils floating on water, and will remain floating on the top of the water. If a particle of some mineral is sunk in water and a layer of oil is floated on top of the water, and if the pressure be reduced above the surface of the oil, the mineral particle will rise as far as the surface of the water, but will not penetrate the oil layer. Three beakers were taken, each containing a saturated CO_2 solution. A layer of oleic acid was added to form a layer over the the surface of one, kerosene in another and petrol in the third. Particles of different minerals were then dropped into the beakers, and their behaviour noted. All the minerals carried down oil with them in the form of a circular "blob." CO_2 gas immediately began to collect under the surface of the oiled particles in the form of very small bubbles, which collected to form one large bubble; on becoming too large it broke away and came to the surface, the oil returning to its original form until another large bubble formed. On adding H_2SO_4 many bubbles, and apparently the excess of the oil, left the minerals, and after standing or heating, quartz, calcite and gypsum became practically free from bubbles. The sulphides, metals and garnet still had bubbles collected on them, but in a different form, each bubble being separate and not combining with the others to form one large bubble under the surface of the oil as was the case with water alone.

A very small amount of oil only is required to oil a considerable amount of any mineral. Some South Mine tailing weighing 24 gm. was put into water, 1 drop (0·026 gm.) oleic acid added, the mixture shaken and put in a vacuum flask. On reducing the pressure the whole product, metallic and nonmetallic, floated to the surface. With 40 gm. only about half was raised.

Flotation in Alkaline Solutions.—Most minerals will float on the surface of NaOH solution, but for a few moments only, and then become wetted and sink.

Hot—When heated in a solution of NaOH, the minerals will not float. On adding an ammonium salt, although gas is given off, no float takes place. With some zinc concentrate that had been previously floated, a good float was obtained in an ammonium carbonate solution. With NH_4Cl partial and nonselective floating was caused. Oleic acid combined with ammonia will cause nonselective flotation, and on adding H_2SO_4 till acid, the silicates will sink and the sulphides remain floating. A sample of zinc concentrates would not float in NaOH, and on addition of H_2SO_4 till acid would float only after heating for a considerable time.

Reduced Pressure. — Bubbles collected on the minerals in NaOH solution under reduced pressure, the solution being free from air. On making acid with H_2SO_4 very few bubbles remained on any of the minerals, but more were on the sulphides than on the quartz and silicates.

Flotation in Saturated Solutions of Gases.—Copper foil and other metals and also all the minerals constituting the Broken Hill tailing will float in water saturated with CO_2, being brought to the surface repeatedly by the gas bubbles. Bubbles form on all the minerals, and are not confined to the sulphides. Very few form on gypsum and fewer on the quartz than on the sulphides. On addition of H_2SO_4 no bubbles remain on the gypsum and very few on the quartz, the garnet still having bubbles attached to it. The action is similar with a saturated solution of SO_2. Hydrogen peroxide is also similar in its action to the saturated solutions. The manner in which bubbles of gas are formed in a solution is important as regards their subsequent attachment to the minerals. A rapid current of bubbles may be evolved, but the bubbles may not attach themselves to mineral particles, and thus may not be of any use in their subsequent flotation. Sometimes the rapid evolution of a gas is useful, as in the case where flocculent masses of the sulphides are held together by gas bubbles insufficient to carry them to the surface; the rush of bubbles underneath these agglomerated sulphides causes them to be carried to the surface. The condition that seems to be best for the attachment of gas bubbles to minerals and metals is when a saturated solution is slowly evolving bubbles of gas that are being thrown out of solution by rise of temperature or lowering of pressure. This takes place to a certain extent in all the flotation processes where a hot solution is used, the dissolved air, CO_2, H_2S and any other gases in solution being driven out when the temperature is raised. In the case of saturated solution of gas, bubbles will form on almost anything introduced into the solution. On addition of acids the bubbles have a tendency to select the sulphides on which to form.

Film Flotation.—Film flotation is the term used to describe flotation when the mineral floats on the surface of the liquid to the depth of one particle only, and not as a coherent scum as in the case of the Potter and other processes. In film processes, as the De Bavay, the sulphides, after being wetted, are brought into contact with the air and fed on to the surface of water, where they remain floating as a film and not as a scum. A sample of zinc blende obtained by film flotation (using H_2SO_4 and oleic acid) was immersed in boiled water and boiled under reduced pressure for three hours, as much of the air being pumped out as possible. All the mineral was then on the bottom of the flask. The top of the flask was then closed, and on bringing the mineral into contact with the vapor above the water, the film was again formed. When examined with the aid of a lens the film was seen to be almost entirely submerged, only fine points of the particles projecting above the surface. A film of magnetic sulphide of iron on the surface of water, when attracted by a magnet, drew up the surface of the water with the mineral. Copper foil will not float in air-free water under reduced pressure, but if allowed to come in contact with the vapor above the water it will float.

Effect of Tarnishing.—If the surfaces of sulphides are tarnished, as a rule these sulphides will not float. Taking the Horwood process as an example, the mixture of lead and zinc sulphides is given a slight roast, which is sufficient to cause the galena to become coated with lead sulphate. The zinc blende is not much altered at the low temperature at which the roasting is carried out. When the material is then heated with an oil and acid solution only the zinc sulphide floats. If galena is coated by treating with strong H_2SO_4 it has a tendency not to float; chromic acid has the same effect. Chlorine and nitric acid, which rapidly tarnish the surface of galena, do not in any way prevent flotation. but on the other hand have a beneficial effect. This may be due to the liberation of free sulphur (which does take place) on the surface of the sulphide, sulphur being one of the most easily floated substances. Some ores that will not float in their natural condition will float after being heated, the slight calcining having a beneficial effect on the flotation. This is noticeable with some of the heavy Tasmanian zinc-lead ores.

Summary.—All minerals absorb gases. Sulphides appear to absorb CO_2 naturally. The consequence of this is: (1) Particles are not wholly wetted when immersed in water. (2) Particles also tend to float when sprinkled on the surface. (3) When immersed in water and heated, the air or gases dissolved in water will collect on the particles, and float or tend to float them to the surface (4) Saturated solutions of gases in water evolve gas which collects on all particles.

The Addition of Acids.—Metallic particles, such as sulphides and metals, when immersed in dilute acid solution, are not wholly wetted, but particles of rock materials become more readily wetted and give off their absorbed gas. When different dry minerals are sprinkled on the surface of dilute cold acid, the tendency for them all is to float as in water. The minerals are more readily wetted by hot acid solutions than cold.

The Addition of Alkalis.—When different mineral particles are immersed in an alkaline solution, they tend to part with their absorbed gases, and will not float even when heated. When sprinked dry on to the surface of the solution, they remain for a few seconds only and then break through.

Flotation of Sulphides.—Given a starting point, any gas generated in or introduced into a solution will tend to collect on the surface of particles. If acid is added, the gases will leave the gangue particles and collect on the sulphides; the latter, therefore tend to float and the former to remain under the surface of the liquid. When substances in the solution, such as carbonates, are attacked by acids, the gases evolved saturate the liquid and commence to grow on the sulphides. Sulphurreted hydrogen, steam, air and any other gases either present in the solution, introduced into it or evolved from the minerals present, assist flotation, provided the sulphide particles have dry points on them for the gases to become attached to. Some sulphides naturally absorb gases, and such sulphides readily float. The bulk of Broken Hill ores may be taken as examples. Other ores require preliminary treatment, so as to alter their surface to some extent. This may be done in certain cases by merely heating to 250° C., or immersing in some solution which will attack the sulphides, such as nitric acid or chlorine solution Sulphides which condense or adsorb gases well will gather together in a solution as flocculent masses, while those that do not, remain individually distinct, like grains of sand.

Heat an Aid to Flotation.—In certain cases sulphides will float in acid solution at ordinary temperature, but for the most part, with dilute acid, flotation starts at about 80° C. The explanation put forward that flotation at that temperature is due to the fact that certain carbonates, such as rhodochrosite, siderite, smithsonite, are attacked and give off CO_2, does not appear to be correct, for under reduced pressure these minerals are attacked but flotation does not take place. The presence of silicic acid and sulphur, as stated by De Bavay as necessary for a coagulum, is not necessary in many cases of successful flotation. When minerals are treated with dilute acids, and then exposed to the air, certain sulphides absorb gases on their surface. If they are alternately wetted and exposed to the air, they will become entrapped by the surface film of water with which they are brought in contact. The appearance of the film is the same as that of a bubble of gas under the surface of a solution with sulphides attached. The sulphides in both cases adhere to the surface film of the bubble or free surface of the liquid, but are mainly below the surface itself. The gas obtained in all cases from experimental work, and also from samples obtained from various flotation plants, was CO_2, with various quantities of nitrogen and oxygen. There is no doubt that most of this gas is generated from gangue particles in the ore, and being more soluble in the liquid than the other gases, collects readily on the surface of the sulphides.

In all experiments with sulphides, carbon dioxide was obtained on exhaustion. This gas must have been present either in the form of a compound easily decomposed by heating or by reducing the pressure, or as a gas condensed on the surfaces of the sulphides and there retained with greater force than other gases present. It was found that the carbon dioxide was the last gas to come off.

Effect of Oils.—The effect of oils has long been known in protecting minerals from being wetted by water, owing to the oiling of their surfaces, and also the affinity of oils for metals and many metallic substances. Nearly all the minerals tried could be coated with oil either in the presence or absence of water. This seems to be connected with the adsorption of gases on the particles themselves, for if the particles are deprived of their gas, then as a rule they do not become coated with oil. It would therefore appear that the oiling of particles is dependent on their gaseous attachments. If the particles are oiled to a minute extent, the gases previously absorbed are retained, and if a solution is heated more gas will become attached to the particles, and flotation will ensue. An oil float may be obtained without acids. Nearly all minerals will float after being oiled.

The flotation is apparently due partly to the lesser density of the oil and the presence of gases absorbed on the minerals or entangled in the oil. If a large quantity of viscous oil is used gases become entangled in it and the product is oily. In course of time the particles will drop away from it, carrying down attached globules of oil. When a lesser amount of oil is used the product becomes a coherent mass like putty, showing no apparent tendency to float. On further decreasing the quantity of oil, the product becomes firmer and breaks up into rounded pellets. With a still less quantity of oil, when shaken up with water in presence of air the product becomes flocculent and increases in volume. This flocculation is due to the absorption of gases. With very minute quantities of oil the flocculation becomes more pronounced, and the tendency to float becomes greater. These results may be obtained with gangue and metallic substances. Gases collect more readily on oiled than on unoiled particles.

Acid gives Selective Floating.—Effects of Acids on Oiled Particles.—The addition of acid causes a more selective action. If oiled gangue and metallic particles are immersed in water, and acid is added, then the oil will tend to leave the gangue particles and become attached to the sulphides. The action of the oil in this respect is similar to that of the gases attached to different minerals, and probably is due to the same cause.

If an ore is treated with an acid and the acid is removed, and if the particles are then oiled, the oil tends to become attached to the sulphides only. This is probably due to the removal of the gas from the gangue, which prevents oiling. If acid and oil are added together, the action of the oil is selective, as in the previous case. The presence of oil on their surface does not prevent minerals from being attacked by acids.

The amount of acid required for selective flotation depends on the composition of the ore, the temperature and the pressure existing at the time. Generally it may be stated that an increase of acid will give a purer sulphide product, but flotation will take place at a lower temperature, and with a lesser amount of acid, with oil and acid than with acid alone.

A selective flotation can be obtained in some cases by treatment with a chlorine solution previous to the addition of oil, and without the addition of acid. Some of the light volatile oils will cause flotation in an acid solution at a lower temperature than the heavier and thicker oils. With some Broken Hill tailing a flotation takes place at ordinary temperature with sulphuric acid and a volatile oil.

Effect of Oils and Alkalis.—Flotation can be obtained in alkaline solutions when oil is used, but in every case tried the action was nonselective. On making the solution acid the action becomes selective, only the sulphides floating.

The flotation of mineral particles appears then to depend on the fact that the particles having gas attached to them become entangled, or caught, by the surface film of a liquid. This film may be (1) the free surface of a liquid, (2) the surface of a bubble of gas under the surface of a liquid, or (3) the surface of a bubble of gas above the surface of a liquid. Once the particles are entangled in this film they are sustained there. That a considerable weight may be supported by this surface film is shown by the experiment of floating fine-grained mineral on the surface film of water."—KENNETH A. MICKLE, *Royal Society of Victoria*: *Engineering and Mining Journal*, August 12, 1911, p. 307. (A. R.)

THE SANTA GERTRUDIS CYANIDE MILL MEXICO.—"The new mill near the Santa Gertrudis mine will receive its principle supply of ore from the Santa Gertrudis Company through its San Guillermo shaft. The ore, after having been dumped by the skip into the 500 ton concrete bin situated at the San Guillermo shaft, is transported therefrom by means of a 30 in. Robins belt conveyer running 100 ft. per min. and divided into two sections, one horizontal, 500 ft. long, the other on a 12° incline, 175 ft. long. The second sections of this conveyer discharges on to a 30 in. flat distributing belt conveyer which in turn discharges into the crushing plant ore bins of 900 tons capacity. These ore bins feed into two No. 6 gyratory crushers, the discharge therefrom passing to a 40 in. by 14 ft. revolving screen. the oversize from this screen passing through two No. 4 gyratory crushers and joining the undersize on a 22 in. Robins conveyer, speed 206 ft. per min. This conveyer is equipped with a Merrick weightometer which automatically and accurately records the weight of the ore handled by the conveyer.

This 22 in. conveyer discharges over the first one of three Vezin samplers where a 10% cut is taken and discharged into a receiving sample bin of 15 tons capacity. The reject falls on a travelling flat belt conveyer for distribution into the battery bin of 2500 tons capacity. The sample from the first Vezin is fed from the 15 ton bins through a stack of 26 × 15 in. rolls; reducing it to about ½ in. and finer. Below the rolls a second Vezin sampler cuts out 10%, the sample tripping into a Challenge feeder which discharges into the third Vezin sampler, which takes a 20% cut. On a 600 ton basis the sample is now 1·2 tons and is dropped into a small bin and fed therefrom through a set of 9 × 9 in. sample rolls, quartered and passed through Jones rifiles, and thereafter is reduced in the usual manner. Rejects from the second and third Vezin samplers, and from the final quartering, etc., are returned to the battery bins by means of an 8 in. elevator.

Stamp and Tube Mill Crushing.—The battery consists of sixty 1550 lb. stamps arranged in units of 10 stamps each, each 20 driven by a 65 h.p. motor belted to one jack shaft The 10 stamp units have belt tighteners. One fourth inch screens will be used on the batteries. Crushing will be in solution, the solution feed coming from one main large steel launder, situated on top and running the length of the battery bins. This feed launder secures its supply from the two 40 ft. diameter by 20 ft. deep solution storage tanks at the top of the mill site.

The pulp from the batteries will pass to six primary Dorr classifiers, the sand discharging to six 5 × 16 ft. tube mills, the discharge from which will be returned to the primary classifiers by means of 10 × 54 in. Frenier pumps, one pump to each tube mill. The slime discharge from the primary classifiers flows to eight secondary Dorr classifiers, the sand discharge of which goes through four 5 × 20 ft. mills, discharge of which is returned to the secondary classifiers by means of 10 × 54 in. Frenier pumps. The slime overflow from the secondary classifiers constitutes the finished product from the mill and flows to a set of five primary 35 ft. diameter by 12 ft. deep Dorr thickening tanks, where the pulp is thickened from about 12 to 1, to 1 to 1. The thickened pulp from the primary Dorr tank goes to the first battery of eight 15 × 45 ft. Pachuca tanks operated continuously in series. The discharge from the last tank in the series is by means of an air lift into a launder where four parts of barren solution are added to wash, and the thin pulp sent to three secondary 35 × 12 ft. Dorr thickening tanks, where the pulp is again thickened and returned by gravity to a second battery of ten Pachuca tanks of the same (15 × 45 ft.) size operated continuously in series, the discharge from the last tank passing by gravity to two storage tanks 35 × 12 ft., equipped with mechanical agitators.

These storage tanks feed by gravity four 90 frame Merrill slime presses, under 35 lb. pressure; size of leaf, 4 × 6 ft., and width of frame, 3 in. The pulp slime from these presses, about four parts of water to one of slime, will enter two 35 ft. diameter by 15 ft. deep Dorr thickening tanks, where between 2½ to 3 parts of water will be recovered and returned by pump to the 40 ft. diameter by 20 ft. deep water supply tank at the top of the mill site. The solution to be precipitated will be clarified by first passing through two 40 ft. diameter by 10 ft. deep settling tanks, equipped with peripheral launder overflows, and then through two 40 × 10 ft. sand filters.

Zinc Dust Precipitation.—Precipitation will be effected by the Merrill zinc dust process, and although the solution will be of one grade it will be precipitated in two circuits, the first circuit passing through two Merrill zinc dust presses and precipitated to barren, and the second circuit passing through four similar presses but precipitated only down to about 10 to 15 grams of silver.

By this method the minimum of zinc dust will be used in the large or four-press circuit and a higher grade of precipitate obtained. The precipitate will be melted in a battery of eight oil fired, No. 400 crucible furnaces, arranged on the arc of a circle, and served by a radial jib crane. The pots or crucibles will be raised or lowered by means of an air cylinder situated on the post of the crane.

All pumping of solution and water is concentrated in two pump houses, one located below the Merrill filter plant, and another and larger house below the precipitation plant. The pumps are of the vertical triplex-plunger type are gear-driven by motors and are installed so that one pump is always in reserve for two circuits. No elevation of pulp is required throughout the plant, except in the case of the Frenier pump returns and of the air lift from the last tank of the first series of Pachucas.

Steel and concrete construction has been used practically throughout the plant, which is situated on a slope of about 17° and is served from top to bottom by an inclined tramway electrically operated.

Complete machine, blacksmith, carpenter and electrical shops with large storehouse have been provided and will be served by the inclined tramway. Side tracks at various levels lead from the tramway to the various floors of the mill and to the shops, so that repairs can be easily and quickly made. I am much indebted to Hugh Rose, general manager of the Santa Gertrudis properties, by whose courtesy I am enabled to send you the foregoing description of the work being carried on."—W. WESTON, *Engineering and Mining Journal*, July 1911, p. 109-10. (A.R.)

TIMBER FOUNDATIONS.—"The Abbontiakoon new stamp mill is supposed to be on order, and it is to be hoped that the engineers have given some thought to the material to be used in the foundations. These in most mills in West Africa are usually made of concrete as a base, but in the Wassau mill the mortar-blocks were of imported pitch pine, which speedily rotted, giving much trouble on the last two years running. In other cases, such as the 50 stamp mill of the Taquah mine, the main base was of concrete, on which were laid cast-iron anvil-blocks to receive the mortar-boxes; the longitudinal sills, strake sills, and the mill posts and ore bins were of wood. The mill started in July, 1907, but the ravages of this climate are so severe that the timbers are rapidly decaying, and indeed are now being replaced in connection with 30 stamps. The present manager, wise in his generation, is preparing to use steel girders and reinforced concrete in place of timber, so that when the full 50 stamps resume crushing there should be no further trouble about foundations during the life of the mill. Pine timbers last fairly well in this climate, if they can be kept either quite dry or else completely under water, but as neither of these conditions can be fulfilled in a stamp mill foundation, other material must be utilised in construction work. Even battery-posts fail quickly if made of wood, while the dampness in the ore is generally sufficient to ensure speedy destruction of a wooden bin. Local timber would be more enduring; Kaku, Kwisaba, and Odum furnish excellent wood, and might give much better results if the trees were 'rung' a year or so before felling, so that the timber could season in an erect position as it stands; but there is no doubt that steel and reinforced concrete will in the end prove most economical. It is found in ore-bins, for example, that pitch pine of 12 × 12 in. section usually rots completely in about 4 years. But is there any reason why the ore-bins both at the 'bank' and in the mill cannot be constructed completely of steel? On all the principal mines the practice, until recently, has been to timber the shafts with 8 by 8 in. pitch pine, a custom which is still followed in some places, but steel setts are now being generally used with great advantage."—*Mining Magazine*, July, 1911, p. 20. (A. R.)

EVOLUTION OF RAND CYANIDE PRACTICE.—"It is no exaggeration to say that the success of the Witwatersrand gold industry is a direct result of the introduction of the cyanide process. For the majority of the mines, the gold won by this process represents the difference between profit and loss; without it, the profitable working of the vast quantity of low-grade banket, now being mined on the Rand, would be impossible. It soon swept away all the costly methods of concentration by frue vanners and other shaking tables, and the subsequent roasting and chlorination of the pyritic concentrate, and thus cheapened the treatment, while it increased the percentage of extraction. On account of its important bearing on the gold mining industry, I will briefly trace a few of the steps in the development of the cyanide treatment.

At first the pulp from the stamp mills was run into retaining dams, from which the sand was afterward dug out and conveyed in Scotch carts or in mine cars, for treatment in the cyanide vats. On account of its slime content not more than 50% of the sand was suitable for treatment by percolation with cyanide solution; and of the gold contained in this, some 65% was extracted. In other words, only 30% of the gold, left in the pulp from the amalgamating tables, was recovered, the remainder being in the untreated slime and in the residue.

Jennings Introduces Hydraulic Classifiers.—The next step was the introduction by Hennen Jennings of the method of direct treatment, in which, by means of hydraulic classifiers, a considerable proportion of the slime was eliminated and a sand product obtained which could be run directly into leaching vats. A further improvement in the collection of the sand was the use of Butters and Mein's mechanical distributor, by means of which the sand is automatically fed into vats, filled to overflow with water. The clean sand is retained, while the slime overflows into an annular launder surrounding the vat.

The treatment of the slimes, which for some time after the introduction of the cyanide process had remained a waste product, was the problem which next engaged the attention of the Rand metallurgists. In 1894 experiments were made independently by Charles Butters, on accumulated slime at the Robinson mine, and by John R. Williams at the Crown Reef mine, with such success that thereafter slime treatment became general. Briefly, the process consisted in causing the slime overflowing from the sand collectors to settle, by the addition of lime, with the subsequent removal of the bulk of the water by decantation. The concentrated slime, so obtained, was then agitated with cyanide solution, which was ultimately drawn off by decantation.

Next Step Diaphragm Cones and Filters.—The separation of sand from slime by the old-fashioned inverted pyramidal hydraulic classifiers and the decantation method for the removal of water or cyanide solution from sand or slime, which have both played a most important rôle in the evolution of Rand cyanide practice, are now giving place to the use of diaphragm cones and vacuum filters. Thus, in the most modern plants, the separation of sand and slime in a mill product is effected by feeding the mill pulp into a cone-shaped collector or diaphragm cone, which differs from previous classifiers in being run nearly full of settled solids, instead of fluid pulp; the sand is drawn off as a thickened pulp from the bottom, while the slime flows over at the periphery. The sand, after passing through a secondary washing cone, which it leaves as an eminently leachable product with less than 1% of slime and about 30% of moisture, is freed from most of its remaining water on a Caldecott table which is a slowly rotating horizontal vacuum filter.

After almost completing a journey round the table, the sand, containing less than 12% moisture, is scraped off by a fixed incline plough and falls into a launder, where it mixes with a stream of cyanide solution and is raised by a sludge pump to a vat already filled with solution. It is fed in by a Butters' distributor, the overflow being returned to the launder to complete the circuit. By this continuous method of sand collection, the solution of the gold is begun within half an hour of the crushing of the ore. The treatment is either completed in the collecting vat, or the charge is transferred to a second vat for final treatment. The process is accelerated by vacuum pump, by means of which the moisture, remaining in the sand after draining, can be reduced to 9%.

As to the slime, the treatment has been much facilitated by the recent introduction of air-agitation tanks and vacuum filters, which enable the enriched cyanide solution to be rapidly drawn off from the slime residue and sent as a clear liquid to the precipitation boxes. The Butters' vacuum filter is now in successful operation at the Crown mines and will be installed at the Bantjes, the New Modderfontein, the Robinson and the Modder B. mines.

The precipitation of the gold was effected in the original MacArthur-Forrest process by zinc shavings, and this method is still preferred, I believe, for the rich solutions; but for weak solutions, such as are obtained in the treatment of slimes, the Merrill patent, in which the zinc dust is employed as a precipitant, is considered to be superior to the lead-zinc couple, and this method is now being used at the Village Deep and will be adopted by the Modder B., Brakpan and other mines."—FREDERICK H. HATCH, "James Forrest" Lecture, *Engineering and Mining Journal*, July 22, 1911, p. 165. (A. R.)

CHILIAN MILLS *versus* ROLLS.—"In *Metallurgical and Chemical Engineering* for June, Rudolf Gahl describes experiments undertaken at the concentration plant of the Detroit Copper Company at Morenci, Arizona, with a view to ascertaining the relative efficiencies of Chilian mills and high-speed rolls when used for the purpose of re-grinding jig tailing. He does not discuss the question from the point of view of power consumed or of keeping the two machines in good working order, but confines himself to the study of the comparative amount of slime produced and the percentage of copper recovered. In view of the facts that the Utah Copper Co. has a large installation of Chilian mills for re-grinding jig tailing and is about to apply a form of stage-crushing to them, and also that at the Miami copper mine, Chilian mills are used in one unit and high-speed rolls in the other, Mr. Gahl's paper is of special interest. His tests were made with a 5 ft. Chilian mill run at 136 r.p.m. and with rolls 36 in. diam., 15 in. long, and running at 140 r.p.m. The oversize from the rolls was as is the usual practice returned to them. The first test was on material that passed through round holes 7 mm. diam, but not through 5 mm. holes, and the discharge screens of both plants had 2½ mm. holes. The analysis of results is given as follows:

Mesh.	Percentage.	
	Rolls.	Chilian Mills.
+20	23·1	11·0
20 to 30	16·0	7·0
30 to 100	23·4	21·2
100 to 200	6·6	9·3
–200	30·9	51·5

It will be seen that the Chilian mill made a much finer product and a great deal more slime. In test No. 2 the material treated was between 5 and 2½ mm. with the same discharge screens as No. 1, and the general result was much the same. He then proceeded to ascertain whether the copper mineral was as efficiently freed from the gangue in the rolls as in the mills, by treating each size produced by the two machines, on Wilfley tables up to 209 mesh, and the slime by panning. It was found that more copper was lost in the Wilfley tailings from the roll product than from the Chilian mill product, and that though more slime was made in the mill, the total recovery was slightly greater in the mill than by the rolls. A third test was made with 5 to 2½ mm. material, with 1 mm. round hole screens on the Chilian mill and wire mesh screens with aperture 0·79 mm. in connection with the rolls. The result obtained was that the eventual recovery of copper was practically the same in both cases, the mill losing more in the slime and the rolls in the sand. If no further facts in connection with the general concentration problem are taken into consideration, the mill would be the best machine to use, because it is simpler and easier to keep in efficient working order. If however any subsequent regrinding of the coarse pulp from the mill or rolls is contemplated, the decision in favour of the mill is not necessarily final, because the coarse product of the rolls contains more copper than can be freed by further regrinding. The author subsequently made experiments with Chilian mills with a view to reducing the amount of slime produced, and endeavoured to reproduce the conditions obtaining with rolls, whereby the material when ground sufficiently fine is immediately removed. He fitted the mill with a 3 mm. slotted screen when treating 5 to 2½ mm. material, and passed the discharge to a wire screen with 0·79 mm. aperture, the oversize being returned to the mill. He does not give full details of the results in his paper, but his conclusion is that by this re-arrangement of the work of the Chilian mill the proportion of slime can be reduced to approximately the same as that produced by the rolls. By such an arrangement the advantages of the rolls from the point of view of slime production would be gained by the Chilian mill while retaining those already possessed by it. He wishes it to be clearly understood that his comparison of the two machines is limited to the sizes of material mentioned, for with coarser material the superiority of rolls is unquestioned."—*Mining Magazine*, July, 1911, p. 70. (A. R.)

ZINC DUST PRECIPITATION.—"One of the noteworthy features in the development of recent cyanide practice is the increasing use of zinc dust as a precipitant for the precious-metal solutions. Originally tried and pronounced a failure, it is interesting to note its recent general adoption throughout the mining countries of the world. Obviously, its abandonment during the earlier years of the industry must have been due to unsatisfactory results obtained through unskilful handling and failure to appreciate the fundamental principles governing the process. It may still be truthfully said of the zinc dust process that its operations requires a more intelligent surpervision than the old zinc-shavings method, but this point is more than off-set by the increased efficiency and decreased operating costs obtained.

Growth of Zinc-Dust Precipitation.—A large majority of the mills built during the last year in the United States, in Canada and in Mexico, have installed the new process, and almost without exception record material saving over the older method. In South Africa, where the dust process was only introduced within the last year, five installations are now either in actual operation or under construction. These installations were made by the Merrill Metallurgical Company, and as a direct result of a series of experiments conducted by the Mines Trials Committee of South Africa, to determine the relative merits of the Merrill process and the usual zinc shaving process, and in spite of the unsuccessful efforts previously made by local metallurgists to apply zinc dust.

In keeping with their usual progressive and broad-minded policy toward new ideas, Mexican Engineers and metallurgists have been quick to see the advantages of dust precipation as applied to silver cyanidation, and most of the recent mills in that country are equipped with this process. At one large plant in Pachuca where the new process was installed, the silver bullion recovered by this method will be in the neighbourhood of 20 tons per month.

It sometimes is claimed by those unfamiliar with the process that the consumption of zinc is higher and the grade of precipitate lower with zinc dust as a precipitant than when using zinc shavings. Data are available to show that in reality the exact opposite is true.

At the Homestake mine, in South Dakota, the amount of zinc consumed in precipating exceedingly low-grade waste solutions was 0·17 lb. per ton of solution with shavings, and 0·12 lb. with dust. The grade of precipitation was $2·35 per lb. with shavings and $4·03 with dust. At this mine, which is now cyaniding about 130,000 tons of ore per month, the total consumption of the zinc dust does not exceed 0·1 lb per ton of ore treated.

At the North Star mine, Grass Valley, Cal. where shavings precipitation was replaced by zinc dust, the consumption of zinc was reduced by one-half and a better grade of bullion obtained. At the Pittsburg-Silver Peak mill, Blair, Nev., cyaniding in the neighbourhood of 14,000 tons of ore per month, and where zinc dust has been used since the beginning of operations in 1908, the consumption of the zinc is 0·167 lb. per. ton of solution, with practically perfect precipitation. The raw product, when removed from the filter presses, contains 12 per cent zinc, and after acid treating this is reduced to 3½ per cent.

At the new mill of the La Blanca company, Pachuca, Hidalgo, Mex., the consumption of zinc is 0·93 lb. for each pound of pure silver produced, and the precipitate taken from the presses at clean-up contains 73 to 75 per cent. fine silver. At the Santa Gertrudis mill, also situated in Pachuca, which began crushing during the latter part of June of this year, most gratifying results are being obtained with the Merrill process. The consumption of the zinc is 1·2 per pound of bullion produced. The raw precipitate from the presses carries 87 per cent. of fine metal and the melted bullion assays 950 fine gold and silver. The average consumption of zinc in Mexican silver mills using shavings is close to 1·5 lb. zinc per pound of fine metal recovered, with a precipitate 60 to 70 per cent. fine, although a zinc consumption of 1 to 1·2 lb. and a precipitate of 75 to 80 per cent is obtained in a few instances, when precipitating exceedingly rich solutions.

Dust Process more Economical.—An interesting comparison of the costs of these two methods of precipitation is furnished by the large cyanide plants operating at El Oro, Mex. These mills are of approximately the same tonnage and the ore milled is similar in character. The costs for precipitating and melting at three of these mills are as follows: No. 1, with shavings precipitation, 30c., Mexican currency, per ton of ore treated; No. 2, with shavings precipitation, 20c.; No. 3, with dust, 16 cents.

As an illustration of the superiority of the dust method over shavings in certain individual cases, the installation of the Goldfield Consolidated Mines Company, at Goldfield, Nev., may be cited. At this plant it is necessary to precipitate solutions containing considerable copper, the amount present being sufficient to plate and polarize shavings after a short immersion in the solution. With dust, however, owing to the fresh zinc being continuously added, this polarization is prevented, and although the precipitate contains upwards of 40 per cent. copper, the precipitation is uniformly good.

Other Advantages of Zinc Dust.—It is to be borne in mind that one of the chief economies of zinc-dust precipitation, as compared to shavings, is that the first cost of zinc dust is from 20 to 25 per cent less than zinc sheets, depending on the fluctuations of the metal market. In cyaniding silver ores, particularly where the consumption of zinc is frequently as high as 3 lb. per ton of ore treated, the economy of the zinc dust is obvious.

Inasmuch as the reactions involved in precipitating metals from cyanide solutions are essentially those of reduction, it is desirable in carrying out any precipitation process that contact with the atmosphere be avoided as far as possible. In the new zinc-dust process this is accomplished by bringing the precipitant and the solutions into contact with the entire exclusion of air and maintaining this condition until the precipitate has been separated from the solution in a filter press. Re-solution of the precipitated precious metal by oxidation is thus prevented. A minimum consumption of chemicals is obtained by virtue of the fact that in this process the contact between the zinc dust and the cyanide solution is kept down to the shortest interval consistent with efficient precipitation.

Another point in favour of the zinc dust process that should appeal strongly to all cyanide managers, and particularly those operating custom plants, is the ability to clean up and convert into bullion at any time all of the precipitated metals in the mill. With the shavings process something like 30 per cent. of the clean-up is carried over in the boxes from month to month and an accurate check on mill recoveries is rendered quite impossible.

The chances of loss of valuable precipitate, due to theft or fire, are materially lessened, the product being collected and stored in a heavy cast-iron filter press, which is opened only at stated intervals for clean up, and then under trustworthy supervision. At the time of the fire at the Goldfield Consolidated Company's mill, the Merrill presses contained upwards of $500,000 worth of gold precipitate, but not a dollar of this was lost, even though the presses became so hot that the precipitates upon removal were found to be completely roasted, and the press cocks were spouting zinc flame during the fire."—C. W. MERRILL. *Engineering and Mining Journal*, August 25th, 1911, p. 407. (A. R).

ELECTRIC TIN SMELTING.—"This article gives an account of two tests carried out to ascertain if it were economically possible to smelt tin electrically. The first test using 20 lb. concentrates containing about 63% Sn, showed an extraction of about 85½%, and a second test gave 90% extraction, the tin obtained being over 99% pure. The slags analysed 0·5% Sn, which was recovered by converting into sodium stannate and electrolysing between iron electrodes. These experiments on the small scale were so successful that large scale trials were undertaken and the following weekly report for the week, July 28 August 4, will show the nature of results obtained.

Ore used 21,056 lb. at 57% ...	12,003	lb. metal
Scrap metal and charge refuse ...	2,324	,,
Total ...	14,327	,,

Tin produced	14,170 lb.
Net yield	98·75%
Total No. of K.W. hours ...	15,113
To be deducted for slag treatment	678
,, ,, ,, ,, preparing furnace, &c.	500
Total K.W. hours used for reduction	13.935
Units per ton	2,220

Electrode consumption 28 lb. per ton of metal.

These results show that a considerable saving can be effected by electric smelting especially if furnaces work in series, and ores are blended. The formation of hardhead is minimised and the furnace yields tin after first hour of operation, while the reverberatory is intermittent. Less labour is necessary, and the amount of carbon used for reduction is only 14% as against 20-25%. Also much less space is necessary for the furnace."—JOHN HARDEN, *Metallurgical and Chemical Engineering*, September, 1911, p. 453. (J. A. W.)

TITANIUM ORES AND THEIR USES.—As TiO_2 titanium is one of the most widely distributed elements of the earth's crust. Rutile, the commonest natural form, is rarely found in large deposits but ilmenite, an iron ore containing titanium occurs abundantly in many localities, and these two minerals are those utilised as sources of titanium.

Rutile is known to occur compact or massive in igneous sedimentary and metamorphic rocks varying in colour from yellow to reddish brown and black. Its hardness is 6·5, and specific gravity 4·2, Ilmenite is an iron black mineral of hardness 5-6 and specific gravity 4 5 to 5. Its composition is represented by the formula $FeTiO_3$, corresponding to 47·3% FeO and 52·7 TiO_2. The deposits which have been worked to any extent are as follows: (A) Europe—Kristiansund in Norway, average production 50 tons rutile per annum. (B) America: Virginia, Georgia, Maine and New York, in small quantities. (C) Australia: at Talunga, in South Australia, where desultory working has been carried on for many years.

The chief use of titanium ore at present is for the production of ferro-titanium, which is produced by reducing the ore with molten aluminium, or if the presence of carbon in the alloy is not objectionable, reduction with carbon in an electric furnace. Analyses of the products produced are given in the following table (*Mineral Industry*, 1900, 9, 720):—

	No. 1 per cent.	No. 2 per cent.	No. 3 per cent.	No. 4 per cent.	No. 5 per cent.
Titanium ...	10·28	25·17	35·64	53·92	75·84
Carbon ...	0·284	0·59	0·75	0·291	0·311
Phosphorus	0·09	0·06	0·06	0·059	0·042
Sulphur ..	0·01	0·011	—	—	—

The use of ferrotitanium has extended of late years very considerably as it is now used in the manufacture of steel rails. Its chemical action is to rid the steel of nitrogen, forming titanium nitrides, the effect of nitrogen being to increase the toughness and reduce the ductility. Exhaustive experiments with titanium alloys show that an alloy containing 10—15% Ti works best. It also prevents segregation of the sulphur, phosphorus and carbon. Comparative tests on the Baltimore and Ohio railway between titanium treated rails and ordinary Bessemer rails show that after five months wear on a heavy curve the former lost 1·45 lb. per yard, whereas the latter lost 4·18 lb. per yard. The following table shows the growing preference for this steel (*Metallurgical and Chemical Engineering*, 1911, p. 121):

	Gross Tons.	
	19 9	1910
Titanium steel	35,944	195,940
Nickel chrome steel	12,207	81
Nickel and electric steel ...	1,464	4,210
Manganese Steel	1,028	390
	50,724	200,621

Other uses to which titanium compounds have been put are the manufacture of titanium carbide electrodes for arc lamps, and also as filaments for glow lamps, the oxides for pigments and the oxalates as mordants for dyeing.—*Bulletin Imperial Institute*, Vol. ix., No. 2, p. 134. (J. A. W.)

IMPROVED METHOD OF SECURING BATTERY POSTS.—"An arrangement for securing battery posts to the concrete foundation block is described by F. W. Sewell, in the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 3. The arrangement is used at the mills of the Guanajuato Cons. M. & M. Company and the Zacatecas M. & M. Company.

A cast iron shoe is bolted to the concrete block, and the shoe, in turn, receives the battery post and holds it in place by suitable bolts and brackets. The shoe weighs 860 lb. and has a bearing surface on the concrete of 4 ft. 6 in. by 1 ft. It has a cavity cast in it 2 ft. long by 10 in. wide and 6 in. deep, into which the battery post fits.

The shoe is secured to the concrete block by two 2½ in. rods, each 5 ft. long, threaded at both ends. The threaded portions are of enlarged cross-section, so that the diameter of the base of the thread is not less than that of the remainder of the rod. The lower ends of the rods pass through broad, square, cast iron washers embedded in the concrete, and are secured thereto by square nuts. Hexagonal nuts or lock nuts are used on the top end to hold down the shoe.

The battery post is held securely in the shoe by means of 1½ in. bolts 2 ft. 7 in. long, connecting brackets on the side of the post with the cast iron shoe. At each end of the shoe are cast suitable projections which are bored vertically to receive the long bolts mentioned. The two side brackets, which weigh about 100 lb. each, are recessed into the 12 in. faces of the post and held in place by five ⅞ in. rods passing through the post from one bracket to the other. These rods are threaded at both ends, and lock nuts hold the brackets firmly in position. Passing vertically downward from each bracket to its respective end of the shoe are two rods, mentioned above, with square nuts to attach them to the shoe and hexagonal nuts to attach them to the bracket. The lowest part of the bracket is 1 ft. 2 in. above the top of the shoe.

In erecting the battery, the cavity in the shoe is half filled with oil, and the foot of the post is cut slightly larger than the cavity, the point being cut so that it will just enter the cavity. When the post is forced into place by means of tightening the connecting rods, aided by blows brom heavy hammers on top, the oil is forced into the end of the post. This insures protection from rotting due to water. The entire post is then oiled occasionally until no more oil is absorbed, when it is painted."—F. W, Sewell, *Metallurgical and Chemical Engineering*. Sept. 1911, p. 475. (J. A. W.)

MINING.

THINGS PROSPECTORS SHOULD KNOW.—"Prospectors, new to the business, who are searching for the ores of gold and silver, and for those of copper, lead, zinc, tin, etc., should know many simple, practical things—things that to the experienced prospector are so commonplace, he thinks them scarcely worth mentioning.

In certain districts the prospector confines his energies to seeking the ores of the metals commonly known to occur there. For instance, in Cripple Creek district, or at Goldfield, he looks for gold ores; at Bisbee, and at Ely, or near Cananea, copper ores are the object of the searcher; in Joplin district, and the surrounding country of Missouri, he looks for nothing but lead and zinc, while in the coast ranges of California, he scarcely expects to find anything else than cinnabar. So it is almost everywhere, and as a result of this many districts have been the scene of several periods of active prospecting. Leadville is a notable instance. Early in the 60's Oro City (now Leadville), was the centre of a prosperous gold district. In the later 70's Leadville became world-famous as a rich silver-lead camp, and now hundreds of men are prospecting old mines and new alike, for the ores of zinc, almost wholly overlooked in the earlier days. Old prospectors are prone to search for mineral veins and deposits of a type for which they have been successful, or with which they are most familiar, as they more quickly recognise the "indications" accompanying these than those in a region in which the formation is strange to them.

There is an old and oft-repeated saying that "gold is where you find it," and this is true, and something to be remembered, for the prospector who goes out into the hills with a pre-conceived notion of how the gold for which he is looking should be found, is likely "to overlook a bet."

The greater number of prospectors these days are probably in search of gold or copper, though they will not disdain a vein of silver, lead or zinc ore, should they see one and recognize it as such. It is important, then, to know in what kind of rocks to look for gold and copper ores. Gold, silver and copper, and also lead and zinc, occur in almost every rock known to geological science, excepting, possibly, the glassy volcanic rocks—obsidian, pearlite, pitchstone, vitrophyre, pumice, and rocks of that class. Therefore it behoves the prospector to examine every rock formation, whether sedimentary, metamorphic, intrusive, or volcanic, when searching for deposits of valuable ores.

Veins dip at all conceivable angles, from vertical to horizontal, and can usually be distinguished by their contrast in appearance with the rocks in which they occur, differing usually in colour, structure and general look, being nearly always either softer or harder than the enclosing rocks, and also, in most instances, of a different colour. There are zones of rock, confined to no particular kind, that do not differ greatly from the surrounding rock in appearance, but which, nevertheless, contain ores in payable amount. These are usually due to impregnation, the valuable minerals having been deposited in them by solutions which permeated such zones, generally, though not always, from below.

All gold-bearing outcrops do not "prospect in the pan." That is, the gold is not always sufficiently coarse to be successfully concentrated by panning, so that it may be seen. Assaying is, therefore, not only often advisable, but absolutely necessary. Many deposits of valuable minerals are extremely irregular in form. Often they are without defined walls, or have but one wall. The causes of these phenomena need not be discussed here, for they are not important to the prospector, whose chief concern is in finding a mineral deposit of value, no matter what its shape or size. Some outcrops are black, being heavily mineralized with black oxide of manganese and iron. Such usually contain gold, silver, lead, zinc and copper. A notable instance is the famous Broken Hill, in New South Wales.

A large white outcrop of quartz is seldom of sufficient value in either gold or silver to be profitable, though there are exceptions to this. However, small veins of white or bluish-white quartz not infrequently contain sufficient gold to make them payable. A zone of schist with lenses and stringers of quartz, lying on either the foot wall or the hanging-wall side of a large mass of quartz, is often found rich enough to pay.

All pay shoots do not reach the surface. The uppermost edges of many pay shoots do not come within many hundreds of feet of the surface. All experienced prospectors are familiar with the *chapeau de fer*, of the French miner, or gossan of the Cornish miner, called "iron blow-out," in the United States. The typical gossan is the result of the oxidation of large masses of iron and copper sulphides, often of both, but there are also iron outcrops, due to an entirely different cause, namely the oxidation of ferro-dolomite, or carbonate of iron, of which former some of the ankerite masses of the Mother Lode of California are characteristic examples. These iron outcrops may, or may not be gold-bearing, depending on whether the ankerite contained gold or not before its oxidation. The gossan derived from the oxidation of pyrites usually contains both gold and silver, derived from the normal sulphide ore which has been oxidized. The great mass of iron ore which at one time formed the outcrop of the Mountain copper mine in Shasta county, California, was formerly worked for silver. A similar outcrop of the Mount Morgan mine in Queensland, Australia, was rich in gold. Both mines are now extensively worked for copper in depth, and the sulphide ores of both mines still carry gold and silver.

Not every iron-stained outcrop, however, is entitled to be called a gossan, as in many places considerable areas, which at a distance appear to be masses of red or brown iron ore, on close inspection prove to be merely a thin coating of iron oxide, due to the oxidation of pyrite which is disseminated in minute crystals through the country rock. This is particularly characteristic of some greenstone schist. Such occurrences are seldom of economic importance. In some instances the red or brown colour may be due to the oxidation of the ferro-magnesian silicates, hornblende, or pyroxene, and the oxidation may extend to considerable depth. Most of these are of no value, but occasionally the colour is due in part to the oxidation of auriferous pyrite, and then the rock will be gold bearing. As a consequence the prospector must be alert and test the ground. If the schist be quartzose, the chances are fairly good for gold, though the amount present may be too small to make the deposit valuable. However, silicious schists are often gold-bearing in payable amount. The Homestake outcrop was of this character, the entire mass, even the veins and masses of quartz included in the schist being dark red in colour.

Often an iron outcrop is found to consist largely of a cellular or spongy mass of quartz, deeply stained by iron oxides, which is the result of the oxidation of the sulphides it originally contained. Copper is sometimes almost wholly absent from such veins, but wherever found, as a thin scale, patches of beautiful iridescent colours (rainbow colours), either on the iron or on the quartz, copper will, I think, be found in the vein in depth. At all events, I have never known this "sign" to fail, in scores of cases scattered through many States. The iridescent mineral may give no trace of copper by test, but seems to be an indication of its former presence. If examined with a strong glass these patches of iridescent colour appear to present a surface somewhat like that of an orange, made up of innumerable little rounded surfaces, closely grouped together. This mineral, which occurs as the thinnest kind of a scale, and is often seen on the surface of pieces of quartz from veins, must not be confused with a somewhat similar iridescence characteristic of what is known as peacock copper ore (erubescite or bornite). Common yellow copper sulphide (chalcopyrite), also often presents a similar display of colours when first taken from the mine, but in such instances it is a mere tarnish, for if the piece of ore be broken it will be found that the iridescence does not extend through the piece, but is confined to natural fracture faces.

Not infrequently dikes are found to be gold-bearing at their outcrop, where they are comparatively soft from being more or less oxidized, but the gold is generally found to diminish with depth, and may disappear altogether when a normal condition of the dike is reached below.

Pieces of ore detached from a vein by natural causes are commonly called float, and such pieces of float by gravity always work down hill, never upward. When trying to find a vein by following the float, keep going uphill as long as a single fragment can be found. In or near the bottom of a canyon or gulch, an occasional exception to this rule may be found, where a piece of float has rolled down the hill on one side of the canyon, and bounded a few feet up the slope on the other side, and lodged there. In such case the fragment is most likely to weigh several pounds, and other similar boulders of float are more than likely to be found on both sides of the gulch, but the prospector will not be long in deciding from which side of the gulch the ore originally came. Where there is much slide rock, or soil on the hill slope, it may require considerable work to find a vein from which the float came. Near the base of the hill it may be quite abundant, but on following it up, the pieces may become more and more scarce, owing to their being buried in the debris on the hillside. Possibly the apex of the vein may be covered by several feet of soil and rocks. In such event the float should be followed up the hillside on the surface as far as a single fragment of it can be found. Above this point the search must be continued by digging, until the apex is discovered.

This leads to the question. 'What is an apex?' It is a question that has been much discussed. In a certain mining law-suit, a miner who had been summoned as a witness, testified, when asked what miners and prospectors generally understood by the term 'apex,' that in his opinion, 'not more than one miner in twenty would know an apex if he should see one lying dead in the road.' However, divested of technicalities, the apex of a vein is its uppermost portion—that part outcropping at the surface, or if covered by a later deposit, as of lava, soil, slide-rock, or any other later formation of any kind, then that portion of the vein which originally outcropped at the surface. The apex of certain deposits, like some of those occurring in limestone, may not reach the surface unless exposed by erosion. The apex of a vein, then, is its highest part, whether exposed or not.

The outcrop of a vein does not always run in the direction of its true strike. This divergence of direction is greater in a vein of low inclination than in one that stands steeply, particularly where the country is rough. In a comparatively level region there will not be so much difference between the direction of outcrop and the true strike of the veins. The strike of a vein may be defined as a horizontal line drawn at right angles to the dip, and the dip as the direction in which a vein descends into the earth. Neither the strike nor the dip of mineral veins are, as a rule, perfectly straight for long distances, for they usually have a wavelike course and dip. Veins which are perfectly straight in strike and dip are seldom very profitable—there may be exceptions, but I am not acquainted with them.

In locating claims the prospector is expected to lay the centre line of the claim along the outcrop of the vein, as nearly as possible, regardless of the true strike of the vein, otherwise the outcrop of the vein may pass through a side line of the claim, and thus cause the miner to lose a portion of his vein, through the operation of the law of what is known as the 'extra lateral right.'

In prospecting, the element of what is called luck, is a most important one. In proof of this assertion it can be stated that some of the most valuable mineral deposits ever discovered have been found by men who knew nothing of geology, and in fact were scarcely able to distinguish one kind of rock from another, and who were unable to even recognise ore when they found it. One of the most noted mines of California was discovered by prospectors who were working in the gulch below the outcrop of the vein, and who for 2 weeks or more sat daily at noon on a vein literally filled with gold and did not know it until one of them chanced, by accident to see the gold. The men who discovered the Comstock lode, did not realise that they had found a wonderful mine the day they went to work on the outcrop of the Ophir mine. So abundant was the gold in their rocker that they were very sceptical as to whether it really was gold. A prospector located a large mass of red iron-stained ore thinking it was quicksilver. It contained no trace of cinnabar, but was rich in chloride of silver. Stratton, the man who discovered the famous Independence and Washington mines in the Cripple Creek district, was an unusally good prospector. He was shown the outcrop, a dike-like mass of dark iron-stained granite. The loose dirt below the outcrop contained gold which could be obtained by panning. A few pieces of white quartz were scattered about. These he picked up and had them assayed. The result was $2 or $3 to the ton. He was not satisfied, and thought the loose dirt too rich for such poor quartz. It then occured to him that it might be the iron-stained granite dike that carried the gold, and he was right, for his first assays ran over $200 to the ton in gold; and he spent the Fourth of July in making the locations which brought him fame and an immense fortune."—W. H. STORMS, *Mining and Engineering World*, Aug. 1911, p. 328. (A. R.)

Large Units in Hoisting.—"The introduction of electric hoists on the Rand has practically decided the long debated question of hoisting from great depths, in favour of stage winding, as against a single lift. By the adoption of a large unit of hauling and a suitable arrangement of loading pockets, the loss, due to the repeated handling of the ore, can be reduced to a minimum.

In the new seven-compartment shaft of the Crown mines the ore is to be raised in eight-ton skips. Even greater units have been adopted at Kimberley, as, for instance, at the Wesselton, Bultfontein, and Dutoitspan mines, where 11-ton skips are in use."—*Engineering and Mining Journal*, August 1911, p. 320. (A. R.)

The World's Deepest Mine.—The deepest gold mine in the world was until lately that of the New Chum Railway Co., at Bendigo, Victoria, Australia. The New Chum shaft is 4,120 ft. deep. It has lately Been surpassed in depth by the shaft of the Victoria Con. Co., also at Bendigo, sunk to a depth of 4,600 ft. More recently the record depth for gold mines has passed from the hands of the Australians, and is now held by the British San Juan del Rey Gold Mining Co., operating in Brazil. At the Morro Velho mines of this company, in the state of Minas Geraes, the 16th level is 4.226 ft. deep; the present lowest level, the 17th, is 4,926 ft. in vertical distance below the collar of the main shaft, and 4,602 ft. below the adit. The Morro Velho ore body dips roughly 45°, so that the workings extend downward along the reef a distance of nearly 7,000 ft. The deepest American gold mine is believed to be the Kennedy shaft at Jackson, Cal., the workings of which are approximately 3,500 ft. below the surface.

As previously stated, the world's deepest mines are in the Lake Superior copper belt. No. 5 shaft, at the Tamarack mine of the Calumet & Hecla Mining Co., was down to a depth last winter of 5,273·5 ft. below the shaft collar. It was planned to extend this depth some 10 ft. farther, to cut out a sump. This extension, which is doubtless completed, makes the Tamarack No. 5 shaft the world's only single shaft exceeding a mile in depth. No. 3 shaft of the Tamarack mine represents probably the world's deepest actual workings. No. 3 shaft is down 5,222 ft., or 58 ft. short of a mile; but some crosscuts and slopes bring the bottom of the workings down to a total depth of 5,368 ft. below the collar of No. 3 Tamarack shaft. As the copper-bearing Calumet conglomerate dips about 38°, the lowest Tamarack No. 3 workings are nearly 9,000 ft. distant from the outcrop of the conglomerate. There must also be mentioned in connection with the world's record shafts of the Lake Superior copper country, the No. 4 Calumet shaft of the Calumet & Hecla Mining Co. This is the world's longest inclined shaft. It is now down to the 81st level, or approximately 8,100 ft. along the slope of the conglomerate, at an incline of about 38°."—*Mining and Engineering World*, Aug. 12, 1911, p. 277. (A. R.)

Earthquakes in Mines.—"While I was interested in the working of the Harvey Hill mine in this province (Quebec) there was an earthquake, violent enough to crack thick stone walls, in the valley of the St. Lawrence. My recollection is that the movement was very perceptible at the surface of the mine but not noticeable enough underground to excite observation. In Chili I made inquiry at every mine I visited as to the damage done underground by earthquakes, and I could not learn that in a single instance even a stick of timber had been dislodged or a shaft put out of plumb.

Subsequently in May, 1887, northern Sonora and southern Arizona were violently shaken by a series of earthquake shocks. The focus of the disturbance was near Bavispe, a small town in the valley of the Bavispe river on the west slope of the Sierra Madre. It was completely wrecked, and as the first shock occurred in the afternoon, when the people were taking their siesta, the mortality was high.

Bisbee and the Copper Queen mine and works were about 100 miles to the northwest of Bavispe. There the first shock occurred shortly after it had destroyed the latter, but no observations were taken at either place to determine the velocity of the earth wave. In the Bisbee the shock was violent enough to dislodge loose rock from the waste heaps, to crack a slag dump resting on alluvial soil, to injure a recently built foundation of a Corliss engine and to rupture adobe walls. But no house was thrown down. In the Copper Queen mine the shock was felt and great alarm created, but no rocks or timbers were dislodged. The shocks continued for over a month with increasing intensity. Several days after the first shock I was underground with a foreman of the mine exploring one of the narrow fissures in the limestone rocks, which there generally indicate the proximity of the ore body, when a shock of very considerable violence occurred. The sensation was as though the walls of the fissure were approaching and would completely crush us. As a matter of fact, however, not a particle of rock was dislodged, and after the event one was doubtful as to the part one's imagination had played in the apparent phenomenon.

In the San Pedro valley to the west of Bisbee, where the soil was deep and saturated with water, the motion was sufficiently violent to throw down buildings and to produce a distinct sense of nausea. Dr. Goodfellow, a competent witness, was in Tombstone, 30 miles northwest of Bisbee. He reported that 'men working at a depth did not know of it at all till they came up out of the mine.' The earthquake was felt distinctly down the San Pedro valley to its junction with the Gila, a distance of about 100 miles to the northwest of Bisbee.

Every destructive earthquake has wrought most damage on alluvial soil, especially if it be moist. That was notably the case in the Charleston and San Francisco earthquakes. If one adopt the commonly accepted figures for the rapidity with which earth-waves are propagated, namely, 5,000 to 6,000 feet per second through solid rock, 1,000 to 800 feet per second in gravel and sand, and very much less when the soil is saturated by water, one can easily see how the passage from one class of material, or from dry material through moist material, must aggravate the effects of the disturbance. Moreover, the velocity as well as the amplitude of the waves must be influenced largely by the character of the rock and by the obstruction offered by rocks whose strike is at a more or less acute angle to the direction of the shocks." James Douglas, *Canadian Mining Institute.—Mining Institute*, Aug. 10, 1911, p. 128. (A. R.)

The Bleloch Theory.—"At the annual meeting of the Eastern Gold Mines, E. Brayshaw, in referring to the properties held by the companies said that if the Bleloch theory proved right the Main Reef series would occur much nearer the surface on their properties. As a matter of fact the company has proved the Bleloch theory to be absurd, for already by boring on Finaalspan the Kimberley reef series was intersected from 1,111 ft. to 1,700 ft. and the Main Reef series at a depth of 3,800 ft. from the surface. In the No. 2 bore hole the Kimberley Reef series was also passed through the bore-hole taken to a depth of 4,500 ft. so that the company has already proved by actual boring that the Main Reef series cannot possibly occur near the surface."—*Mining and Engineering World*, Aug. 19, 1911, p. 358. (A. R.)

TIN DISCOVERY IN RHODESIA.—"There is a good deal of interest in Rhodesia and in British mining circles at present over the recent discovery of pegmatite veins containing up to 5% metallic tin, spread over an area of some 25 square miles in the Enterprise district, 25 miles east of Salisbury, Rhodesia. The pegmatite lodes are, some of them, as much as 50 ft. wide, and yield excellent crystals of cassiterite.

The lodes in the Enterprise district bear, we understand, a close resemblance to those in America, Australia and Cornwall, and are not at all like the ore at Zaaiplaats, in the Transvaal, which is generally recognised to be a freak.

Though cassiterite has been prospected for in Rhodesia for some years no tangible results have as yet been achieved; but the discovery in the Enterprise district is highly promising, and may not improbably enable Rhodesia to add tin to the list of the base metals she produces. When pegged a few months ago a number of claims were offered to, and declined by, one of the leading mining houses. At the same time the main features of the ground located were made known, and further investigations led to discoveries, the announcement of which caused a sensation throughout Rhodesia scarcely ever before equalled in the history of the country.

The deposit is situated about 5 miles north of the Planet and Arcturus gold mines and within 25 miles of Salisbury, in a district already renowned for its somewhat peculiar auriferous schist ore bodies. The discoverer has secured an area of some 25 square miles, and numerous other claims extending for many miles beyond this area have been pegged by other persons. The discoveries comprise a number of lodes in tin ore bodies, extending from 20 ft. to 30 ft. in width, and although only 'scratching' on the surface has as yet been attempted, the results may be said to be remarkable. Specimens containing fully 20% of tin have been exhibited in Salisbury and Bulawayo, and assays of a number of samples taken from the outcrop at various points give the average of 5·7%. The matrix is a mixture of quartz and purple lithia mica, forming a somewhat peculiar variety of rock known as 'greisen,' which is a well-known tin carrier in many parts of the world, and, as is usual, the occurrence is near a large granite mass, stretching for many miles, with a number of small granitic intrusions in the neighbourhood. The cassiterite appears to be disseminated through the greisen in minute crystals invisible to the naked eye, though some of the samples contain crystals over half an inch in length. The walls are epidiorite and actinolite schist, and the greisen body containing the cassiterite appears to be heading for the granite contact some distance off, with every indication of permanence. It is, of course, impossible as yet to estimate the importance of the discovery. No very precise information has been given regarding the character of the ore body, and no one at present seems to know whether it is an extensive reef or a chute in a reef, or even a pipe like that of the Zaaiplaats mine in the Transvaal. Further particulars will no doubt soon be forthcoming, but in the meantime the most that can be said is that the indications are highly favourable, and that, should the reported values be confirmed on further development, the discovery will be one of the greatest importance to the country.

Tin mining in South Africa is only a comparatively modern industry, which has made a promising start in the Transvaal and Cape Colony. But the indications in the Enterprise district, whatever they may be in other parts of South Africa, are undoubtedly similar to, if not identical with, those prevailing in the great producing centres of the Mallay region, Bolivia, and Australia—that is to say, the deposits occur at granite contacts, and the lodes, almost without exception, lie close to large granite masses or alongside their offshoots.

The Enterprise district is not the only locality in Rhodesia where indications of tin have been met with. In the Lomagundi district some years ago a discovery was made, and the Mashonaland Agency spent upwards of £7,000 in opening up the ground. But ultimately work was stopped and the claims abandoned."—*African World—Mining World*, Aug. 5, 1911, p. 238. (A. R.)

THE DISTRIBUTION OF GOLD.—"*Economic Geology* for May, 1911, contains a valuable article by F. C. Lincoln entitled 'Certain Natural Associations of Gold.' The paper is divided into two parts, the first dealing with the recorded occurrences of primary gold in rocks and waters, and the second describing the association of gold with various minerals. The first part collects together the result of the author's and other investigators' enquiries into the presence of primary gold in rocks as distinct from lodes and veins. He tabulates the results, dividing the rocks into igneous, metamorphic, and sedimentary, and gives the figures for the gold contained in various kinds of water. He includes also records of the presence of gold in minerals of organic origin, such as coal, in living organisms. In interpreting these records he prefaces the warning that the method of assaying rocks for minute traces of gold is of importance, because the reagent employed is as likely to contain gold and silver as the rocks, a circumstance pointed out in this magazine recently; also that many investigators have taken their samples too near a vein, not fully appreciating the distance to which secondary impregnations may extend. In the records quoted by the author the contents vary widely, the figures for igneous rocks ranging from nothing to 5 grm. per metric ton. After careful consideration of the various cases he takes the average gold content of igneous rocks at 62 mgm. per metric ton, that is, one grain per ton of 2,000 lb., and the silver content at 3,741 mgm. per metric ten or 2·4 dwt. per ton of 2,000 lb. He also concludes that the gold favours neither acid nor basic rocks, but that the silver is more plentiful in basic. As regards sedimentary rocks his recorded figures point to an average content of 15 mgm. or 1 cent per ton and show that the coarser the texture the greater the content. The sea-water is estimated to contain 28 mgm. Assuming that the contents of sea-water have been derived from the destruction of igneous rocks, Mr. Lincoln works backward and deduces from the above figures for sedimentary rocks and sea-water that the original content of the igneous rocks must have been 55 mgm. of gold per ton, a figure not far off his average of recorded investigations. This result agrees substantially with De Launay's generalizations. The author proceeds to show that no magmatic segregation of primary gold in igneous rock has ever taken place, and that no placers of commercial value have ever been produced from the primary contents of eroded igneous rocks. These and other occurrences of gold are discussed in detail and the facts are marshalled in such a way as to lead to a theory of the origin of auriferous lodes, thus: (1) gold has not been found in volcanic emanations and its presence in plutonic emanations has only been inferred; (2) gold has been found in the circulating waters of the earth's crust; (3) there are a great number of valuable deposits of primary gold in geologically young conglomerates, several in geologically old conglomerates, and two known cases in metamorphic rocks; (4) there are none in igneous rocks. Presented in this order the facts suggest that the gold of many lodes may have been derived by circulating waters from buried placers, and Mr. Lincoln enumerates the varying sources in order of importance as follows: (1) depressed beds of open-textured sedimentaries, (2) depressed beds of open-textured tufaceous rocks, (3) fractured older lodes, (4) fractured compact igneous rocks, (5) fractured compact sedimentary rocks. As a specific case he considers that the Tertiary lodes owe their gold mainly to the leaching of pre-Cambrian placers by magmatic or meteoric waters put in circulation by the intrusion of the andesite. The above is a mere outline of an interesting paper that should be studied in detail."—*Mining Magazine*, July, 1911. p. 71. (A. R.)

MISCELLANEOUS.

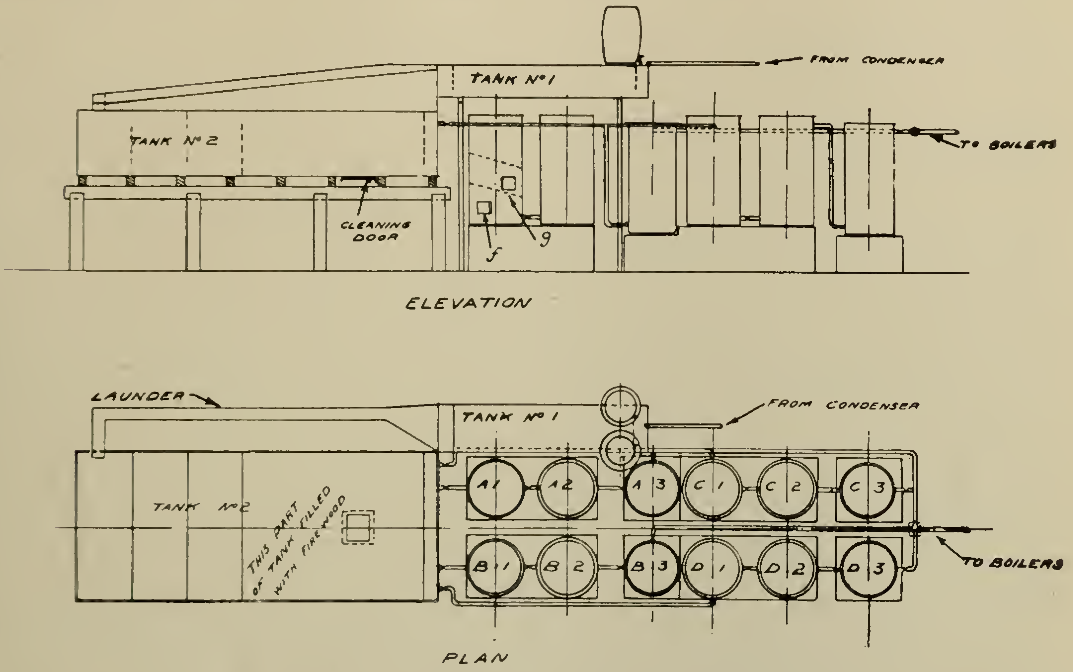
Price of Glycerine.—"Several notices have appeared in the technical Press recently announcing that the price of dynamite and allied explosives will advance in the near future in a way that will be highly inconvenient to those engaged in mining operations, owing to the increased price of glycerine which is the base of nitro-glycerine manufacture. *The Times* of June 13 contains a timely article on the commercial position of glycerine and explains the reasons for the enhanced price. Glycerine is a by-product in soap manufacture; when animal and vegetable oils are treated with alkali, soap and glycerine are formed. Except for its use in pharmaceutical preparations, glycerine was of small commercial value until nitro-glycerine was invented. and nowadays owing to the restricted circle of explosive makers little is heard of it as an ordinary market commodity. The 'spent lye' left behind in the manufacture of the best soaps contains about 9% of glycerine. The relative amount of glycerine and soap produced by the reaction varies widely according to the nature of the raw material and the kind of soap produced. Most of the soap makers boil the lye down to produce a crude glycerine containing 80%, though some of the producers of poor quality soap throw it away. Only a few of the large houses make a refined product; the others either sell the crude to the bigger firms, to the limited number of explosive manufacturers who undertake refining, Nobels being the chief, or to merchants who buy for America. The British Government cordite works do not refine, but buy mostly from Nobels. At the beginning of 1907, crude glycerine was worth £28 10s. per ton, and refined £50 per ton. By the end of the year crude had advanced to £68 and refined to £105 per ton. The cause is due chiefly to the advance in the price of oils and fats. The soap maker has found it impossible, or at least injudicious, to recoup himself by increasing the price of soap, owing to its being a national commodity necessary to the poorest classes, so he has had to turn his attention to the other outlet of his products. It is interesting to note that the chief reason for the advance in the price of fats and oils is their increasing use in the manufacture of margarine. Some of the largest soap makers have found it necessary to protect themselves by becoming their own producers of vegetable oils. Taking all these facts into consideration it is obvious that the price of explosives will, unfortunately for the mines, continue to advance."—*Mining Magazine*, July, 1911, p. 67. (A. R.)

Electrical Resistance of Pure Metals at Very Low Temperatures.—"For several years Prof. H. Kamerlingh Onnes, of Holland, has carried out exceedingly interesting experiments on the enormous decrease of electrical resistivity which pure metals undergo at very low temperatures. He has now perfected his experimental facilities for the use of liquid helium (which by the way was found to be an excellent insulator) and the results of his determinations of the resistivity of pure gold and pure mercury at liquid helium temperatures are given in a recent issue of the *Proceedings of the Koninklijke Akademie van Wetenschappen, Amsterdam*, of which a translation in abstract appears in the *London Electrician* of August 4. The author's recent experiments with gold have greatly strengthened his former conclusion that the resistance of pure gold vanishes at helium temperatures. Even more interesting are the experiments with mercury. The measurements were made with the differential galvanometer by the method of overlapping shunts (Kohlrausch) and also by the method of the measurement of current strength and of potential difference. The value of the mercury resistance used was 172·7 ohms in the liquid condition at 0° C; extrapolation from the melting point to 0° C by means of the temperature coefficient of solid mercury gives a resistance corresponding to this of 39·7 ohms in the solid state. At 43° K (43 degrees on the krypton thermometer) this had sunk to 0·084 ohms, that is, 0·0021 times the resistance which the solid mercury would have at 0° C. At 3° K. the resistance was found to have fallen below 3×10^{-6} ohm, that is, to one-ten-millionth of the value which it would have at 0° C. As the temperature sank further to 1·5° K, this value remained the upper limit of the resistance. The next step was to look for the point at which the resistance first becomes measureable as the temperature is raised. The temperature of this point was found to be slightly more than 4·2° K, at which the resistance was found to be 230 microhms or one-hundred-thousandth of the resistance (solid) at 0° C."—H. Kamerlingh Onnes, *Metallurgical and Chemical Engineering*, September, 1911, p. 469. (J. A. W.)

The Elimination of Oil from Return Feed-Water.—"It is well known that any considerable amount of oil allowed to enter boilers with return feed-water is dangerous and destructive to the tubes and plates, but it is not perhaps so generally realised that oil is destructive even if admitted in very small quantities, and, by accumulating on the heating surfaces, it greatly reduces the efficiency of evaporation. It has been determined that the transmission of heat through a boiler plate will be retarded more by a film of grease 0·001 in. thick than by $\frac{1}{4}$ in. of scale. The oil also assists other deposits from the water to adhere to the surfaces, and necessitates more frequent cleaning. In most cases the cost of removing the oil before the water enters the boiler will be more than saved by the reduction in boiler-cleaning expense; but even if this is not so, the treatment is justified by the better evaporation obtained and the protection it gives the boiler.

Most of the free oil present in exhaust steam can be removed by one of the types of mechanical separators on the market: or where jet condensers are in use, the oil will float on the water in the tail pit or tanks and may be skimmed off, but that part of the oil that becomes emulsified and gives the water a milky appearance is more difficult to deal with, and cannot be separated even by passing through filter paper. It is necessary to treat the water chemically, or otherwise to cause the oil to coagulate before it can be separated by filtration.

At the plant of the Kalgoorlie Electric Power and Lighting Corporation, Limited, several methods and arrangements of filters were tried, but they were found to be either ineffective or unworkable, on account of difficulties in dealing with the filters.



Apparatus for Eliminating Oil from Return Feed-Water.

An electrical method, which consisted of passing the oily water through a tank containing iron plates as electrodes, and submitting it to an electrical current equal to about 1·1 kw. per 1,000 gal. was very effective in causing the oil to coagulate, but it had to be abandoned owing to the corrosion set up in the feed pipes and in parts of the boilers. This appeared to be due to electrolysis, although every care was taken to insulate the electrical portion from pipes, etc.

The treatment now in use has proved to be very effective and cheap in working, and although it is not considered in any way novel, a description of it may be of interest to others who have similar conditions to contend with. The method consists of treating the oily water with alum and common soda in the proportions of approximately 0·35 lb. of alum and 0·30 lb. of soda per 1,000 gal. of water, for water containing by analysis 3·5 gr. of oil per gal. of water. The quantities of chemicals would, of course, be varied to suit the amount of oil contained in the water. The apparatus employed in the treatment should have sufficient capacity to allow ample time for the chemicals to act on the oil. In the plant referred to about 50,000 gal. per 24 hours are treated. The capacity of the tanks and filters is such that five hours are required for the water to pass through. A longer time would probably be an advantage, and would reduce the consumption of alum and soda. As the precipitate formed is very light and easily broken up, it is desirable that the plant should be so arranged as to reduce the agitation of the water, as much as possible, in its passage through the tanks.

The alum and soda are each dissolved in the barrels *A* and *S* respectively, and the solutions are allowed to drip from the barrels and mix with the incoming greasy water in Tank 1. The water passes under and over baffles in Tanks 1 and 2, and through a quantity of rough filtering material or firewood in part of Tank 2. Each of the drums in the four sets of filters, *A*, *B*, *C* and *D*, contains a bed of filtering material between the perforated trays indicated by dotted lines on one of the drums. This filtering material may be coke, charcoal, woodwool, shavings or any fairly open or porous bed. Sand is most effective, but clogs up too quickly to be used in the plant mentioned. The tank 2 has a cleaning door in the bottom, through which, after the water is drained off, the accumulation can be discharged. This tank requires cleaning monthly. The filter-drums are provided with drains and cleanings doors, through which the filter-beds can be quickly removed and renewed. The first drum in each set is cleaned twice weekly, the second drum weekly, and the third fortnightly. On entering the apparatus the water contains 3,500 gr., and on leaving 850 gr. of oil per 1,000 gal. The cost of chemicals amounts to 1·15d per 1,000 gal. The treatment has been in constant use some two years with satisfactory results, and the boilers are in excellent condition, requiring considerably less frequent cleaning than they did formerly."—E. C. CROCKER.—*Monthly Journal of Chamber of Mines of Western Australia*, July, 1911, p. 176. (H.A.W.)

DECIMAL TIME FOR MINES.—"Mariners have their system of 'bells' for keeping the time of day, the geologist has his 'eras' and 'formations,' while the ordinary mortal since time immemorial has had to use the cumbersome system of seconds, minutes, hours, and days. In nearly every country in Europe, and Latin America, the decimal system has been applied to weights, measures and coinage. In the United States such application is permissible by law and has been adopted in technical work and in large industrial organizations. The American laughs not one whit more heartily at the Englishman's crude pounds-shillings-pence system than does the European or Latin American at our equally cumbersome inches-feet-yards-mile system. Yet in measuring time, the decimal system which has proved its advantages in all other directions, has not as yet been seriously considered.

It has remained for a Frenchman to evolve a decimal time system and a suitable watch to divide the solar day of 24 hours into decimal divisions. The decimal watch, known as a *cémeter*, has been particularly designed for the mining industry, though the same watch may serve for any time-keeping purpose. Instead of the ordinary watch's division of the day into two shifts, each separately recorded on the usual 12-hour dial, the new watch dial is divided into 100 spaces, and a hand revolves about the dial once in a whole day. No. 50 represents, therefore, midday or noon. Each of these hundredth parts is called a *cé* (pronounced 'say') which comes from the same root as cent in cents, per cent, century, centimeter, etc., meaning hundred or hundredth; the *cé* is equivalent to 14·4 minutes or nearly our quarter-hour. The *cé* is again divided into 10ths, 100ths, and 1000ths, known as *decicés*, *centicés*, and *millicés*. Another watch-hand records the *decicés* and a third, very fine needle corresponding with our ordinary 'seconds' hand, records the *centicés*. Each of these latter is approximately 1 second—0·864 second—long. Time is recorded as so many *cés*; thus 3·456 (*cés*) represents 12:49:46 a. m. Similarly 78·9 (spoken in much the same way as we would say 'six fifty-six') means 6:56:10 p.m. The labourer who has worked between these limits has 'put in' a day's work represented merely by the difference between the two numbers, or 75·444 *cés*, or 0·75444 of a day. Compare this with subtracting the difference between 6:56:10 p.m. and 12:49:46 a. m., and then computing the fractional part of a day thus represented.

The division of time into such awkward units as our hours, minutes and seconds is clumsy enough, even without the awkward break into a. m. and p. m. In large plants, especially plants which operate throughout the 24 hours of the solar day, untold sums are squandered annually through the clumsy a.m.-p. m. and hour-minute-second systems in recording the working time of employes or the working time and efficiency of machinery and power consumption. And in underground mines where the light of day never penetrates, and where a distinction between a. m. and p. m. would be most essential on the dial of a watch the ordinary watch fails utterly to make such a distinction. In short, the a. m.-p. m. division fails to work where the proper recording of a. m. or p. m. time is essential.

The new *cémeter* will be welcomed as a boon in abolishing the separating counting of a. m. and p. m. time. The rest of the propaganda of the *cémeter* (the division and recording of the solar day in decimal time), is an admirable conception, which will, however, require many year's time in overcoming the inertia of conservatism, before a change can be effected from our hours-minutes-seconds system."—*Mining and Engineering World*, Aug. 19, 1911, p. 323. (A. R.)

THE METALLURGICAL FUTURE OF INDIA.—"The development of the natural re-ources of India and their resulting industries is, and must necessarily be, more a matter of judicious government encouragement than in countries where private enterprise is more in accordance with political conditions. The existence of various intricate systems of established rights renders interference with existing customs a process of grave importance, and one which cannot always be entrusted to a commercial body with more prospect of permanent success than can the public safety be handed over to the care of the half-educated amateur imported politician.

Previous to 1899 the complications caused by a fluctuating exchange made the extension of mining concessions impracticable, the financial return being too uncertain to form a steady inducement to private enterprise. The regulation of the coinage question which came into effect in that year, however, permitted the Indian Government to take the matter up and frame regulations which offered more scope for private enterprise.

Some idea of local conditions, and the care which must be exercised in making alteration, however slight, in the usages of centuries, can be gathered from the comprehensive and instructive paper recently read before the Indian section of the Society of Arts by Prof. Sir Thomas H. Holland, formerly director of the Geological Survey of India.

Taking the products in order, coal naturally leads in importance. During the past 25 years the production of coal has risen from 1¼ million to 12 million tons a year, and still increases, the best being that obtained from the Giridih field. This field is estimated to contain about 70 million tons.

The coal fields of Assam are at present worked principally for the local market, expense of transport cutting them off from Calcutta, as well as the quality of the coal, which is friable and liable to spontaneous combustion, and thus not easily carried for long distances.

The development of this industry depends very largely upon railway extension, both as providing a market and a means of transport. The available foreign markets, says this paper, are comparatively small, the principal consumption being due to locomotive wants. It seems possible, however, that a market for shipping might be cultivated where the quality of coal permits. At present, and since 1900, the exports of coal largely exceed the imports, which latter have been generally limited to small quantities to meet local and temporary variations in the demand for special coals. The exports have, on an average, been over half a million tons per annum.

With regard to petroleum the difficulties due to ancient customs for a considerable time hindered the development. When these were overcome the output increased rapidly, rising during the past ten years from 37¾ million gallons to over 233½ million gallons (1909). Certain of the fields, however, show signs of exhaustion, the Twingon, in Upper Burma, for instance. On the other hand, three promising properties remain to be developed in Burma, the Khodaung, Singu and one on the other side of the Irrawaddy.

At present the indigenous production is insufficient to meet the country's requirements in kerosene and petrol, of which 90 million gallons are imported annually.

The production of iron is at present low in the peninsula, only one company operating with favourable results (the Bengal Iron & Steel Company), although another (the Tata Iron & Steel Company) has now been floated with a capital of £1,500,000 sterling to exploit the hematite ore deposits of the Mombhanj State in Orissa. The new works will be within forty-five miles of the principal ore supplies and 180 miles from the collieries. The ore in the Mombhanj is rich, carrying over 60% of iron, and exceptionally easy to work, standing in hills above the general plain. In a period when rich ores were in demand they might thus be worth the attention of European furnaces.

Data of other fields do not seem available, although iron is known to exist, in small quantities at any rate, throughout the peninsula.

Turning to minerals raised chiefly or entirely for export the author finds that gold is only mined in Mysore and at Hutti in the Nizam's dominions. The principal opening for development appears to be on the upper reaches of the Irrawaddy, where dredging operations for alluvial gold have been carried on for the past nine years. The production in 1909 was valued at £32,730.

Manganese is a flourishing industry and appears capable of considerable and rapid expansion whenever the market provides the necessary encouragement, as shown by the output in 1907, which expanded from 571,500 tons to 902,291 under this stimulus. The falling off in steel manufacture and lower prices have since caused the output to drop to 600,000 tons.

In mica the country has a valuable product, which only needs control by organised limited liability companies to become of first-class importance. Even under present circumstances the production is equal to 61·78% of the quantity produced by the three principal mica-bearing countries—India, Canada and the United States. The present methods of winning are hampered by the disadvantages consequent on divided effort and may be regarded as mere prospecting. Numerous deposits still lie untouched.

The further development of India in this connection may bring forward copper as a potential product. Copper ores are known to occur, and, in the past, have been considerably worked in the Nellore district of the Madras Presidency, in Rajputana, in Chota Nagpur, and at various places along the lower Himalayas from Kulu in the northwest to Sikkim and Bhutan in the east. Operations are now in progress by the Cape Copper Company to test the probable value of a copper-bearing country in Bengal. Prospecting operations are also in progress in the Darjeeling district and adjoining parts of the Sikkim State.

Lead, zinc and silver ores are known to exist and are being prospected on a considerable scale in upper Burma. Pitchblende and wolfram are also found, the latter in some quantity. Corundum is widely distributed in various forms, but it is possible that, in face of competition from natural and artificial abrasives in the countries of principal consumption, it would not pay at present to develop the Indian mineral for export to Europe and America.

The principal requisite for further development in the production of minerals in this country appears to be means of transport."—Sir T. H. HOLLAND.—*Metallurgical and Chemical Engineering*, August, 1911, p. 421. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

FAULTS AND DYKES: A GEOLOGICAL STUDY OF THE WITWATERSRAND.—By J. S. OLVER. (Messrs. J. C. Juta & Co.)

"This book is likely to be of decided value to mining engineers generally, and which is especially interesting to those connected with mining on the Witwatersrand. The author is a mining engineer who has had eight years' experience of mining in the Transvaal, and who has made a special study of underground problems. He formed the opinion very early in his career upon the Rand that there was a great need for the introduction of more scientific methods into the conduct of underground operations, into the study of reefs and values and the true economy of mining, and more especially into the consideration of fault problems. This book represents the fruit of his personal study. He has endeavoured to elevate the handling of fault problems into a science according to methods of his own, and whether or not his book be accepted as a standard, it must be recognised that it is the outcome of a large amount of original thought and study. Every fault movement is resolved into three components, each of which is referred to a definite basis. The methods of determining the true movement of a fault are clearly set out; and, when the components of its movement have once been determined, the effect of the fault upon any intersected plane may be accurately calculated. A remarkable feature of the book is, indeed, the degree of certainty with which the most complex problems seem to have been worked out, as well as the magnitude and diversity of the movements involved in these problems. The author shows that the visible movement of a fault is generally estimated by the mere raising or lowering of the plane of a reef or bed, and that the true movement is often vastly greater than the apparent movement. Many illustrations are given for the purpose of showing the possibilities of faulting, and some of the effects of interacting faults are positively startling. Most of these illustrations are drawn from actual practice.

Mr. Olver also describes in detail his methods of securing the data required for the solution of fault problems. Readers will probably be surprised at the minuteness of the examination entailed, and at the mass of detail which must be observed and kept in mind if fault problems are to be solved in a satisfactory manner.

The second part of the book deals with the general geology of the Witwatersrand. Mr. Olver conceived the idea some three years ago that many of the apparent contradictions in the geology of the Central Rand could be explained by the intervention of faults, and he has since then devoted a large part of his time to the study of this large problem. Whatever the final opinion of geologists with regard to his conclusions may be, it must be recognised that he has made a very thorough study of the area concerned, and that his conclusions are the result of careful thought."—*S.A. Mining Journal*, Sept. 2, 1911, p. 1090. (A. R.)

OPERATION OF THE GOLDFIELD CONSOLIDATED MILL.—By J. W. HUTCHINSON. (Pamplet, 20 pages. Illustrated. San Francisco: *Mining and Scientific Press*; London: *The Mining Magazine*.) Price 2s.

"This is a reprint of a series of articles that appeared in the *Mining and Scientific Press* during May and June. It gives full practical details of the work done at one of the most modern all-sliming plants, and will be of great help to the cyanide metallurgist."—*Mining Magazine*, July, 1911. (A. R.)

GOOD ENGINEERING LITERATURE.—By HARWOOD FROST, formerly editor of the *Engineering Digest*, Chicago Book Co., 226, South La Salle Street, Chicago, Ill.

"The subject of this book is one of growing importance to both the practising engineer and the student of engineering, and the author has endeavoured to bring together information that may serve as a guide for technical men who may have aspirations toward literary success. The book contains 420 pages, is cloth bound, and is 5 in. × 7½ in. The price has been kept at the low figure of $1 in order that its purchase may be no hardship to the beginner of professional life. Every engineer is sooner or later called upon to do some form of literary work, such as the preparation of reports, specifications, and contracts; the writing of descriptive papers for technical societies and periodicals, the discussion of engineering problems with committees, boards of directors, and public gatherings composed of men with little or no technical knowledge. In the course of some professional duties as well as in ordinary correspondence and conversation the engineer will find that the ability to speak and write clearly and forcibly, to express his thoughts and understandings, and to describe his works so that others will understand him, will prove one of the most valuable items in his mental equipment.

The book is divided into 22 chapters and is furnished with an index. Chapter I. is on Literary Expression; Chapter II. is on Rhetoric and Grammar; Chapter III. deals with Orthography and Punctuation; Chapter IV. treats on Words and Phrases; Chapter V. has Inspiration and Motive in Literary Work as a subject; Chapter VI. shows the essentials to success in literature; Chapter VII. tell what to write about; Chapter VIII. advises on the collection and arrangement of material; Chapter IX. is on Exercising the Memory; Chapter X, explains the preparation of manuscript; Chapter XI. covers the field and policy of technical journals; Chapter XII. shows how to write up articles; Chapter XIII. is on the rights of an author in his works; Chapter XIV. Copyright; Chapter XV. Relations between Author and Publisher; Chapter XVI. the Law of Libel; Chapter XVII. Preparation of Illustrations for Reproduction; Chapter XVIII. the Making of a Book; Chapter XIX. Indexing and Filing; Chapter XX. Literary Criticism; Chapter XXI. The Engineer's Library; Chapter XXII. List of Technical Indexes appearing Serially.

This book will be found good reading, as it contains much information which men are apt to forget after finishing their school days. Very few men are able to dictate a letter satisfactorily, and it is believed that by reviews and writings of this kind, this important branch of business will be strengthened."—*Mines and Minerals*, Aug., 1911, p. 3. (A. R.)

MODERN PRACTICE IN MINING, Vol IV. "The Ventilation of Mines." By R. A. S. REDMAYNE, M.Sc., M.Inst.C.E., M.Inst.M.E., M.Inst.M.M., F.G.S., His Majesty's Chief Inspector of Mines, Late Professor of Mining in the University of Birmingham, Certified Colliery Manager, etc. 6s. 6d. net. (Longmans, Green & Co., London.)

"In the introduction to this book the author says: 'So far as the author is aware, there is no work devoted entirely to a comprehensive consideration of the subject of mine ventilation, though there are several excellent books which treat of certain sides of the subject. It is hoped, therefore, that this little volume may be of some service to the young mining student in arriving at an elementary knowledge of the principles and practice of mine ventilation.' The term comprehensive, in its full sense, scarcely fits the work under review, but as an aid in acquiring elementary knowledge of the subject of mine ventilation the book may be recommended. On examination, the mathematical portion of the text, it must be confessed, does not impress us as a model of clearness, or even of exactitude, and, unfortunately, it is precisely that part of it which the student will principally look to for assistance and enlightenment. There is an interesting section on fans, and it may be added that a summary of the recent paper by Mr. Penlerick on the East Rand Proprietary Mines' scheme of ventilation finds a place in the volume. As a general review of the subject, the book is one which should undoubtedly prove of use, and, since references are given, the reader who wishes to pursue his enquiries more deeply will be able to check the calculations and formulæ contained in it."—*South African Mining Journal*, Sept. 9, 1911. (A. R.)

Blount, Bertram, and Bloxam, A. G. Chemistry for Engineers and Manufacturers. 2nd Ed. Revised and Enlarged. 8vo., pp. 404. *C. Griffin.* Net 14s.

Cole, Grenville A. J. The Changed Earth: An Introduction to the Record and the Rocks. Cr. 8vo., pp. 234. *Macmillan.* Net 1s. 6d.

Czapeck, Frederick. Chemical Phenomena in Life. (Library of Living Thought). 12mo., pp. 162. *Harper.* Net 2s. 6d.

Dutton, Thomas. The Cult of the Open-Air Treatment of Tuberculosis. Reprinted from the Medical Times. Cr. 8vo., pp. 12. *Penfold.* Net 6d.

Ely, Leonard W. Joint Tuberculosis. Illust. Roy. 8vo., pp. 256. *Simpkin.* Net 12s. 6d.

Kaup, W. J. Machine Shop Practice. Cr. 8vo. *Chapman & Hall.* Net 5s. 6d.

Mastin, John. The Chemistry, Properties and Texts of Precious Stones. 12mo., pp. 122. *Spon.* Net 2s. 6d.

Roscoe, Sir H. E., and Schorlemmer, C. A Treatise on Chemistry. Vol. I. Non-Metallic Elements. New Ed. Completely revised. 8vo., pp. 968. *Macmillan.* Net 21s.

Rosenhain, W. K., and Archbutt, S. L. The Constitution of the Alloys of Aluminium and Zinc. 4to., sd. *Dulan.* Net 2s. 6d.

Subject.—List of Works on Chemistry (including Alchemy, Electro-chemistry, and Radio-activity) in the Library of the Patent Office. 18mo., sd., pp. iv. 214. Patent Office.

Abstracts of Patent Applications.

(C.) 503/10. George Hardy Stanley. Improvements in methods of classifying or grading pulverulent material. 13.10.10.

This patent application refers to an invention consisting in the combination of a spitzlutte or a similar apparatus, with a reciprocating screen placed in such a way that it will return any oversize product back to the bottom of the apparatus for discharge.

(C). 504/10 and 505/10. George Hardy Stanley. Improvements in classifying or grading apparatus. 13.10.10.

These two patent applications are practically similar, and refer to improvements in classifying apparatus consisting in the combination with an hyraulic classifier or a launder, of a reciprocating screen placed in such a way that oversize particles will be returned to the bottom of the apparatus for discharge (either free or mechanical discharge).

(C). 548/10. John Fraser Price. Improvements in apparatus for treating crushed ore products with cyanide solution or other liquid. 10.11.10.

This application relates to apparatus for agitating and aerating crushed ore and solution, and consists of a hollow rotatable cylinder provided with vanes and transverse partitions forming pockets, which elevate and drop the pulp on its passage through the cylinder. At the discharge end, similar vanes are provided which lift the pulp into a launder which conveys it wherever desired. A pipe is also provided to convey compressed air or steam into the cylinder to facilitate the aeration.

(C). 569/10. William Colin Hazelhurst. Tube mill and liner. 21.11.10.

This application relates to improvements in end liners of tube mills, which consist in providing segmental wearing plates to be bolted to head or head lining of the mill. These plates are constructed with ribs, suitably spaced to automatically hold pieces of the grinding charge and so take up the abrasion.

(C). 643/10. John Collins Clancy. The treatment of ores bearing precious metals. 30.12.10.

This application applies to a new process for treating ores containing precious metals, and is based upon the re-action of cyanamide and a cyanogen-bearing compound or "prussiate."

The applicant claims to have discovered these mixtures to be very active solvents of the precious metals contained in ores.

The claims of the inventor include :

1st. The use of an amide or amidine solution containing a cyanogen bearing compound.

2nd. The use of cyanamide solution containing a cyanogen bearing compound.

3rd. The process of treating ore containing precious metals by subjecting them to the action of the cyanamide solution containing a ferro or ferricyanide compound.

4th. The process of treating ore which already contains or is in association with soluble or insoluble "prussiates" by subjecting them to the action of a cyanamide solution.

(C). 179/11. Frank Ondra. Apparatus for precipitation or treatment of liquid. 27.3.11.

This application refers to an apparatus for continuously treating liquid containing metals in solution with a precipitant, separating the precipitate and passing off the treated liquid.

It consists in the combination of two vessels connected at the bottom, and the means for passing liquid into the upper part of the vessel and out on the top of the second vessel. The necessary appliances for supplying the precipitant fluid in the first vessel, and sundry other appliances for controlling the outflow of liquid, separating the precipitate obtained, etc., are provided.

(C). 306/11. Frederick Walter Cindel. Tube Mill Feeder. 29.5.11.

This application relates to improvements in feeding pebbles or broken rock to tube mills, and describes a device consisting of "a bin or hopper, a reciprocating tray and a cam on the peripheral part of the mill and means for the cam to operate the reciprocating tray."

(C). 452/11. James Henry Alling. Process for recovering metals from ores. 11.8.11.

This application relates to the recovery of gold or other metals by a process of mixing the pulp containing the auriferous material with a solution of salt, or another chloride, and passing same through an electrolytic apparatus which has anodes of graphite and cathodes of mercury. The intention of the process is that the coarse clean particles of gold are caught direct by the mercury, while the finer particles are dissolved by the nascent chlorine, and then deposited electrolytically on the same material acting negatively as a cathode.

An apparatus described consists of a sluice box, where the cathode mercury is employed in riffles and the graphite anodes act as baffle plates. The salt solution is added at the head of the box from a special supply tank.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 486/11. Irving John Cox. Improvements in or relating to the treatment of sulphuric acid. 1.9.11.

(C.) 488/11. Hans Grohmann. Furnace for producing endo-thermic gas reactions. 1.9.11.

(P.) 489/11. William Gowan. Improvements in rotary grinding machines. 2.9.11.

(P.) 490/11. Edward Emery Hardach. Drill bar. 4.9.11.

(P.) 491/11. John Rankin Smith Rockcliffe. A device for electric surface warning bells applied to hoisting engines. 6.9.11.

(P.) 492/11. Walter Daniel Douglas. Improvements in wind mills. 6.9.11.

(C.) 493/11. Henry William Rice. Improvements in or relating to wire bound boxes. 6.9.11.

(P.) 494/11. George Chisholm. Improvements in feeding means for tube mills and similar rotary grinding machines. 6.9.11.

(P.) 495/11. Ernst Heinrich Thiel. Improvements in rock drills. 7.9.11.

(C.) 496/11. Robert Davidson (1), Charles Harrison (2). An improved method of cleaning silver and electro-plate. 7.9.11.

(P.) 497/11. Basil Vivian Blundun (1), Frederic Houghton Rogers (2). Improvements in shaking chute conveyors. 8.9.11.

(P.) 498/11. John Lang (1), John Johnson (2). Improved roller supports. 8.9.11.

(P.) 499/11. Charles Edward Stewart McCann. Calcining furnace or oven. 9.9.11.

(P.) 500/11. Joseph Johnson. Arrangement for converting reciprocating into rotary motion also rotary into reciprocating motion. 9.9.11.

(P.) 501/11. Kenneth Kemp. Improved crushers. 11.9.11.

(P.) 502/11. Harry August Stockman. A new or improved corrugated metal reinforcement for building and other structural purposes. 13.9.11.

(P.) 503/11. Phillip Henry Penny. Improvements applicable to classifying cones and the like. 13.9.11.

(P.) 504/11. Sidney Harold Hills (1), George Wynne Vice (2). Jockey for ore trucks and the like. 13.9.11.

(P.) 505/11. George Hardy Stanley. Improvements relating to processes and apparatus for screening ore and the like. 13.9.11.

(P.) 506/11. George Hitchens. Improvements in stretcher bars for rock drills. 15.9.11.

(P.) 507/11. Henry Jeremiah Smith. Improvements in jockeys for mechanical haulages and the like. 15.9.11.

(C.) 510/11. John Broad Roberts. A system and mixture for lubricating air compressors and purifying compressed air. 16.9.11.

(P.) 511/11. Joseph Henry Dale. Improvements in the linings of tubular grinding mills. 16.9.11.

(P.) 512/11. William Henry Kendall. Improvements in rock drills. 16.9.11.

(P.) 515/11. David Shannon. Pneumatic concentration. 19.9.11.

(P.) 516/11. Richard Bennett Thomas (1), Frederick George Fell (2), and Wm. Delbridge Thomas (3). Improvements applicable to the chucks of rock drilling machines and the like. 19.9.11.

(P.) 517/11. William Mackenzie. Improvements in linings for tube mills and the like. 20.9.11.

(P.) 518/11. Joseph Nicholas Ross. Sifting, washing, sorting, and concentrating of ores, soils, and other substances. 21.9.11.

(P.) 519/11. Jan Benjamin Potgieter. Machinery worked automatically by the action of a magnet upon iron. 21.9.11.

(P.) 520/11. David Laird. A coating for metallic or earthenware surfaces of corundum or carborundum to prevent abrasion. 21.9.11.

(C.) 522/11. William Climie. Improvements in and relating to gas producers. 22.9.11.

(C.) 523/11. John Cecil Burton (1), Hubert Henry Burton (2). Improvements in or relating to the jointing of ropes of rubber to be used for wheel tyres and the transmission of power. 22.9.11.

(C.) 524/11. Bean's Bunsen Smokeless Furnace Company, Ltd. (1), Orestes Utah Bean (2). Improvements in or relating to the furnaces or flues of internally fired boilers. 22.9.11.

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(P.) 528/11. Louis Lawrence Bremont (1), Friederich Rolle (2). Improvements relating to mine trucks and similar vehicles. 28.9.11.

(P.) 529/11. William Colin Hazelhurst. Improvements in linings for the ends of tube mills. 28.9.11.

(P.) 530/11. Ferdinand Clemens Driessen. An invention relating to the lighting of fuse-wire as used with explosives in blasting operations. 29.9.11.

(C.) 531/11. Arthur Francis Berry. Improvements in or relating to electrical transformers. 29.9.11.

(C.) 532/11. The Flottmann Engineering Co., Ltd. (1), Adolf Sgonina (2). Improvements in connection with rock and like drilling apparatus. 29 9 11.

(C.) 533/11. Turbo Amalgamator and Extraction Co., Ltd. (1), Alfred Andrew Lockwood (2). Improvements in the amalgamation of the precious metals, and in apparatus therefor. 29.9.11.

(C.) 534/11. George William Gay. Improvements in or connected with automatic couplers for the rolling stock of railways. 29.9.11.

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SPENCE, A. G. B., *l/o* Denver ; Vogelstruis Estates and Gold Mines, Ltd., P. O. Box 116, Roodepoort.

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Vol. XII. NOVEMBER, 1911. No. 5.

Proceedings
AT
Ordinary General Meeting, November 18, 1911.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, November 18th, Mr. E. J. Laschinger (Vice-President) in the chair. There were also present:—

43 Members: Messrs. W. R. Dowling, J. Gray, T. Johnson, F. W. Watson, H. A. White, Prof. G. H. Stanley, W. A. Caldecott, E. H. Johnson, R. G. Bevington (Members of Council), H. Adams, R. A. Barry, W. Beaver, A. J. Bowness, J. Chilton, W. M. Coulter, C. Crichton, R. C. Evans, R. Gascoyne, B. C. Gullachsen, F. G. Guthrie, W. H. Jane, A. J. Johnson, A. M. Kratz, G. A. Lawson, J. Lea, G. Melvill, S. Newton, W. J. North, A. C. L. Olsen, E. A. Osterloh, B. E. Pollard, W. S. V. Price, J. F. Pyles, S. C. Quin, E. Roberts, A. R. W. Stacpoole, W. A. C. Tayler, A. Thomas, Prof. G. R. Thompson, C. Toombs, J. Watson and H. Warren.

21 Associates and Students: Messrs. H. Ball, J. V. G. Clarence, C. L. Dewar, I. B. Dicks, J. Gibson, W. J. R. Hunter, W. D. Jack, A. King, L. T. Leyson, G. H. Little, B. W. McLuckie, G. F. Mathews, G. S. Paterson, F. J. Pooler, R. A. Porter, A. Purser, A. M. Thomas, J. Thorlund, P. A. Tucker, L. A. Womble and S. A. Woolf.

9 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as published in the October *Journal*, were confirmed.

NEW MEMBERS.

Messrs. T. Johnson and R. Gascoyne were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that both the candidates for membership had been duly elected as follows:—

FINLAYSON, KENNETH JOHN, P. O. Box 94, Benoni. Cyanider.
WOOD, EDWARD THOMAS, P. O. Box 77, Roodepoort. Sampler.

The Secretary notified that since the last meeting of the Society the following Associates and Student had been admitted by the Council:

As Associates—
HARRIS, FREDERICK, Brakpan Mines, Ltd., P. O. Box 3, Brakpan. Assistant Assayer.
PATERSON, GEORGE STEWART, Anglo-French Exploration Co., Ltd., P. O. Box 2927, Johannesburg. Mining Engineer.
SMIDT, SAMUEL DE, Robinson Deep G. M. Co., Ltd., P. O. Box 1392, Johannesburg. Mine Captain.
WHITE, PERCY, 58 Doran St., Jeppes Extension. Junior Amalgamator.

As Student.—
THORLUND, JOHANNES, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

GENERAL BUSINESS.

Mr. Andrew F. Crosse (*Past-President*): I wish to bring before the notice of members what may be regarded as a new idea for this Society. As you know, meetings have, in the past, been held every month, and the Council has decided to hold the December meeting this year on the 16th December—Dingaan's Day. That, of course, is not a very good day, but it is better than the Saturday following. Now I do not see why we should always have twelve meetings a year, and if someone will second me, I think it would be rather a good plan to drop the December meeting. This would also be in the nature of a holiday for our Secretary, who has a great deal of work to perform, in addition to which, the proximity of the Christmas holidays is not conducive to a big attendance at the December meeting.

There was no seconder to the proposal.

The Chairman: It seems to me that there is something in Mr. Crosse's suggestion, but again, it is obvious that we have so much work on hand, that I fear it would rather seriously hamper the affairs of the Society if we were to drop the December meeting, and of course the *Journal* which belongs to it. I should like to suggest that Mr. Crosse should bring this matter up before the Council for their consideration, and if they consider that it would be a wise and desirable thing to do, they could refer the matter to members at a regular meeting. In the meantime we would be able to do nothing as regards the December meeting this year as the Agenda is prepared and everything ready for the meeting.

PROPOSED MINING EXHIBITION.

The Secretary: Some months ago a proposal was submitted to the Council by Mr. Andrew F. Crosse, that an exhibition or museum of articles of interest to chemists, metallurgists and miners on the Witwatersrand, should be inaugurated by the Society. The subject has received much consideration by the Council and on the suggestion of Mr. William Cullen, the original scope of the proposal has been enlarged, and it has been decided now to hold a mining exhibition in Johannesburg, in February, 1912.

With the co-operation and kind assistance of the Council of the School of Mines this exhibition will be held in the Assembly Hall of the School of Mines, which has been lent for the purpose.

The scope of the exhibition may be outlined generally, as follows:—Safety and rescue appliances (such as helmets, stretchers, and other ambulance appliances, etc.); chemical, metallurgical and mining apparatus and models of every description; specimens of crude and manufactured natural products of the country, as for instance: asbestos; graphite: mica; salt; fireclay; limestone and shales for the manufacture of cement; magnesite; oil-shales; copper, iron and other ores; etc., etc.; plans, diagrams and photographs of mines, works and plants, machinery and apparatus, etc., etc.

During the time the exhibition is opened next February, it is proposed to have a series of lectures and demonstrations by experts on some of the subjects dealt with by the various exhibits, such as life-saving in mines, ambulance work, mine ventilation, manufactures, etc.

The desire of the Council is that the exhibition should be the means of bringing before all those engaged in mining and metallurgical work on the Witwatersrand and elsewhere, the latest developments in apparatus of every description likely to be of use, or used, on or in the mines; and by means of the lectures and demonstrations to bring before those who are employed on the surface or underground, the best methods to be adopted for the saving of life, by safety or ambulance appliances, by the allaying of dust, ventilation, etc.

The space available for the exhibition is not large, and consequently the sizes of the exhibits will, to a certain extent, have to be limited. It is proposed to reserve some portion of the space for commercial firms desirous of exhibiting apparatus, models of interest, etc., for which a moderate charge will be made in order to cover part of the expense. It may be pointed out, however, that no selling of goods can be permitted in the Hall, and that all exhibits will be subject to the approval of the Exhibition Committee.

Members of the Society who are desirous of exhibiting apparatus or models of any description are requested to communicate with me as soon as possible, and it is hoped that mining and metallurgical men who may have models of any device or apparatus connected with their work will avail themselves of this opportunity to bring them before the notice of the public.

In order that this exhibition may achieve the success it deserves, the Council hopes this scheme will commend itself to you, and that it will receive your hearty support and co-operation.

The exhibition is primarily for the information and benefit of those engaged in mining work, and is not in any way looked upon as a means for financial gain to the Society. No soliciting for orders for goods or apparatus exhibited will be permitted, and admission to the Hall will be free.

ACCIDENTS IN TRANSVAAL MINES.

(With Special Reference to Falls of Ground.)

By James Chilton (Member).

Mining has always been regarded as a dangerous occupation and no amount of caution or vigilance can ever make it otherwise; wherever it is carried on, loss of life and human suffering may be regarded as part of the price that must be paid for the production of mineral wealth. Although absolute safety is impossible and accidents cannot be completely avoided, yet they may be reduced in number, and the aim of the writer is to direct the attention of mining men to this feature of our mining practice with a view to bringing about a reduction in the accident rate.

For some time past, the opinion has been steadily growing, that there is an excessive sacrifice of human life going on in the Transvaal mines, Our accident death rate has been the subject of many interrogations in the British Parliament; has received severe criticism at the hands of a Royal Commission: it has been discussed *in camera* by the Mine Managers' Association; and every year it occupies a great space in the Government Mining Engineer's Report: yet in spite of this publicity and discussion, our accident rate goes on increasing, until to-day we hold the ignoble distinction of possessing the highest accident rate of any mining district in the world. Our coal mines are practically free from explosive gases and our gold mines are seldom visited by a tidal wave of calamity such as is common in mining districts in other countries; still it will be found at the end of the year that the Transvaal rate far exceeds any other. It will come as a shock to most minds to realise that on every full working day in 1910 three persons were killed and six injured. Probably the most alarming feature is the serious increase in the number of accidents per 1,000 employees, which rose from 9·45 to 10·23—an increase of over 300 separate accidents. During the year under review 915 persons were killed and 1,677 persons injured, a number that can only be regarded as appalling when compared with the English rates.

TABLE I.

Death Rate per 1,000 Employees.

		England.	Transvaal.
1903	...	1·35	4·04
1904	...	1·35	4·75
1905	...	1·42	4·53
1906	...	1·42	5·55
1907	...	1·46	4·71
1908	...	1·46	4·18
1909	...	1·61	5·29
1910	...	1·91	4·29
*1911	...	—	3·95

It will be seen from the above table that our death rate is far in excess of the English rate, and this fact is at once a criticism of our mining methods, a challenge, and an appeal to us all. When compared with England on a tonnage basis the position is still more humiliating, for while that country raised one million tons for a death rate of 4·472, the Transvaal death rate was 31·51 per million tons raised.

The definition of what constitutes an accident varies in different countries, so that no useful purpose can be served by comparing ordinary accident rates. In England an accident is an injury, which, in the opinion of a qualified medical man, may result in an injured person being incapacitated from work for at least seven days. This period in the Transvaal is extended to fourteen days, and other countries have varying periods. The Commission on Mining Regulations, 1909, reporting, say, "the accident rate in the Transvaal is abnormally high," and this certainly must be the opinion of every person who has given serious thought to the subject.

Many reasons are brought forward to explain this deplorable state of affairs, and an examination of the Mines Department statistics for the last 7 years is interesting.

The following table, which deals with 11,631 accidents can only be regarded with mingled feelings of pleasure and regret. One of the pleasant things worth noting is the remarkable reduction that has taken place during the last seven years in accidents due to defective plant or material. In 1903, 4·35% of all accidents were due to defective plant or material, or in other words to the fault of the employer, this in 1910 has been reduced to ·87%. This decrease is extremely gratifying, and great praise is merited by our mechanical engineers, to whom this remarkable diminution is chiefly due.

TABLE II.—All Accidents.

	Due to defective Plant or Material.	Owing to danger inherent in work itself.	Carelessness or ignorance of injured.	Through fault of fellow workmen.	Total Accidents
1903	4·35	45·96	34·82	9·85	649
1904	3·24	60·78	30·84	8·14	895
1905	2·15	58·49	30·74	8·62	1,207
1906	1·33	48·76	38·99	10·92	1,657
1907	2·02	55·07	33·94	8·97	1,538
1908	1·05	61·23	27·90	9·82	1,620
1909	1·59	65·09	23·43	9·89	1,882
1910	·87	69·22	22·35	7·56	2,183
*1911	·51	69·03	20·31	10·15	1,172

Column 2 gives cause for some anxiety, for it points to the fact that our mines are becoming more dangerous to life and limb. Since 1903 there has been a gradual increase of accidents "owing to danger inherent in the work itself." In that year 296 accidents were attributed to that cause, and in 1910 this had risen to 1511 or otherwise 69·22% against 45·96%, an increase of 23·26%. This alarming increase is not due to abnormal circumstances but is probably brought about by some cause that is making its presence felt increasingly year by year. Mining statistics prove that as our mines become deeper they also become more dangerous. The Mining Engineer's report for 1910 gives a list of the twelve gold mines of the Witwatersrand which show the highest and lowest accident death rates for the year, and it is a remarkable fact that only one deep level mine appears in the list having the lowest death rate. While rejecting abnormal accidents, 8 out of the 11 mines showing the highest death rate are deep levels.

Dividing these producing mines into outcrops and deep level divisions it will be seen that the deep level death accident rate is 6·72, while the outcrop rate is only 1·67 per 1,000 at work. It follows not unnaturally that our deepest mines head the list. The above figures do not apply to developing mines, as no complete list of these has been made.

Many reasons can be advanced for this increasing death rate in our deep level mines; with mining at greater depth we get greater roof pressure, and this will be found increasingly difficult to support. This aspect is dealt with further under "Falls of Ground." Miners assert that the increase is due to the methods of "speeding up" now employed in our gold mines and especially in our deep levels, where the rush to keep the mill going and to earn a living wage is such as to leave little time for thought or preparation for meeting an emergency. Dangerous roof, misfires, and other dangers are disregarded in the rush and hurry of the day's work. This acceleration of speed is acknowledged in the daily speech of the Rand miner whose "deep level move" is synonymous with the "all-fired hustle" of his American *confrère.* With deeper mining a longer time is taken up in getting from the bank head to the working faces, consequently the time spent there is correspondingly shorter. In outcrop mines most of the underground employes walk to their working places, but in deep levels this is impossible. The omission of the dinner hour on most mines now, compels the white overseer to do as much work in 9 hours as he formerly did in 10. This all accentuates the rush, and to drill the round and complete the bench requires greater speed and higher skill than was necessary in the earlier days of the gold fields. The modern tendency to shorten hours of labour, although desirable from many other points of view, will certainly increase the accident rate. It will be noticed from Table I. that the accident death rate in England has gone up by 30% since the introduction of the eight hours' system. Miners' phthisis, due largely to the increase of rock-drills in our deeper mines, further contributes, by eliminating the old experienced miner, to our increasing death rate. Carelessness or ignorance on the part of the injured was responsible for 22·35% of the accidents in 1910, a decrease in this class of accident of 12·47% over the seven years reviewed. Ignorance in our native worker is the natural product of his environment, and accidents from this source will always bulk largely in our mining returns. The Government Mining Engineer, Mr. R. N. Kotzé, in his report this year, says, "the majority of the native workmen on these fields are mere savages, and upon their arrival are totally ignorant of mining and the dangers connected therewith. It is impossible to make them understand what they must, or must not do, as in many cases the compound manager can only speak to them with the aid of two interpreters." When it is remembered that 41 dialects are spoken in the Rand compounds it will be seen that the language difficulty is a formidable one. If we could persuade our native workers to remain at the mines for longer periods, our accident rate would appreciably decrease, for after a period spent among the home delights of Kafirdom, the old worker usually returns with a mind that is almost a perfect blank, and his mining education must begin all over again. At present as soon as a native has begun to appreciate the dangers of mine life, his short contract finishes and he returns to his native territories. Dr. Maynard in his report on native mortality, 1910, does not agree with the above view that old native workers are less liable to accident than raw natives, for he says "the accident death rate shows comparatively little tendency to decrease with increased period of work." The slight difference in rate is probably due to the fact that when any extra hazardous work has to be done on the mines, old and experienced natives are usually taken as assistants, consequently their accident rate is increased till it almost rivals that of the new recruits.

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* For six months only.

* For six months only.

The carelessness of white workers is difficult to deal with; threatenings, dismissal, prosecution, all have been tried with indifferent success. The only remedy the writer can suggest is education in mining safety methods. The excellent work of the S. A. School of Mines needs no advertisement, and it has done and is doing a great work in raising the educational level of the mining population, but its work consists in giving a general mining education rather than the laying of particular emphasis on any special branch of the subject. The writer feels that the "prevention of accidents" needs distinctive treatment, and the recommendation of the Royal Commission on Safety in Mines, England, 1908, seems to suit our local conditions and merits careful consideration, viz., the institution of lectures on "Safety in our Mines." The Government and the Chamber of Mines, with the assistance of this Society, the Mine Managers' Association and any other that felt interested, should appoint a lecturer who would go carefully through the Mining Engineer's reports, and give a study of the most important accidents that have happened during the year and the lessons taught by them, and exhibit after the lecture safety appliances, sprays, respirators, safety helmets, rescue apparatus, and any invention or new method of work that will tend to reduce our mining death rate. If the above were found unworkable lectures by specialists ought to be tried. The writer recalls with pleasure and profit two lectures given by members of this Society in mine recreation halls—that of Mr. Wm. Cullen on "Explosives," and of Dr. Macaulay on "Miners' Phthisis." These lectures drew large and attentive crowds and are still keenly debated wherever miners meet together. This principle could be extended till the whole mining field was covered by lectures on such subjects as "Dustless Drilling," "Ventilation," "Mine Gases," Timbering," "Rescue Work," and any other subject that vitally affects the miner. The lectures could be given at convenient centres, and with the co-operation of mining officials would do a great work in abolishing that perilous half-knowledge which is so common in the Transvaal.

Falls of Ground. It is well known that the majority of serious accidents are due to falls of ground. With frequent vibrations set up by explosions and the constant percolation of water, combined with roof tension and distortion of strata, ground which may appear sound and ring clearly, may in a few days collapse without sign or warning. As accidents from this cause usually occur singly, they do not excite anything like the sensation produced by a large spectacular disaster and most of them pass unnoticed or only arouse a languid interest. Yet in 1910 this steady drip of casualty amounted to 22·57% of the total accidents for the year. Table III. shows the accidents due to this cause since 1904:

TABLE III.

Accidents due to falls of ground.

	Deaths.	Injuries.	Percent. of accident.
1904	98	97	17·65
1905	158	160	20·65
1906	236	260	22·66
1907	212	233	22·21
1908	219	239	23·27
1909	267	178	24·39
1910	268	318	22·57
*1911	128	189	21·74

Mining men all over the world have observed that most accidents from falls of ground occur in districts or mines having good roofs. When the hanging wall is known to be insecure, special precautions are taken and various systems of timbering are tried until one is evolved suitable to local conditions: on the other hand, with a good roof wall the most elementary preventive measures are neglected, pillars are left too far apart, or not left at all, until the overhanging weight overcomes the resistance of the roof beds and a fall occurs. The caving in of large areas is not uncommon in the Transvaal mines,† but these very seldom cause loss of life or injury to underground workers, for this movement can usually be noticed months before the actual collapse takes place, and all work is suspended until the roof has settled down or ceased to move. It is the isolated slab, the small area, the "casual" (!) piece of hanging that is the cause of our high death rate, and it is to these rather than to large areas that attention must be directed. The Transvaal Mines Regulation, Article 97, states:

> "Before commencing work either at the beginning of the shift or after blasting the ganger of the shift shall be the first to enter the working place and until he considers it safe he shall not allow any workman to proceed there except those required to make the place safe."

This regulation if carried out would prevent a large number of accidents, but in most mines it is disregarded, and many natives enter the working place before the ganger arrives, with the too frequent result that a slab of hanging kills or injures some of them. Miners on contract allow this practice because they profit by the extra work done, and many mining officials wink at the contravention, having the short-sighted idea that it is good mining practice. In some of our mines an underground official makes surprise visits to the stopes in the early morning and promptly prosecutes any native found there before his white ganger has arrived.

Back-stoping and resuing are probably the best methods of preventing disastrous falls of ground, but these methods are only suitable with ideal conditions. The former method is advantageous with highly inclined stopes, but has few merits in flatter workings, whilst the latter is only suitable for hand labour conditions, and since hand-stoping, owing to scarcity of native labour is likely to become a "lost art," the practice of resuing will probably accompany it to the limbo of forgotten things. In 1910 falls of ground caused the death of 265 persons and serious injury to 318. The mine hospitals treated 15,600 patients for minor injuries due to this cause, whose stay in hospital averaged four days, and £27,069 was paid as compensation for these accidents.

In this class of accident a steady increase is taking place every year, and it will be seen from Table IV. that our rate has risen from 17·58 in 1903 to 24·39 in 1910, an increase of 6·71 in seven years:

TABLE IV.

Principal causes with percentage of casualties.

	Explosives.	Falls of Ground.	Falling Material.	Trucks and Tramming.	Machinery.	Other Causes.
1903	19·20	17·58	11·01	11·68	12·46	28·07
1904	15·84	17·66	10·59	11 94	14·03	9·05
1905	36·82	30·36	10 92	10 86	10·35	20·74
1996	21·88	22·66	10 96	7·57	6·12	30 68
1907	25·42	22·21	9·99	10·98	5·09	26·31
1908	23·52	23·27	10·87	10·67	5·84	25·82
1909	17·63	22·57	14·93	12·35	6·52	26·00
1910	16·96	24·39	12·06	9·94	5·52	31·13
*1911	17·39	23·83	14·62	15·83	5·30	23·03

* Six months only.

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* Six months only.

† "Mine Subsidence" by A. Richardson. This *Journal*, Vol. VII., March 1907, pp. 279-289.

This advance is not due to abnormal conditions as will be seen from the above Table, but is due to some factor that year by year tends to increase while every other class of accident moves in the opposite direction and shows an appreciable diminution. With increasing depth it follows naturally that roof pressure will increase even though the angle of dip remains unchanged. Most of our deep mines, however show a serious flattening of the ore body, and, as this becomes more horizontal with increased depth greater pressure will be thrown upon the hanging walls and pillars, causing falls of ground to become more numerous. The detonation of heavy charges of explosives due to the increased use of rock-drills subjects the roof walls to great shock and vibration and greatly increases the number of subsidences and falls underground. Accidents due to tension blasts or "air blasts" are also becoming more numerous as we advance in depth. This latter class is difficult to account for and more difficult to prevent, but is probably due to some distortion of strata that is taking place in the Witwatersrand beds. For the prevention of accidents from falls of ground, the author advises the appointment of roof inspectors and the systematic timbering of stopes.

Roof Inspection.—In some of the Transvaal gold mines an official has been appointed whose sole duty is to prevent accidents from falls of roof and sides. This person visits the working places as often as possible and submits a daily report to the underground manager, giving a detailed account of the condition of the roof and sides in the places visited. This report, coupled with the reports from the shift bosses, enables the underground manager to keep in touch with the varying roof conditions all over the mine, and measures of precaution can be taken before the roof pressures get beyond control. The Mines Regulation Commission 1909, regards this official with scanty favour in spite of the favourable reports of the Inspector of Mines, believing that it will lead to a division of responsibility and increase rather than lessen the accident rate. The general tendency in most mining districts is to increase the number of inspectors both Government and local, for the need of more effective mine inspection is being gradually realised. England is creating a new body of State mine inspectors which will probably be largely made up from the ranks of practical workers underground, a procedure that has called forth some criticism. America is earnestly discussing the appointment of a "Mining Patrol" whose sole duty shall be the prevention of accidents underground. We fear that an increase of State Inspectors however large can only touch the fringe of the accident question, and it is to mining officials and local hanging wall inspectors who alone are thoroughly acquainted with the local conditions, that we must look to for a reduction in our accident rate.

Restraining the over-confident and instructing the ignorant is not always a pleasant task, but it is one that must be faced before our accident rate will be reduced to reasonable proportions

The following is a specimen leaf from the hanging wall inspector's report book of a central Rand gold mine :

The duty of a hanging wall inspector is to test the roof in all working places, paying particular attention to faults, slips, dikes, greasy or steatitic partings, fracture planes, cleats and rolls in the hanging walls. For testing purposes a small hammer is usually carried. To ascertain

Hanging Wall Inspection Report Book.

B^4 Main Reef Leader	Satisfactory.
C^5 ,, ,, ,,	More timber needed along the fault. Pointed this out to hand-stoper.
J^6 ,, ,, ,,	Miner here not taking down loose ground. Roof very rugged.
K^7 ,, ,, ,,	In excellent condition.
J^{10} ,, ,, ,,	Good.
D^8 South Reef	Had to stop the top machine owing to its being rigged up under bad hanging that vibration of machine was likely to bring down. Reported this to the Section Shift boss.
M^9 ,, ,,	Roof very bad, will probably get worse as stope is worked up to the dike. More timber wanted here at once.
K^{11} ,, ,,	Secure.
^{12} ,, East Reef Drive	Sets are needed between E and F. Pillars are crushing badly.

(Sgd.) John Smith,
June 6, 1909.
Mine Overseer's initials.
J. R. S.

if the rock is safe the inspector taps it with the hammer: the hollower the sound, the more unsafe is the roof. By placing one hand against it whilst striking with the other a slight vibration is often felt. This is a good practice, but not to be relied on implicitly, being useless where the detached slab is over 3 ft. thick. When unsound roof is found in any stope the working miner is warned of its presence and measures of prevention such as timbering or packing are at once taken. When the unsound hanging is small and easily detached it is taken down with a crowbar, but if found more difficult is blasted by the miner going off shift, or the roof natives are sent in, these latter being native hand drillers working under the supervision of a sectional shift boss, whose sole duty is the drilling of holes in bad ground. In stopes that are so highly inclined that hanging wall and foot-wall are equally dangerous, the hanging wall inspector must examine both.

Systematic Timbering.—At present there are two methods of stope timbering in the Transvaal mines. In one no timber is used except under isolated slabs or areas that are obviously dangerous. In the other, while dangerous ground is closely timbered additional supports are inserted at fairly regular intervals under hanging that apparently needs no support. This latter is a near approach to what is known as systematic timbering. This method was first employed at the Courrières Collieries in France where owing to the peculiar formation of the coal seams, ordinary methods of timbering were found to be insufficient to prevent accidents from falls of roof and sides. Experience there showed that a large proportion of accidents occurred from falls of roof in which no slip or fault could previously be detected, and this danger could only be met by timbering being done at regular intervals regardless of varying roof conditions. After a year's trial of the new method it was found that the accident rate from falls had decreased 87·5%.* In 1901 a Commission of Mining Inspectors was appointed by the English Home Office to visit these collieries and report on the system with a view to its adoption in the English coal mines. This report being favourable, systematic timbering was embodied in the Coal Mines' Regulation Act and became law in the United Kingdom. This law though severely criticised when promulgated is universally recognised as a step in the right direction and the new regulations dealing with the system, which have passed through Grand Committee this year and will come into force on January, 1912, are as follows:—

"Clause 51. The manager shall by notice specify the manner in which the supports are to be set and advanced and the intervals to be observed on roadways and at the face.

(*a*) Between each row of props.

(*b*) Between adjacent props of the same row.

(*c*) Between the front row of props and the face.

If the Inspector of the division considers that the system of supporting roof is unsatisfactory by reason of the distances fixed or any of them being excessive or otherwise he may require the manager to fix some lesser distance or otherwise modify the system."

The author contends that this system or a modification of it is desirable in the stopes of our gold mines. It is not intended to apply this to drives and shafts—where owing to the hardness of the rock and the narrowness of the workings, falls seldom occur. Mining returns show us that 95% of the accidents due to falls occur in stopes so that its application to other workings is unnecessary. As in the English law, these rules should preferably be framed by persons acquainted with the *local* conditions and should not be dictated by the Mines Department, for obviously rules suitable for the West Rand would be found not adaptable to Central Rand conditions and *vice versa.* These should be made by the mine manager and the underground manager to the best of their ability in conformity with experience and should be liable to alteration at any time in accordance with altered conditions or evidence of insufficiency. On the occurrence of an accident, measures of precaution which would have prevented it should be incorporated into the rules, and the frequent repetition of the same type of accident would thus be avoided. All methods of timbering must be based on the methods of working, the nature of roof, slips, faults, and foot-wall; these differ considerably even in the same mine and at times in the same stope. The maximum distances between supports can only be presented in these rules, and it will frequently be necessary to set timber over and above the requirements of the rules. The erection of these additional supports must be left to those working in the stope who alone are aware of the momentary changes in the roof conditions. The chief objection to this method will probably arise from the thought that it will increase our mining costs. When introduced in England, many mining engineers opposed it on that ground, saying that the mines could not bear the additional expense involved by the new rules, and many of the mines working on a narrow margin of profit would be compelled to shut down, but experience has shown that these fears were groundless, for a very high authority, Mr. Isaac Hodges, in his presidental address before the Midland Institute of Mining,

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*See *Transactions, Institution of Mining Engineers*, Vol. 20, p. 164

Civil and Mechanical Engineers, says that systematic timbering has not increased the working costs in the coal mines of Great Britain.[*]

In 1910 the Transvaal bill for round mining poles and lagging amounted to £162,895 or less than 1½d. per ton over the mineral out-put ; this coupled with the expense of setting would probably amount to 4d. per ton. If systematic timbering were introduced, probably our timbering costs would rise to 6d. per ton. Against this rise of 2d. per ton we would have a larger labour force, a greater freedom from accidents, less compensation costs, the recovery of large quantities of broken ore that is now buried under collapsed stopes, less cost in the re-winning of stope faces, and less waste rock sent to the surface. The amount saved on these items would leave a handsome profit over and above the cost of systematic timbering. Even if systematic timbering raised our mining costs considerably, humanity demands that this or some other method must be tried, and this waste of life must not continue. In the crusade to reduce accidents we can count upon the support of the leaders of the mining industry for one of them has assured us "that the question of health and of life has always come and will always come before considerations of profit and gain"[†].

The Chairman : I am quite safe in saying that this very able and valuable paper deserves our best thanks. We always give a vote of thanks to the author of a paper, and I think it is always deserved, but I think that this paper is especially valuable. We have had a great agitation during the last few years on the question of miners' phthisis, but this question of the mortality and injury due to accidents has not received the attention the other question has. I do not know what a comparison would show, whether there are more deaths due to accidents than as the result of miners' phthisis, but these figures which have been presented to us to-night, and with which most of us are more or less familiar, have been brought to our minds afresh. I think Mr. Chilton has put his arguments moderately and at the same time with the force which a subject of this kind, brought before a scientific and technical society, demands. I feel ill-fitted to discuss this paper, but I trust that our mining members, whom it immediately concerns, and whose lives it concerns to some extent, will take this matter up thoroughly and give the Society the benefit of their views on the subject.

I tender the thanks of the Society to Mr. Chilton for his most valuable contribution.

Mr. Tom Johnson (*Member of Council*) : Mr. Chilton has given us the figures for the death rate in England and the Transvaal. I should like to give a few additional figures for other countries, as follows :—

	1907	1908
England ...	1·46	1·46
Transvaal ...	4·71	4·18
Germany ...	2·38	2·55
U.S.A. ...	4·67	3·40
Austria-Hungary	1·27	1·13
France ...	1·14	·95
Belgium ...	1·04	1·07
Russia ...	2·20	—

The actual death rate per million tons in 1907 was 4·5, in 1909 5·25 and 1910 6·3. The high figures in 1909/10 are due to the series of explosions which occurred in those years. Dr. Maynard seems to say that there is no benefit in boys staying on the mines for a length of time. I think there is a benefit, and I think the accident figures for outcrops help to explain it, because, as a rule, the boys will stay on an outcrop mine longer than they will on a deep mine. They do not like riding in the cages ; they would sooner walk, and they therefore stay longer on the outcrops, thus becoming more used to the mine, the mining instinct also increases. The same thing also applies to ourselves. The longer we stay, the better able we are to look after ourselves. The author thinks that the modern tendency to shorten hours will have the effect of increasing the death rate, but Mr. Pickering, the Inspector of Mines for Yorks, does not think so. The author explains that since the introduction of the eight hour system in England the death rate has gone up 30%, but that, of course, is due to the explosions to which I have referred. There was the Maypole disaster, 1908, West Stanley, 1909 (68 deaths), the Whitehaven (136), and the Pretoria Pit, Bolton (344), in 1910. I think that that is why the death rate went up those years.

With reference to Mr. Chilton's remarks about the School of Mines doing good work, I suppose it does, but if really good men are wanted, they must be men who have also worked in mines for some time, and thus had good practical experience. Men who have learnt both sides of their business are better than any School of Mines men who have not been long underground.

The author has stated that 22·57% of the accidents in this country are due to falls of ground. Well, that is low in comparison with the other kinds of accidents. In the Old Country the deaths due to falls of roof and sides are between 40% and 50% of the total number of accidents.

As regards the author's remarks about the boys going into the working places before the ganger, this is becoming a pretty common thing on the

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[*] *Transactions of the Institution of Mining Engineers.* Vol. XL., 1910-1911, pp. 168-175.

[†] R. W Schumacher, *Transvaal Chamber of Mines*, Aug. 18, 1910.

Rand, but of course it should not be so. As the author says, it is conducive to many accidents.

In Table IV. there are a few misprints. The 1905 percentage of accidents due to explosions, instead of being 36·82 should be 26·82, and falls of ground instead of being 30·36 should be 20·36.

There are several other reasons for accidents than these which have been given in the paper. It is getting to be thought that the more boys a man can be made to look after, the cheaper it is. I hold myself that that is a fallacy. And it is the same with the three machine work. When a man's attention is spread over a large area he cannot watch so closely, and also I do not think the work is done as efficiently or so cheaply per ton with three machines as with two, or with one for that matter. The machine men will stick holes in close to the roof, and that is bound to loosen the roof and give it more chance to break away. Again, a machine man is often in a hurry to start drilling, and does not wait to dress down the face, so that when starting to drill top holes, pieces of rock are knocked off, and the boy working below is struck. I think, too, that there has been an increase in the accidents, due to the increase in the total number of machines, and you will find that the number of machines used is going up steadily year by year. Another thing is that working faces are not moved fast enough. If these were moved faster, the roof would not have time to cut as it does at present. It might be referred to as running away from danger. This is not theory, but a practice which has been carried out year after year in England in the coal mines, and in fact in every other country.

The author says that as the dip becomes flatter, there is more pressure. I hardly see how more pressure would be obtained through the change of dip unless it is coming from a very steep dip to a flat dip, because in that case there would be a break, and the roof would be weakened by the bending which had taken place. The strain is there already before taking out the ground. That would not apply so much to a gradually decreasing dip. So long as the cohesion has not been destroyed, depth is the determining factor.

As regards the air blast, I think that there is no such thing as an air blast causing an accident. It is rather an effect. If the roof is moving, it has either got to break itself, break the pillars, or break the floor. Whichever is the weakest has got to go, and it is all from the same thing—pressure of the roof—and the air blast comes when that break takes place.

As regards roof inspectors, I do not know whether there is much good in them or not. I am inclined to think that if shift bosses who understood their work were employed, their districts were made smaller and their number increased so that that they could look after their districts efficiently, the effect would be just as good as having roof inspectors. If a roof inspector is appointed, he is expected to travel over a great amount of ground, which the shift bosses and the mine captain have already travelled over. If these latter officials were to look after the roof—as they could do were my suggestions carried out—there would be no need for special roof inspectors. I think that the shift bosses should be able to look after their districts and keep them safe.

As to Inspectors of Mines, I do not think that there are enough of them in the Transvaal. The inspectors at present have no time to visit a mine except when they go to see about an accident, and then they have no time to inspect anything except the site of the accident. Inspectors should be appointed to do nothing else but inspect the mines in rotation, and they should have nothing to do with accidents at all. And they must be practical men. It would be better if they were men who had been mine managers. They might see things in the best mines which were against the regulations, but if it could be seen that the manager was trying to carry out the regulations to the best of his ability, it would not do to stick to the strict letter of the law but rather to the spirit of the law. Then again, I think that the Inspectors of Mines should make more examples when they find things wrong. There is plenty of room for it. Punishment is about as good as anything for stopping accidents.

I do not think, in regard to the systematic timbering as carried out in the Old Country, that it has cost anything more than before, because if the timber is set properly there is a better chance of recovering it again.

There have been many accidents due to the use of explosives. I have investigated many cases of fatal accident, and have generally found that the men have been doing something which they ought not to have been doing. But these facts do not come out at the time. They are only heard of afterwards. If facts were brought out more openly at the time of the accidents, there would be less chance of the same thing happening again.

Then in regard to discipline, orders when given, should be carried out. In the Old Country, if a man is seen breaking rules he is summoned by the mine people. This should be done here also, when men or bosses do not carry out the rules and regulations that are made in the interests of safety.

Mr. W. S. V. Price *(Member)*: I notice that the title of Mr. Chilton's paper makes special reference to "falls of ground"; so I will confine myself principally to that branch of the subject, because there are far more men injured by falls of ground than by any other kind of accident. From my own experience of falls of ground (which have occurred at mines on which I have worked) I can say that they have mostly been caused by sheer carelessness, ignorance, or callousness on the part of those responsible for supervising the work. In one instance, a converged fault was supported by bits of lagging-poles, the biggest stick under it being not more than 7 in. in diameter. Nevertheless, when the mine overseer and shift boss were spoken to about it, and were told that the money spent on block-holing waste and sending it up to the surface along with reef would more than pay for building "pig-sties" which would effectually support the ground, they sneered at the idea of learning anything from a common miner. The ground fell in, in course of time, and a big section of good reef was lost; later on the stopes just over it also caved in and more payable areas were lost, or cost so much more to get at. Fortunately no one was hurt as it occurred between shifts.

Four years ago, I was giving evidence before the Mining Regulations Commission, and I was asked if I would hold a mine inspector criminally responsible in case of a fatal accident occurring after he had visited a mine. Of course, I held that it depended on the nature of the accident, the time that had since elapsed, and whether he had visited that particular section; and I quoted the case I have just mentioned, for the inspector had visited the mine less than a week before the ground fell in; and although he passed this place he had no eyes to see it and its totally inadequate supports. He was apparently so intent on finding some small dereliction of duty where one or two might be injured, that he could not see the place where 30 or 40 natives and whites were in danger. Mr. Johnson spoke of having more inspectors. We do not want that kind, but rather some like those that are employed in England from among the ranks of the working miners—practical men who know the kind of work that ought to be done.

The author mentioned carelessness, etc. as the cause of accidents; but is it not the class of men who are employed underground now that is largely responsible for the increasing number of accidents? On some mines an interpreter is needed underground if one wants to be understood. I am not speaking of the natives, most of whom speak and understand some sort of Kafir lingo: but of the foreign whites, for unless one speaks Italian one cannot understand them or be understood by them. I claim that these fellows are a real danger; and I cannot understand how they came to be allowed to have blasting-certificates, since they certainly could not answer the simplest question in English or Dutch, and cannot be expected to know the least thing about the Mining Regulations. It would be amusing to listen to one of these trying to hold a conversation with an Englishman, the pair of them using Kafir as the medium of conversation. It would be amusing, I say, if it were not for the issues at stake. I know of one case where a man was injured in a blasting accident through the other man not being able to make him understand, although he said, in my hearing, that he quite understood. It would almost seem nowadays as if anyone, except a miner, is good enough to be a shift boss; we see samplers, timbermen, surveyors, pipe-fitters, etc. in that position—where none but thoroughly practical men should be. In many cases where ground has fallen in stopes it would not have happened if the hanging had not been broken into by careless miners, overlooked by ignorant or incompetent shift-bosses. The same holds true in shaft sinking, either vertical or inclined; we find holes drilled and blasted that should never be allowed. In one case which I remember, a few years ago, of a vertical shaft caving, it was almost entirely due to the way the side holes had been drilled, looking out so much that the ground was shaken many feet away.

In fact, all things considered, we ought to congratulate ourselves on our *good luck* in getting so few accidents in spite of our system, or lack of system. Nevertheless, safety measures pay for themselves in most cases; for it is cheaper to stop a shaft from caving in than to sink another or to re-excavate one that has caved in.

PRACTICAL APPLICATIONS OF THE SPECIFIC GRAVITY OR MOISTURE FLASK

By H. Stadler (Associate).

Mr. W. J. Sharwood's exhaustive treatise on "Measurement of Pulp and Tailing"* is presumably familiar to all millmen who have to deal practically with the handling of pulp. The present note does not pretend to add much that is really novel, but I trust that some practical conclusions drawn from the above and other papers on this subject† will be welcome to those whose daily duties leave very little spare time for a thorough study of technical literature. The principal object of this note is to show the great practical possibilities of the specific gravity flask, which in future should prove to be one of the most useful and indispensable instruments for efficient and reliable control of the conditions of mill-pulp.

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* *Mining Magazine.*

† See F. B. Hyder's "Specific Gravity Estimation of Pulp." Colorado Scientific Society, Vol. ix., p. 417; and W. A. Caldecott, this *Journal*, Vol. ii., p. 375.

In practice it has been found that flasks having a large mouth ground true, the ground edge to be used as the mark of capacity, are more convenient for taking fair samples than the usual glasses with a mark below the mouth. In consequence of sand adhering to the glass above the mark, the latter are more difficult to fill accurately.

I. Determination of Density (specific gravity) of Solids.

If the specific gravity of the solid ore under treatment be not taken at an accepted average value it may be determined by any well-known method, for instance:—

If the material is dry, a known weight (in grams) of the material under examination is placed in the flask, moistened with water and shaken to remove air-bubbles and water added up to the mark (in the suggested flask up to the top), and the total weight of the contents is then taken. The density is obtained by the use of formula 1. Wet material, such as mill pulp is preferably dried and weighed after the determination of the specific gravity of the pulp.

II. Determination of Specific Gravity of Pulp.

Almost all factors determining the nature of pulp from the millman's point of view, are governed by the specific gravity of the pulp. If the metric system of weights be used, the specific gravity is at once obtained by dividing the nett weight of the pulp by the volume occupied, or when a 1,000 cc. flask is used by pointing off the last three figures of the nett weight of the pulp (in grams).

The specific gravity of slime pulp, in which all solids are freely suspended, may be measured by a hydrometer.

III. Determination of Percentage (by weight) of Dry Solid and Water.

Although the value of the specific gravity is representative of the consistency of pulp, the percentage of dry solid or moisture conveys a more definite impression to the average mind.

The usual method of determining the percentages of solids and moisture, by drying and weighing, is very troublesome, and also, as done in practice, unreliable and inaccurate. Hence methods based on specific gravity, (formula 3), are vastly quicker and more convenient.

Any alteration in the composition and in the rate of flow of pulp shows itself by a marked change in the percentage of moisture and therefore gives a very reliable, sensitive, and effective control in the uniform distribution of the pulp and the regular working of classifiers. For instance, on a mine where a set of 8 similar classifiers was fed with the same battery pulp, the moisture in the underflow of the different classifiers varied, at the same moment, from 35 to 75%, thus showing that the distribution of the pulp over the classifiers was very irregular.

It is to be hoped that with such a convenient means at hand, millmen will pay more attention to the control of individual units, as the total average always gives more or less passable figures, which are no guide to detecting faulty work.

IV. Tonnage Measurements of Total Pulps (Tp) and of Dry Solid in Pulps (Ts).

Tonnage measurements of flowing quantities are generally made by running pulp for a measured time into a vessel of sufficient capacity to give fairly accurate results. Scales for weighing the large quantities collected are not usually available, and the drawing off of the water, and the drying and weighing of the solid portion in such quantities still less practical.

All these inconveniences are avoided by reckoning the tonnage from the volume occupied by the total pulp (in cubic feet) and its specific gravity, by formula 4 for the total pulp, and by formula 5 for the dry solid portion only.

The simplest manner of carrying out these measurements is to run the pulp into a box of known capacity, large enough to take at least one minute's run of the stream. The exact time required for filling is taken by a stop-watch, the point of overflow being easily and exactly determinable. The specific gravity is determined from separate representative samples taken simultaneously, and the calculated weight of pulp or dry slime passing during the time (*t* in seconds) are then used to give the hourly or daily quantities.

If, in all measurements, boxes of equal capacity are used, for instance, 16 cub. ft. (the volume of half a ton of water) all the constant factors may be condensed into one. For a box of the above capacity the formulae for Rand ore are:—

Tons of total pulp in 24 hours: $(\mathrm{Tp}) = \dfrac{43{,}200p}{t}$

Tons of dry solid only $(\mathrm{Ts}) = \dfrac{(p-1)\,68609}{t}$

Since the capacities of launders, vats, pumps, etc., as well as the rates of overflow of classifiers are determined by the *volume* of the pulp, the "fluid ton" of 32 cub. ft. (equal to the volume of 2,000 lb. water, on the assumption that one cub. foot of water weighs 62·5 lb.) is frequently used as a unit in mining work. The tonnage of pulps of known specific gravities are readily converted into fluid tons by formula 7 for the total pulp and by formula 9 for the dry solid portion.

The same method is also applicable on a large scale to tonnage measurements of vats, etc., provided the consistency of the pulp is not changed by drawing off water or slime (as is the case with sand-collecting and slime settling vats), or by adding cyanide solution (as happens with slime treatment vats). For large tonnage measurements the practical utility of this method is therefore confined to agitation or circulation vats, and in all other cases, for lack of better methods, we must resort to the method commonly practised, inaccurate as it is, of estimating the dry solid by the established dry weight of the settlement volume. This varies considerably with the nature and the fineness of the ore, and should therefore be checked frequently and corrected if necessary by actual test. For such tests the method I describe is very suitable as no drawing off of water nor drying is necessary.

At the Knight Central G.M. Co. the daily tonnage measurements of the mill pulp for several years past have been regularly taken by the gravity method, which, after careful investigation, was found to give most reliable results. In addition there is the advantage that it can be carried out without waiting for settlement of the slime. After filling the first transfer tank with solution and after circulation (the whole operation taking about four hours) two samples are taken, one at the delivery pipe, representing the pulp at the bottom of the tank, and one from the upper part, to get a representative average for the whole content.

V. *Relations of Flowing Quantities in Classifiers.*—The relation of flowing quantities in classifiers may be calculated from the percentage of any of their components measured at the inflow, overflow, and underflow. No component can so quickly and reliably be measured as the percentage of dry solid taken with the specific gravity flask. From the percentages of dry solid so obtained, the tonnage of the total pulp underflow (as a percentage of the total pulp inflow) is calculated by formula 10 and the tonnage of dry solid underflow (as a percentage of dry solid inflow) by formula 11. In neither case is it first necessary to calculate the percentage of solid, since the flowing quantities of the underflow can directly be determined from the values of the specific gravity, by formula 12 to get the percentage of the total pulp, and by 13 to get that of the dry solid portion only.

Accuracy.—The accuracy obtainable by the specific gravity method is only limited in practice by the accuracy with which the means at our disposal determine the correct values of (1) the volumetric weight of the pulp (w), (2) the density of the solid (d), and (3) the specific gravity of the liquid (δ).

(1) *Accuracy of Volumetric Weight of Pulp.*—When using a 1000 cc. flask an error in content or in weighing of even 10 cc. or grams respectively, affects the specific gravity (p) of the pulp in the third decimal only. For a 1 : 1 pulp the error is 0·007% which increases with the dilution of the pulp.

A variation in (p) of 0·01 in a 1 : 1 pulp (Rand ore) gives an absolute error of 0·74% in the calculation of the *dry solid or moisture percentages*, corresponding to 1·48% of the true value. With the dilution of the pulp the error increases progressively. The accuracy of the tonnage measurement of the *total pulp* varies as that of (p), whilst the tonnage of the *dry portion* shows an error of 2·17% which increases progressively with the dilution of the pulp. If we allow such rough work in actual mining practice the annexed curve gives us the percentage of moisture or solid within ½ to 1%.

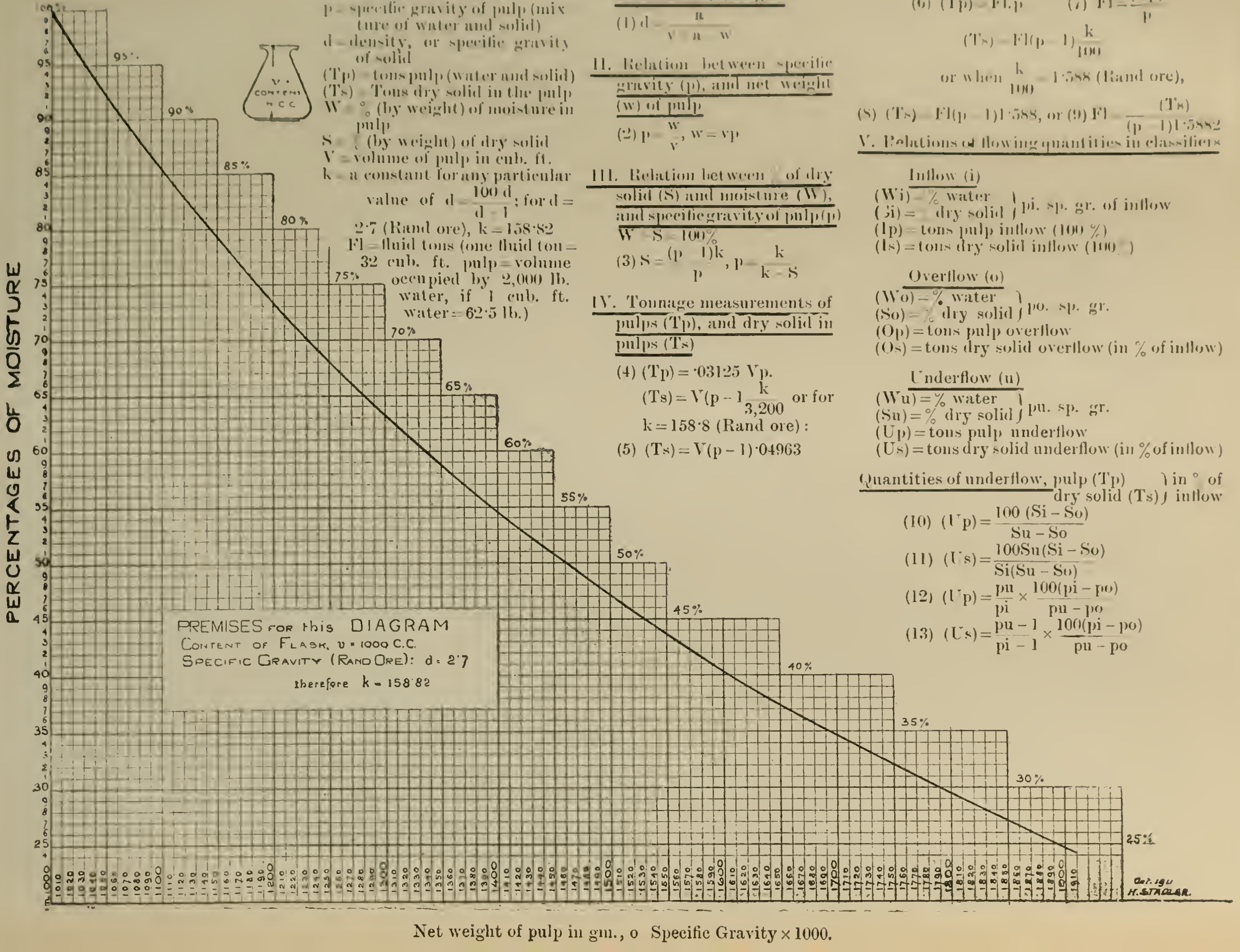
(2) *Accuracy of Density of Solid.*—It seems that the average density of Rand ore is, at least, for individual mines, fairly steady. The constant (k) in the formula varies with the density as follows :—

Density:	2·4	2·5	2·6	2·7	2·8	2·9	3·0
K.	171·4	166·6	162·5	158·8	155·5	152·6	150·0

A variation from the generally accepted average density of 2·7 for Rand ore to 2·8 affects the accuracy of the *specific gravity* (p) in a 1 : 1 pulp by 0·96%, which error decreases slowly with the dilution of the pulp. The tonnage of the *total pulp* is affected to the same extent as (p), whilst the *dry solid* portion shows a constant error of 2·8% independently of the consistency of the pulp. For the *percentage of dry solid and moisture* the error is 2·06%, which increases with the dilution of the pulp. When much concentration of the pyritic portion of the ore takes place the actual specific gravity must be ascertained in each case, as an increase from 5 to 10% of pyrite causes the specific gravity of the solid to rise from 2·7 to about 2·8%.

(3) *Accuracy of Specific Gravity of Liquid.*—All the formula are based on the specific gravity of pure water, but they are applicable to ordinary mixtures of cyanide solutions with enough accuracy for practical purposes, except in case of excessively thin pulps, such as turbid water, overflow of classifying vats, etc. For exact scientific work or for much heavier liquids (sea water = 1·027) the true specific gravity (δ), corrected for temperatures, is taken into consideration in the following formulæ :—

PRACTICAL APPLICATION OF SPECIFIC GRAVITY OR MOISTURE FLASK.



w = net weight of (v) cc. pulp in gm.
v = content of flask in cc.
p = specific gravity of pulp (mixture of water and solid)
d = density, or specific gravity of solid
(Tp) = tons pulp (water and solid)
(Ts) = Tons dry solid in the pulp
W = % (by weight) of moisture in pulp
S = % (by weight) of dry solid
V = volume of pulp in cub. ft.
k = a constant for any particular value of $d = \frac{100\,d}{d-1}$; for d = 2·7 (Rand ore), k = 158·82
Fl = fluid tons (one fluid ton = 32 cub. ft. pulp = volume occupied by 2,000 lb. water, if 1 cub. ft. water = 62·5 lb.)

I. Determination of density (d) of material (a) in gm.

(1) $d = \frac{a}{v + a - w}$

II. Relation between specific gravity (p), and net weight (w) of pulp

(2) $p = \frac{w}{v}$, $w = vp$

III. Relation between % of dry solid (S) and moisture (W), and specific gravity of pulp (p)

$W + S = 100\%$

(3) $S = \frac{(p-1)k}{p}$, $p = \frac{k}{k-S}$

IV. Tonnage measurements of pulps (Tp), and dry solid in pulps (Ts)

(4) $(Tp) = {\cdot}03125\ Vp.$

$(Ts) = V(p-1)\frac{k}{3{,}200}$ or for k = 158·8 (Rand ore):

(5) $(Ts) = V(p-1){\cdot}04963$

Tonnage and fluid tons

(6) $(Tp) = Fl.p$ (7) $Fl = \frac{(Tp)}{p}$

$(Ts) = Fl(p-1)\frac{k}{100}$

or when $\frac{k}{100} = 1{\cdot}588$ (Rand ore),

(8) $(Ts) = Fl(p-1)1{\cdot}588$, or (9) $Fl = \frac{(Ts)}{(p-1)1{\cdot}5882}$

V. Relations of flowing quantities in classifiers

Inflow (i)

(Wi) = % water, (Si) = % dry solid } pi. sp. gr. of inflow
(Ip) = tons pulp inflow (100 %)
(Is) = tons dry solid inflow (100 %)

Overflow (o)

(Wo) = % water, (So) = % dry solid } po. sp. gr.
(Op) = tons pulp overflow
(Os) = tons dry solid overflow (in % of inflow)

Underflow (u)

(Wu) = % water, (Su) = % dry solid } pu. sp. gr.
(Up) = tons pulp underflow
(Us) = tons dry solid underflow (in % of inflow)

Quantities of underflow, pulp (Tp), dry solid (Ts) } in % of inflow

(10) $(Up) = \frac{100\,(Si - So)}{Su - So}$

(11) $(Us) = \frac{100Su(Si - So)}{Si(Su - So)}$

(12) $(Up) = \frac{pu}{pi} \times \frac{100(pi - po)}{pu - po}$

(13) $(Us) = \frac{pu - 1}{pi - 1} \times \frac{100(pi - po)}{pu - po}$

Specific gravity of pulp $= \frac{100\,\delta p}{100d - S(d-\delta)}$, and

Percentage of dry solid $S = \frac{100\,d(p-\delta)}{p(d-\delta)}$

In stating that the extent of the error introduced by ignoring the higher specific gravity of cyanide solutions, is not generally appreciated, Mr. Hyder was misled by erroneous information (accepted by him with the reservation "assuming this to be correct,") as to the average density of ordinary cyanide solutions. The strength does not generally exceed 0·18% for sand, 0·025% for slime, and 0·3% for black sand. I am indebted to Mr. M. T. Murray, lecturer at the S. A. School of Mines, for the exact determinations of the specific gravity of an ordinary cyanide solution, assaying 0·126 KCy and 0·01 alkalinity. He obtained a value of 1·0021, and from the data given above it will be seen that the small variation from the specific gravity of pure water is negligible.

In conclusion I wish to record my appreciation of the courtesy of the Mines Trials Committee, in allowing me, whilst in their employ, to publish these notes on my own responsibility.

The Chairman: I have much pleasure in extending the thanks of the Society to Mr. Stadler for his paper. It is to be hoped that the results he has obtained will, as he surmises, be of some use in the practical work on the mines.

Mr. F. J. Pooler (*Associate*): I should like to ask the author a few questions on points that occur in this paper. From what I can see, the term "density" as applied to solids and "specific gravity" referring to liquids are used in the same sense. It is generally understood that the density of a substance is the *mass* of unit volume, and specific gravity is, theoretically, the *weight* of unit volume, or as generally used, the ratio between the weight of any volume of the body and the weight of an equal volume of water. When using the metric system, the weight of a cubic centimetre of any substance in grammes is equal to the specific gravity. No such simple relation holds, of course, if British units are used. It seems to me that the question should be definitely settled by the Standardisation Committee for metallurgical and mining terms. To avoid confusion in the future, will the author kindly explain why he has chosen density in speaking of solids and specific gravity for liquids, slime, pulp, etc.?

The method suggested by the author, of using a wide mouth flask, ground true, for liquids is obviously the only quick way for dealing with slime or pulp, but is, I think, not so good as that of filling a flask with water to a mark on the neck, when finding the specific gravity of a dry solid, such as sand. The sand can all be washed down from the neck of the flask with the water that is added in the experiment, and if this water is added till its meniscus reaches the mark which acts as a tangent to the lower part of the curve, two advantages are gained: 1, the results can be read more accurately; 2, there is less chance of spilling liquid when the flask is carried to the balance for weighing. In using a flask of water, it is not at all easy to allow for the drop on the top being of the same size each time. In the table given below, the numbers in column A are weights obtained by this method, the drop at the top being removed by a knife blade and the flask wiped dry outside, while column B shows results when filling the flask to a mark on the neck, the inside of which was dried by a hollow cylinder of blotting paper before weighing.

	A	B
	Gram.	Gram.
	100·545	92·920
	100·565	92·915
	100·520	92·900
	100·610	92·895
	100·570	92·900
	100·510	92·900
Average	100·553	92·905

The maximum variation in the first case is 0·100 gm., and in the second case only 0·015 gm. The percentage error is small in both cases, and in mine work would probably not matter, but using wide mouth large flasks as recommended by the author, the error would be larger, as the drop at the top of the flask increases with the diameter of the mouth, and the difficulty of removing it also becomes greater. Generally, it will be agreed, I think, that neither of these methods is as accurate as the one where a perforated stopper is used, the bottle being filled to the top of the perforation; and this method requires little if any longer time than either of the others.

When "the specific gravity of slime pulp in which all solids are freely suspended" is measured by a hydrometer, I should be glad to know if this has not to be done very quickly, while the liquid is well agitated. Also, does not the agitation increase the difficulty of getting accurate readings? It seems to me that otherwise, the solids settle so quickly that the readings will show a fairly rapid progressive fall. I tried to find the specific gravity of a pulp made by suspending 150 gm. of sand (passed through a sieve of 560 meshes to 1 sq. in.) in 500 gm. water and found that the value fell to 1, almost within a few minutes. If the author can give me an explanation of the reason why such a pulp gives fairly accurate values for specific gravity by this method I shall be grateful. From purely theoretical considerations it seems that while pure water gives an accurate result, a progressive inaccuracy will result as the pulp gets thicker, till the limit, when the sand is dry, is reached, when the results will be useless. In fact, theoretically, there seems to be no obvious reason why the hydrometer floats to different levels in pulps of various consistencies.

In apologising for these questions, I may say that as I am daily engaged in trying to train the future millmen and cyaniders of the Rand, these points will be extremely useful in practical work for our classes.

FUTURE ECONOMIES IN RAND PRODUCTION PLANTS.

(*Read at April Meeting, 1911.*)

By C. O. Schmitt, M.Inst.M.M (Member).

REPLY TO DISCUSSION.

Mr. C. O Schmitt (*Member*): The author is under a great obligation to those who have discussed the paper, although he cannot but regret the undue attention given to the introductory remarks, which, as Dr. Simon justly pointed out, were written not to discredit the future of the Rand, but merely intended to show the real necessity for further economy.

When writing the introduction, the writer was certainly under the impression that the statements made were common knowledge, and the storm in a tea-cup resulting caused him considerable surprise. It is to be hoped that the information volunteered by the critics has allayed public anxiety regarding the future of the Rand, and while nothing can be gained by dissecting the question further, it is only right that the author should briefly traverse some of the statements made, and justify the position originally taken up.

In the rush and excitement to refute an imaginary attack on the gold mining industry of the Rand, the points brought forward in the paper proper have been almost neglected. This is unfortunate, as no amount of talking and writing will increase the quantity of gold actually in the ground, but the amount of gold susceptible of exploitation, and ultimately available for distribution, can be sensibly increased by improved methods. As Mr. H. S. Denny said: "If only the cost factor could have been dealt with in the earlier stages of the history of most mining camps, what wonderfully enhanced profits would have resulted." No doubt, history will repeat itself, and a similar remark will one day be made with reference to the present time.

It will be convenient to reply to the discussion under the following headings:—

(*a*) The value of ore in depth.
(*b*) Mining costs, and
(*c*) Economies in reduction plants.

The Value of Ore in Depth.—The author, in presenting Table I. and Fig. 1 of the paper, qualified the same by saying that they represented his own interpretation only of the results obtained from the Witwatersrand. The figures given may overrate the value of the ore near the surface, or may underrate the value at, say, 5,000 ft. vertical depth, but the tendency for the value of the ore to decrease in depth remains. Mr. Marriott denies this, although he brought no documentary evidence to support his remark, merely saying: "As far as reef values were concerned, there was no substantial evidence to-day, even for the most confirmed pessimist, to prove a general deterioration." Mr. Marriott also pointed to the well-known fact that during the earlier days of the Rand only the highest grade ore was mined, while large quantities of low grade ore were left standing. In reply to this, it could be pointed out that, for a given area, a much greater tonnage of high grade ore has been obtained from near the surface than the tonnage of low grade ore now obtained from greater depths, and this still holds good after making allowance for the greater angle of dip near the surface. Large quantities of the low grade ore originally left behind in the shallower mines now serve to swell the total tonnage milled at the present reduced grade, and to temporarily reduce working costs, as will be shown later on.

Mr. Truscott points to the low value of the ore milled by some of the outcrop mines, particularly on the West Rand, to prove that depth alone does not account for low values. The author's reply to this is that his figures, which are only approximate, are intended to state the average of the whole Rand, and do not represent the high values of the Central Rand, nor the low values of the West Rand.

Mr. H. S. Denny, while declining to commit himself to a definite opinion regarding the decrease in grade with depth, showed by means of an example how the grade of a mine apparently dropped, although the value of the ore *in situ* continued the same. Yet in his closing sentences Mr. Denny states that, to assume the value for the fourth row to be the same as for the first row, would give too favourable a basis; in other words, the values are less in depth than near the surface.

Nothing short of a detailed investigation of all the material available will give definite information on the point: hence, for the moment, there remains nothing but to indicate the tendency and await the solution at a later date. To wind up this portion of the reply, the writer wishes to cite two independent authorities, *i.e.*, Dr. F. H Hatch in the course of the "James Forrest" Lecture, 1911, stated: "I have not been able with the material at my disposal to come to any definite conclusion on this point, but I am inclined to the view that a gradual impoverishment in depth does exist." Dr. Hatch goes on to say that it would be possible to settle this really vital question from existing records and assay plans. Mr. F. Hellmann, in the course of his evidence before the Mining Industry Commission, on October 24th, 1907, said: "I think one of the first causes of the depression is the undesirable fact that the reefs have decreased in value in proportion to their depth, and that fact is pretty well known and realised."

Mining Costs.—The author's statement that mining costs will not be able to keep pace with the reduction in grade, and in fact, will move in the opposite direction, has been seriously questioned by a number of critics, notably Messrs. Marriott, Truscott, Hutton and Johnson.

Mr. Marriott was good enough to unwillingly indicate an item tending to increase costs which the author has not mentioned, in saying, "that it would be necessary to use greater discrimination as to what rock was chosen for the mill, and what was left behind." Now, we all know that if there is one thing which tends to raise mining costs, it is the necessity of having to search far and wide for payable ore, because not only does the actual mining become expensive but the expenditure on development is necessarily distributed over a small tonnage, thus raising the cost per ton inordinately.

Regarding the expected increase in the cost of labour, Mr. Marriott differs, and assumes that the abnormal price paid for skilled labour will be reduced and that native labour will be largely displaced by mechanical methods. The last part is also subscribed to by Messrs. Hutton and Johnson. Dealing with native labour first, it should not be forgotten that the cost of machine drilling has only quite recently begun to approach the cost of hand drilling, and while the writer agrees that, ultimately, mechanical work will be cheaper than human labour, the day when this will be the case may still be a long way off. Meanwhile, with more time lost to reach the working faces, and less favourable conditions for efficient work, the cost of native labour is increasing with greater depth. That the pay of white labour per unit of work done must ultimately be reduced is obvious. Pending the arrival of this desirable period, the cost of living continues as high as ever; the contaminating influence of native labour prevails, and more time is lost in getting to work at the deeper working places. In order to maintain the efficiency which did, or at least should have prevailed in the shallower mines, ventilation must be resorted to at no small cost. That ventilation increases the efficiency of the labour force is obvious, but as at best it can only equal the efficiency in shallower mines, the author's point, that the cost of labour increases with depth, is undoubtedly established.

Mr. Hutton points out that a good deal of ventilation has already been done at considerable cost by means of the high pressure air service. This is well known, and only the want of reliable figures regarding the cost of the work prevented the author from qualifying his remarks accordingly. Whatever the cost of ventilation, it must increase with depth.

The cost of mine drainage is, as the author stated, not a large item, but unless the water disappears altogether, it cannot be less in the future than it has been in the past. Mr. Johnson's statement that the water will in future be caught at a lesser depth, and that it can be used for ventilation as also suggested by him, is more than the writer can reasonably be expected to believe.

It is to be hoped that Mr. Johnson is right when he says that the cost of breaking rock underground will be reduced in future due to the application of scientific principles to blasting operations. However, this does not dispose of Mr. Cullen's statement that the price of explosives will rise owing to a scarcity of glycerine.

With regard to the cost of hoisting, Mr. Truscott contradicts the author's statement, and calls in the aid of cheap electric power. Mr. Marriott is willing to admit an increased cost of ropes and power per unit hoisted, but says that as vertical shafts have reached their maximum depth, and as the cost of future inclines is to be charged to development, these inclines can (according to him) be run at no greater expense than in mines near the outcrop. The writer has so far not been aware that the mere transferring of an item of expenditure to another account will reduce the absolute cost. Mr. Hutton hopes to out-balance the increased cost due to greater depth by cheap purchased power and reduced maintenance costs, and Mr. Johnson expects to reduce hoisting costs by working the engines efficiently. Some lack of knowledge of this important subject is displayed in the criticisms offered by these gentlemen, and the writer thinks that only good can be done by throwing some further light upon the point.

For our purpose a comparison will be made between two mines, "A," working at a vertical depth of 2,000 ft., and "B," at a vertical depth of 5,000 ft.; the tonnage mined and hoisted through one shaft to be 2,000 tons per day of 16 hours; mine "A" to be equipped with either a vertical or an incline shaft to reach the depth of 2,000 ft.; mine "B" to be equipped with a vertical shaft about 3,000 ft. deep, and two successive inclines, each 3,000 ft. long. To hoist 2,000 tons in 16 hours, and to handle the men and supplies, each shaft will have to be equipped with a rock hoist and a man hoist.

Power.—In order to give due weight to the criticism made, it is assumed that mine "A" is provided with steam winders having their own boiler plant, coal bunkers and water purification plant; in fact, a typical plant of the old type. Mine "B" to be equipped with electric winders both at the surface and underground; the power to be purchased at the cheapest available rate, viz., ·56d. per unit. The cost of power for the steam winders to be 1·25d. per B.H.P. hour, or practically three times as much as found during the test on the Whiting hoists at the Village Deep mine.[*] The power consumption, ignoring all losses and repeat work, always done in stage winding, will be as follows:—

Mine "A": rock hoist, 250 h.p., man hoist, say, 150 h.p. = 400 h.p. (300 k.w.)

Mine "B": rock hoists, 625 h.p., man hoists, say, 375 h.p. = 1,000 h.p. (750 k.w.).

Working 300 days at 16 hours per day, the expenditure for power per annum will be:—

Mine "A" $\frac{300 \times 16 \times 400 \times 1\cdot25}{240} = £10,000 + 20\%$ for losses, £12,000.

Mine "B" $\frac{300 \times 16 \times 750 \times \cdot56}{240} = £8,400 + 33\%$ for losses, £11,200.[†]

Labour.—Mine "A" will have two hoists, and as they work two shifts, two sets of engine drivers and cleaners will be required. Mine "B" will have six hoists, two for each stage of hoisting, and the labour force required is as follows:

Mine "A": 4 engine drivers at £300 per annum … …	£1,200
4 natives, at £50 per annum	200
	£1,400
Mine "B": 12 engine drivers at £300 per annum … .	£3,600
12 natives at £50 per annum	600
	£4,200

Ropes, Guides, Rails, Oil, etc.—Each rock hoist would haul 600,000 tons per annum, and each man hoist a corresponding number of men and supplies, so that a new set of ropes must be allowed for each hoist once a year. Shaft guides in vertical shaft and rails in incline shafts also have to be renewed from time to time, so that the annual expenditure for each winder can be taken at £600. The expenditure under this head will, therefore, be as follows:—

Mine "A," 2 hoists, at £600 …	£1,200
Mine "B," 6 hoists, at £600 …	£3,600

Capital Cost.—The cost of two steam winders plus boiler plant for mine "A" will not be more than £30,000, and the cost of the electric winders for mine "B" can be taken as under:—

Two winders on surface … …	£25,000
Two winders underground, 1st stage	35,000
Two winders underground, 2nd stage	40,000
Total … …	£100,000

The charges for interest at 6%; depreciation at 4%, and redemption of capital at 2%, or a total of 12% will be:—

Mine "A," 12% on £30,000 …	£3,600
Mine "B," 12% on £100,000 …	£12,000

Summarising the annual expenditure for the two mines, we find:—

Item.	Mine "A."		Mine "B."	
	Total.	Per Ton Hoisted.	Total.	Per Ton Hoisted.
	£	d.	£	d.
Power …	12,000	4·80	11,200	4·48
Labour …	1,400	·56	4,200	1·68
Ropes, guides, oils, etc. …	1,200	·48	3,600	1·44
Capital charges	3,600	1·44	12,000	4·80
	18,200	7·28	31,000	12·40
Without capital charges …	14,600	5·84	19,000	7·6

It can be seen from the above that, notwithstanding the excessively heavy charge for power in the case of steam winders for Mine "A," the cost of hoisting per ton is greater for Mine "B" to the extent of over 5d., which is more than the whole cost of power when winding from 5,000 ft. Even if capital charges are neglected, the difference is still 2d. per ton. At the present time a large number of winding plants are already electrified and enjoy the full benefit of power at ·56d. per unit. Allowing £10,000 for the cost of electrification, which is more than ample, hoisting costs for Mine "A" would then be as follows:—

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[*] *Journal of the S.A.A.E.*, Vol. xi., p. 176.

[†] The extra loss is due to unnecessary hoisting at the upper end of the lower stages, and greater frictional losses in the incline shafts as well as greater transmission losses.

Power, 400 h.p., 300 k.w., plus 20% loss at ·56d.	£4,000
Labour, as before	1,400
Ropes, guides, oils, etc., as before ...	1,200
Capital charges, 12% on £40,000 ...	4,800
	£11,600
Cost per ton hoisted	4·56d.

Without capital charges, £6,600 or 2·64d. per ton

In this case the difference in favour of hoisting from a depth of 2,000 ft. as against 5,000 ft. would, therefore, be nearly 8d., and if capital charges are neglected, 5d. per ton hoisted. In other words, the cost of hoisting increases practically at the rate of 2d. for every 1,000 ft. vertical depth.

For the benefit of those who doubt the necessity of three stages for hoisting from a vertical depth of 5,000 ft., it may be stated that, in order to hoist from this depth in two stages, a vertical shaft of 4,000 ft. depth is required. However, three stage hoisting from the assumed depth will be required. It may incidentally be mentioned that if the ore from great depth is to be hoisted by the existing hoisting engines in the upper or last stage, all the disadvantages common to the existing plant will tend to increase the hoisting costs beyond those shown for Mine "B" in the table. If this is to be avoided a considerable expenditure on capital account will be required for new engines, skips, tipping gears, underground loading boxes, etc., on which interest must be paid in addition to the capital spent on existing plants.

The writer is willing to admit that under favourable circumstances, the cost of underground transport, despite the greater distances to be travelled, will not increase provided suitable mechanical means for handling the ore are developed. At present, the necessary straight drives are not existent, if we ignore the only mine capable of boasting of such, nor do we know whether petrol, electric, or compressed air locomotives are the best, or whether some other form of haulage is more suitable.

The increased demand for power in hoisting has not been denied by the critics, but Mr. Marriott and Mr. Johnson take exception to the increased cost of air power. One would think that a lengthy dissertation is not needed to demonstrate that it costs more to send air to a depth of 5,000 ft. vertical through 10,000 ft. of piping than to a depth of 2,000 ft. through 2,000 ft. or 3,000 ft. of piping. Whether the air is produced on the mine or whether it is obtained from a supply company has nothing to do with the point in question.

The arguments brought forward by the various critics are purely speculative, and based neither upon precise knowledge nor facts scientifically established. A good deal has been said about reduction of working costs by introducing mechanical methods and by the use of cheap electric power, yet it has been definitely asserted in the *Star* (Johannesburg) newspaper of November 16, 1911, that the introduction of electric power has had the very opposite effect. The writer definitely believes that working costs underground will ultimately be reduced probably in all departments, but it would be absurd to discount to-day results which are in the dim future. In view of the foregoing, the author still ventures to think that mining costs will rise as the depth of the mines increases, unless very material savings, not at present apparent, can be effected in any, or all the items constituting mining costs. If further confirmation be required, it can be obtained from recent statistics published by the Transvaal Chamber of Mines.

It is generally acknowledged that surface expenses, particularly reduction costs, have been materially reduced during the past few years, and they are now certainly less than in 1908. Yet working costs during the completed nine months of the year 1911 are as high as they were in 1908. To be correct, 18s. in 1908, and 17s. 11d. in 1911.

Mr. Hutton actually credits the introduction of the tube mill with the whole reduction in working costs which, as may be seen from the following statement, reached their lowest level (17s. 1d.) in 1909 :—

Yield, Working Costs and Working Profit for the Witwatersrand, 1905-1911.

Year.	Yield.	Working Costs.	Working Profit.
	s.	s.	s.
1905	35·82	23·167	12·653
1906	34·51	22·167	12·343
1907	33·94	20·83	13·11
1908	31·39	18·00	13·39
1909	28·917	17·083	11·974
1910	28·50	17·582	10·971
1911*	27·917	17·917	10·00

These figures are graphically shown in Fig. 13, and it will be seen that while there was a very decided drop both in costs and yield from 1906 to 1908, the drop in yield has since continued at a decreased rate, but working costs reached their low-water mark in 1909, and since then have risen at the rate of 5d. per annum, thus emphasising the tendency to reduce the margin between yield and costs. Now, the author ventures to say that this can be explained, and

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* For 1911 the figures are for January to September only.

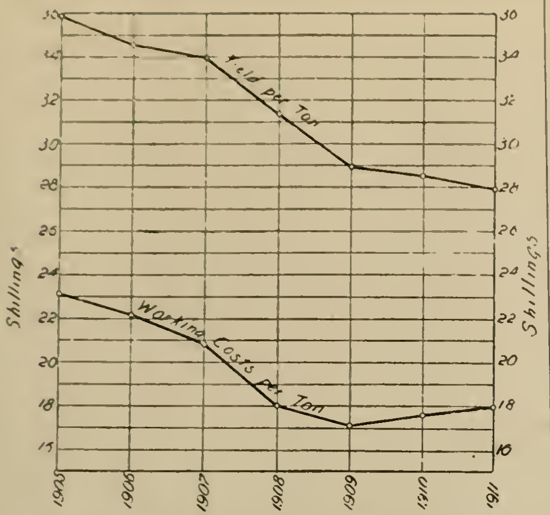

FIG. 13.—Yield and working costs for Witwatersrand 1905-1911.

partly used to illustrate his statement that the grade must fall, and working costs rise with increased depths of the workings.

With the introduction of the tube mill, increasing the capacity of reduction plants, and improving extraction, accompanied by a shortage of unskilled labour, the attention of those concerned was directed to the large tonnage of low grade ore already developed, but so far ignored in the endeavour to keep up the grade, while unconsciously keeping up working costs. With the addition of a considerable quantity of this ore to the monthly tonnage, handled by the same labour force and plant, and taking into account that development charges had already been met, it is obvious that mining costs fell considerably, and were accompanied by a corresponding drop in the yield. This state of affairs continued for more than two years, and gradually, as the supply of low grade ore already developed became less, normal conditions returned ; that is to say, the low-grade ore, if the supply was to be kept up, had to be obtained and paid for like the remainder, and this explains the turn of the tide in working costs. However, it does not explain the continued fall in the yield. Of the two explanations readily available, the first : That the grade fell with increased depth, is not sufficiently weighty, as there was no appreciable increase in the depth beyond the fact that some very deep mines started milling, and their tonnage is too small to carry so much influence ; yet it cannot be dismissed altogether. The other explanation is that, in order to keep up the tonnage on the enlarged basis, rendered possible by the now considerably increased capacity of the reduction plants, it became necessary to take in further blocks of low grade ore. That this is the case can be inferred from numerous company reports and chairmen's speeches of the period in question. Of course, there have been further contributory causes, such as the starting of new mines, which naturally commenced at the point then reached by older mines, and probably went farther seeing that they were not saddled with a similar heavy expenditure for reduction plant, or development, or both ; or some of the mines may have encountered patches of lower grade ore. A study of the monthly and quarterly returns will throw much light on the point. The nett result is that, at the present day, the returns for the whole Rand show a falling grade and rising working costs, thus confirming the author's statement. However, as the grade and working costs are changing more rapidly than could be expected from the increase in depth, there must be influences at work which, it is to be hoped, temporarily only, have upset what might have been considered the proper relation between working costs and yield at the present day. That this state of affairs cannot continue for any length of time is obvious, because the outlook would then be much more serious than the author indicated, yet he cannot refrain from calling the attention of his critics to the fact, and to invite their very serious study of the position thus disclosed.

Before concluding this part of the reply, the writer wishes to refer to Mr. Marriott's statement that "there was no value in the length of life when it was obtained by reduction of profit." This is a most important statement, because it shows clearly where the interests of the country at large and of the mine owners are at variance. From the point of view of the country it is desirable to see the greatest possible tonnage of ore extracted because every ton of ore mined and treated means an almost constant amount of money spent in this country. On the other hand, it is conceivable and it has in fact been proved that the mine owners would probably do better by mining and treating only a limited tonnage of higher grade ore in the shortest possible time with a maximum profit, but thereby reducing the country's share in the yield to a minimum. Of course, the investor is entitled to a fair return on his money, but as far as this country is concerned this money is represented, (as there are only very few shafts 4,000 ft. deep now existing on the Rand, and as more shafts of this depth are not likely to be sunk in the future, except in portions of the Rand not yet exploited) by the amount actually spent on the development and equipment of the mines only, and not by the money lost through channels which it would be futile to indicate.

However, as the greater tonnage will, with the existing reduction plants, give a longer life, and will, incidentally, reduce capital charges per ton, the lower working profit does not necessarily mean a corresponding reduction in the actual profit so that a long life does not necessarily mean a reduced profit.

Economies in Reduction Plants.—In discussing the proposed modification of the sorting plant, Messrs. Claudet, Truscott and Hutton are anxious about the possible increase in the value of the waste rock discarded. By referring to the paper, it will be seen that there is no possible reason why the value of the waste rock should, in the modified plant, be greater than at present. It will be remembered that at present from 30% to 50% of the total rock hoisted is not subjected to sorting because it is immediately upon reaching the surface disposed of as "fines." Now, in the modified plant, the extra waste rock is not obtained by closer sorting of that portion of the rock already subjected to sorting, but principally by sorting the portion now passed direct to the reduction plant. Incidentally, it is also expected to increase the efficiency of the work by better arrangements, principally in washing and screening. Since there is no reason to expect a smaller percentage of waste in the fine product (– 2 in. + ⅜ in.) than in the coarse rock, any objection to the proposed system could only be from a working point of view, but until this is settled by experimental work, judgment should be deferred.

Rather than expect an increase in the value of the waste rock a decrease, if anything, should be realised because of the greater facilities provided for doing efficient work. It has not yet been denied that there is generally at least 30% of waste in the rock hoisted, while in many cases the percentage is much higher. The average value of the waste now sorted out is probably from ·5 dwt. to ·75 dwt., and certainly not 1·25 dwt. as Mr. Truscott assumes, so that his argument, based on this value, falls to the ground.

The author greatly regrets the error which crept into Tables IV. and IVa., because, as the result thereof, the benefits derivable from the proposed modifications were much reduced. The correct figures for lines 5, and 8 to 12 for Tables IV. and IVa. have been reproduced in the attached tables. Both Mr. Truscott and Mr. Leupold were good enough to point out the error made, and the latter kindly calculated the correct cost of sorting per ton of rock submitted to sorting, viz., 0·128s.

Mr. Truscott rather deprecates the saving of 5d. per ton shown in the corrected Table IVa. as not very attractive, but the writer ventures to think that even a smaller amount is and should

Corrected Figures for Table IV. Lines 5 and 8 to 12.

Line Number.		Rate per Ton. Shillings	Fig. 2. Tons.	Fig. 2. Shillings	Fig. 5. Tons.	Fig. 5. Shillings	Fig. 8. Tons.	Fig. 8. Shillings	Fig. 6. Tons.	Fig. 6. Shillings	Fig. 9. Tons.	Fig. 9. Shillings	Fig. 7. Tons.	Fig. 7. Shillings	Fig. 10. Tons.	Fig. 10. Shillings
5	Sorting	·128	70	8·96	85	10·88	81	10·37	85	10·88	81	10·37	106	13·57	97·8	12·62
8	Total costs			363·04		274·12		277·19		263·62		268·93		266·31		271·18
9	Costs per ton hoisted			3·63		2·74		2·77		2·64		2·69		2·66		2·71
10	Saving per ton hoisted due to modified plant					·89		·86		·99		·94		·97		·92
11	do. do. in pence ...					10·68		10·32		11·88		11·28		11·64		11·04
12	do. do. in per cent...					24·5		23·7		27·3		25·9		26·7		25·3

Corrected Figures for Table IVa,

Line Number.		Rate per Ton. Shillings	Tons.	Shillings	Fig. 5a. Tons.	Fig. 5a. Shillings	Fig. 8a. Tons.	Fig. 8a. Shillings	Fig. 6a. Tons.	Fig. 6a. Shillings	Fig. 9a. Tons.	Fig. 9a. Shillings
5	Sorting	·128	70	8·96	50	6·40	40	5·12	50	6·40	40	5·12
8	Total costs			363·04		324·72		325·69		312·63		317·88
9	Costs per ton hoisted			3·63		3·25		3·26		3·13		3·18
10	Saving per ton hoisted due to modified plant					·38		·37		·50		·45
11	do. do. in pence ...					4·56		4·44		6·0		5·4
12	do. do. in per cent...					10·5		10·2		13·8		12·4

be greatly welcomed on the Rand, where, as a rule, engineers have to be satisfied with very much less.

As a result of the alteration to Tables IV. and IVa., Tables XI. and Tables XIa. also have been changed, and are reproduced in the altered form.

Mr. H. A. Tute fears that the necessary labour required for the increased sorting is not available, and expects that the already limited supply of natives for underground work will suffer. The writer, in reply, would like to refer Mr. Tute to the statements made by the various critics of the paper, who have pointed out how labour will become cheaper, as the result of mechanical methods to be introduced. However, as extra labour is only needed for sorting pure and simple, and as the tonnage submitted to sorting is raised from 70 to 106 only, while the waste removed is raised from 16 to 30 tons, the increased labour force need not exceed double of that now employed. This is an insignificant number, and not at all likely to disorganize underground work.

Corrected Figures for Table XI.

	Fig. 5.	Fig. 8.	Fig. 6.	Fig. 9.	Fig. 7.	Fig. 10.
Saving in operating costs per ton hoisted in pence (Table IV.) ...	10·68	10·32	11·88	11·28	11·64	11·04
Total saving in pence	12·05	11·61	13·35	12·65	13·00	12·33

Corrected Figures for Table XIa.

	Fig. 5a.	Fig. 8a.	Fig. 6a.	Fig. 9a.
Saving in operating costs per ton hoisted in pence (Table IVa.)	4·56	4·44	6·00	5·40
Total saving in pence	4·95	4·80	6·51	5·87

Mr. Meyer also mentions the question of insufficient labour, and further, its inefficiency, as throwing obstacles in the way of the author's proposals. Now, one of the points made by the writer was that so far not sufficient attention had been paid to sorting, also pointing to the necessity of not only employing better labour, but making proper arrangements for enabling this labour force to do really efficient work. The extra work it is proposed to do in the modified plant does not require more skill than the work now done; merely the volume of work is increased, and facilities will be provided for doing it more efficiently. Mr. Meyer also thinks that in order to sort out the increased percentage of waste rock, night work will be necessary, thus increasing the risk of doing bad work when working with artificial light. Whether sorting has to be done at night depends entirely upon the hoisting plant, because nobody will dream of doing night work merely because the sorting plant is too small. The writer has, on various occasions, condemned the use of artificial light in sorting work, particularly in connection with underground sorting, incidentally referred to by Mr. Marriott in discussing the paper.

With regard to the modification of the sorting plant, the proposals made do not embody any new features, or inventions, but merely show that by using the results of recent experimental work in laying out future plants, coupled with more intensive sorting, a considerable saving in working costs can be effected without increasing the total tonnage handled at the expense of the grade, as has been done so far. Obviously, the percentage of waste rock which can be sorted out depends upon the barren rock mined, and sent to the surface, and as this is admittedly very much greater than the percentage of waste now sorted, it must be profitable to remove the balance if at all possible, rather than crush and treat it in the reduction plant at considerable expense. Hence, while the application of the principles advocated by the author may be limited in scope, it is too much to say that they cannot be adopted with beneficial results.

The data furnished in the paper regarding better work in the crushing plant rendered possible by the proper allotment of work between the first and second stage, based upon experimental work, and now already confirmed by work upon a large scale, show conclusively how very much cheaper a modern plant based upon double stage crushing will be than the erstwhile single stage crushing plant, both in capital cost and working expenses. There was nothing new in this, but it was not generally known notwithstanding the author's statement to that effect in 1909.*

The item of capital charges as used in the paper has been misunderstood by Mr. Marriott. Capital charges as the writer understands them, and as has been made perfectly clear in the paper, are the charges for interest on, and the redemption of the money actually invested, and also for depreciation of the plant. Mr. Marriott, apparently, only considers the latter item which, of course, is comparatively small, and which, as he says, is provided for to keep the mine in a state of efficiency. Actually, however, capital charges are an amount to be set aside regularly to return to shareholders at the end of the mine's life, in addition to their regular dividends, an amount equal to the capital actually invested in the mine, but not necessarily the amount actually spent by the shareholder in acquiring his interest, thus enabling the shareholder to re-invest his capital when the mine is worked out. Unfortunately, the question is greatly complicated by the fact that the life of the mine is not known at the start, and, as has been seen, is a variable quantity, depending on many diverse factors. Unless capital charges on this basis are taken into account, there is no scale by which to measure good or bad engineering, because a mine with a large expenditure on capital account might appear to be as favourably situated as one with a small expenditure, while, in reality the latter may be financially sound, and the former totally unsound. While the influence of capital expenditure (for the reduction plant) on the extraction and therefore the yield is immediately apparent, its bearing on working costs while not so readily seen is nevertheless very pronounced, and in order to get a true picture of the mine's affairs capital charges as well as working costs should be deducted from the yield to show the actual profit. As things are to-day the average shareholder when he receives and presumably spends his dividend, does not know that he is also dealing with a portion of his capital.

In view of the foregoing a consideration of the point is, therefore, well within the scope of the paper.

Concluding, the writer wishes to thank the various members named for their courteous criticism, and hopes that the questions raised will not be allowed to lapse, for the reduction of the *true* working costs must remain the ultimate object of every effort on behalf of the mining industry.

THE AMALGAMATION OF GOLD IN BANKET ORE.

(*Read at March Meeting, 1911.*)

By W. R. Dowling, M.I.M.M. (Vice-President).

REPLY TO DISCUSSION.

Mr. W. R. Dowling (*Vice-President*): I wish to thank those contributors to my paper who have come forward with discussion, but must admit disappointment at not having succeeded in drawing discussion from many members who could have contributed important data. The subject of amalgamation has received a great deal of attention during the last year or two, and methods have undergone considerable modification. I think that the time was opportune for a complete review of the subject, and that the usefulness of the Society is discounted when members who can come forward, will not do so. However, I will reply to those members who have offered their views, in the order of date.

It would be unfair to deal severely with Mr. Laschinger's "carping criticisms," which are quite of interest as expressions of opinion from the mechanical engineer's standpoint, and as distinct from the experience of those who are engaged in the treatment of gold ore, and are actually responsible for the conduct of the necessary operations. After consulting some of the older mill managers, and reference to Mr. J. S. Curtis' original paper as well as his contribution to this paper, I find that the popular screen was about 900 mesh, and I therefore adhere to my original statement. With reference to the elevation of pulp, the surface contour of the reduction works site is of course a considerable factor in deciding the best arrangement. Unfortunately, on the Witwatersrand, owing to the country being comparatively flat, there is very little aid from nature in the way of fall. Nevertheless, if one elevation of pulp can be made to serve, as at the Simmer and Jack and Simmer Deep, thus simplifying the work from an operating point of view, the cost is reduced and efficiency enhanced. An operator naturally looks at things from the operating point of view. It is he who is responsible for the working costs, maximum running time and general efficiency in operating the plant, which is built and often designed by the mechanical engineer, but in which these points are liable to be lost sight of.

I agree with Mr. Laschinger that if in arrangement (A) the elevation were higher it would be as in (B), but unfortunately it is not so in actual fact, but I do not agree with him that arrangement (B) does not represent the actual case at the Simmer and Jack, as he suggests. The elevators, in the shape of two wheels, at the cyanide works of that mine, which Mr. Laschinger seems to have in mind have been out of action for a year or more. Mr. Laschinger admits that the single elevation where possible has advantages, since the Bantjes Consolidated mine has only one elevation. As a matter of fact elevation of pulp was a side issue in my paper, and the City Deep arrangemement (C) was introduced on account of the seven shaking plates for amalgamation, rather than on account of the double elevation of pulp.

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* *Journal of the S A A Eng.*, Vol xv., p 113

Mr. Laschinger suggests that in arrangement (E) the stamp mill is either very high up in the air or else the tube mills and plates are deep down in the ground. I do not think it is necessary to go to extremes in either. Since there is no amalgamation of battery pulp the tube mill cones may be placed closer to the stamp mill, and since, say, ten stamps would serve one tube mill the length of the stamp mill as a whole is reduced. Thus if one tube mill be taken as the equivalent of 20 heavy (2,000 lb.) stamps, the length of mill necessary to equal the City Deep in crushing capacity would be about the length occupied by about 130 stamps in a double row of 65 each. The fall necessary to collect the pulp after the tube mill plates is not great, and my critic has suggested the remedy, whereby the crushing plant may be divided into sections for elevating purposes, and the pulp of a limited number of stamps and tube mills elevated together. An inspection of the Cason mill of the East Rand Proprietary Mines will convince anybody that in scheme (E) there is no necessity for extremes in height of stamp mill foundations or depth of excavation for tube mills and plates, and this system is in course of installation at the Robinson Deep mill.

Mr. Laschinger seems to object to the tube mill circuit wherein the effluent of the tube mills is mixed with the stamp mill pulp and re-classified, and states that battery tailing is not classified and the coarse returned to the bins for restamping. The latter part of Mr. Laschinger's statement is very true, the coarse portion of the tailing pulp is not in practice returned to the mortar boxes for recrushing. The reason for not doing so is simply that the stamp has been proved to be most inefficient when recrushing coarse sand. This cannot be said of the tube mill, as this machine has proved, all over the world, to be the most efficient recrushing machine. Mr. Laschinger's difficulty then comes down to classification, and I would point out that the effluent of the tube mill may quite safely be allowed to join the tailing pulp for dilution and reclassification, provided the classifying cones are large enough. A single large diaphragm cone per tube mill, to deal with a due proportion of tailing pulp plus tube mill return, will afford the desired result and Mr. Laschinger's "ideal of simple and effective classification." All attempts to produce a finished product in one operation simply make for inefficiency and increased cost.

I am somewhat at a loss to understand my critic's meaning when he says "the locking up of gold on amalgamation plates is not an argument in favour of cutting down the total plate area to an absolute minimum: it is only an incidental advantage or an argument in favour of not having excessive plate area." It strikes me that the argument against locking up gold as in the case quoted, where £20,000 worth of gold became available for distribution, is one likely to appeal to shareholders. Mr. Laschinger says that "it is well known that most gold collects where there is a fall or impact of the stream against the plate and that the turbulent flow brings the particles of gold into contact with the amalgamating surface." My opinion is that more gold is caught at the fall because there is then more gold in the pulp to be caught than after it has travelled down the plate.

I understood that the City Deep had put in concrete launders, but I know of at least one recently erected large mill where sheet steel is used. Mr. Laschinger presumed correctly when he concluded that in the use of the word "cement" for lining launders I meant cement concrete. Since cement to bind the material and give a smooth surface to the launder, is the essential feature in a concrete launder, as is rubber in a rubber covered belt, it seems as reasonable to speak of "a cement launder" as of "a rubber belt." Mr. Laschinger suggests that amalgam traps are useful in catching iron. A trial of trap efficiency for this purpose was made on four traps in the Simmer East mill, one to each five of 20 stamps. These traps caught 26·8 lb. of iron in 52 days. It was estimated that 8,300 tons of ore were crushed during the time, and that $\frac{1}{2}$ lb. of iron was worn away from shoes and dies alone for every ton of ore crushed. Thus at least 4,150 lb. of iron were produced of which the traps caught only 0·65%. This hardly compensates for the loss of fall entailed by the introduction of traps.

Although one has to admit that our electrical friends have been slow in producing an efficient magnetic separator, credit must be given to Mr. Walter C. Brown for installing a small separator on the Simmer and Jack East. This machine is described in the proceedings of the S. A. Institute of Electrical Engineers.* The magnetic field is produced on a slowly rotating inclined steel disc 15 in. diameter set to receive the thick underflow of one of four diaphragm cones. The machine separates about 100 lb. of iron per day, and over 200 lb. after a stop of several days. This keeps the iron in the tube mill circuit at so low a content that no trouble of banking is experienced. Judging by the size of the iron particles caught on the magnetic separator no coarse iron appears to get so far. Another form of combined amalgam and iron separator has been installed by Mr. A. C. Holtby in the tube mill circuit at the Simmer & Jack, and consists of a large diaphragm cone of which the underflow delivers upon a magnetic separator. This has proved very efficient and it is to be hoped that a detailed account of it may later be available for the members of our Society.

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* Transactions *S. A. Institute of Electrical Engineers*, Vol. I, May 1910, pp. 122-125.

It requires nearly a two inch cube or square of any foreign material to choke a diaphragm cone classifier. With reference to Mr. Laschinger's query as to black sand caught by amalgam traps, I have no hesitation in saying that the tube mill circuit with efficient classification will deliver on to the tube mill plates for subsequent recovery more values either in the shape of amalgam, black sand or metallic gold than any amalgam trap yet installed on these fields.

In reply to Mr. Mather Smith I think we can safely assume that all gold in deep level pyritic banket ore that passes the amalgamating plates is more or less encased in or attached to fragments of silica or pyrite, as stated by Dr. W. A. Caldecott in his contribution to this paper. The finer the ore is crushed the more gold is exposed for recovery, both by amalgamation and cyaniding. Judging by the assay values of the grades of sand after treatment, although no percentages are given, it appears that this ore could with advantage be crushed finer. I sympathize with Mr. Smith in his struggles with his spitzlutten. I do not think that there is any likelihood of the safety cone accumulating too much coarse material so as to cause trouble, as well arranged cones when fitted with diaphragms will always give an underflow of the coarse material and an overflow of the finer. I have not tried Mr. Smith's idea of a 12in. pipe instead of the dewatering cone, but am inclined to think that a thick underflow would not be obtained owing to the lack of settling area. With reference to a plate with a hole in the centre instead of the usual diaphragm, I may point out that any means of supporting settled solids such as perforated plates, truck wheels, serrated discs and so on are merely modified forms of the diaphragm.

Coming to Mr. H. A. White's very interesting and well presented contribution to this discussion. I regret having overlooked the reference he has recalled to Mr. H. S. Denny having stated that the stationary plate was superior to the shaking plate as long ago as 1904. A complete investigation of the subject of amalgamation as to fall of plates, ratio of water, and area of plates can not well be undertaken by a single mine, and, as he suggests, the Mines Trials Committee might well pursue the matter further on the lines indicated. Though I do not think that market considerations should enter our discussions, yet for the general peace it may be well to maintain an even output. Since it is not proposed to do away with amalgamation entirely, the additional gold sent to the cyanide works does not increase to such an extent as to alter the present position materially. The value of the pulp entering the cyanide works of a low grade mine where the tube mill pulp only is amalgamated, is still probably poorer than that of a richer mine where amalgamation may be carried to excess. In the course of his remarks Mr. White has rather replied to the abstract from the Mining Magazine appearing in the Mining Journal of April 15th, 1911, where Mr. H. S. Denny in advocating fine crushing in cyanide solution proposes to eliminate amalgamation and depend entirely on the cyanide works. Here the 10% difference between theoretical and actual extraction may be embarrassing, although such a difference is rather high.

Referring to the alleged losses due to black sand reaching the cyanide works it appears that the efficiency of the classification is a very important factor. In the case of the Simmer Deep the difference between the amalgamation recovery before and after the elimination of the battery plates as quoted by Mr. Thomas, is so small, 56·8% before the change and 55·5% after, that whether any particular portion of the battery pulp passes over plates is of less importance than that the classification should be efficient. In the case of the Simmer and Jack the classifiers are unduly pressed and there is a smaller amalgamation recovery, and a consequent higher final pulp value. When it is found that the final pulp value is not appreciably higher there is only one conclusion left, and that is, that the gold whether existing detached or attached to particles of ore, is separated from all portions of the pulp, when efficient classification obtains. The case Mr White quotes where similar material to black sand found in the launders after the sand return spitzkasten assayed 40 ozs. and was equivalent to 5% of the total value of the slime is very interesting, but must be considered quite exceptional since no tube mills were in use. I have tried to find a similar little bonanza but could only collect 5½ oz. of gold, which works out in my case at 0·6% of the gold content of one month's slime. Analyses of this material gave mercury content of between 0·02% and 0·03% and this helps to account for some of the mercury loss in amalgamation, and indicates very finely divided and partly amalgamated black sand, which had escaped retention on the plates.

Replying to Mr. G. A Robertson, I was in error in conveying the impression that the Knights Central elevates its pulp twice. There is only one elevation of pulp to the tube mill classifiers, the underflow passing through the mills and over four stationary plates, and the overflow also over plates on the way to the cyanide works. I would, however point out that the plate house was not designed on the reduced plate area basis, nor is it usual to build a plate house of four times the floor area of a pump house to form a roof for the latter. Mr. White has suggested using strips of plates a foot wide for the purpose of ascertaining experimentally the area of plate necessary for amalgamation. This is quite a different proposition to that suggested by Mr. Robertson. There is no advantage in having plates in steps and the essential true plane of a plate is soon destroyed by moving. There is no particular necessity for the monthly event known as "clean-up day," as steaming may be done at intervals during the month. My critic's expression of "full amalgamation results" is rather vague and conveys nothing, but I think it is generally accepted that amalgamation of banket ore crushed through 64 and 36 holes to the square inch screens is unsatisfactory, due to excessive scouring of amalgam off the plates by the coarse particles, and that the battery plates in such cases had better be eliminated. With reference to the question of cost in cyanide consumption of dissolving the additional fine gold in the cyanide works against the mercury consumption for the same gold, I would say that a pennyweight of gold more or less in material treated by cyanide makes no practical difference in the amount of cyanide used per ton, as other factors such as precipitation and cyanicides in the ore determine the strength and quantity used. Sodium cyanide can actually dissolve about its own weight of gold. In the plant quoted by Mr. Robertson it is perfectly clear that the reason for the all-slime overflow of the classifiers is the want of feed to the tube mill, and the remedy is to start the tube mills some little time after the stamp mill to allow some coarse material to collect in the large cones. If the crushing in the tube mill is not good, unlimited classification would not assist. Most of Mr. Robertson's remaining criticisms are dealt with in my reply to Mr. Laschinger.

Mr. Jas. E. Thomas has supplied actual data of the two systems at the Simmer Deep, before and after the removal of the plates from the stamp mill. The figures speak for themselves, and fully confirm the contentions and arguments set forth in my paper, and I wish to thank him for his contribution.

In replying to Mr. Filmer's contribution, I welcome this as coming from one of the younger members of the Society. Although our membership is large, and the attendance at the meetings always good, there is not enough done by the large majority of members. I am afraid, however, that Mr. Filmer's scheme of classification and prevention of coarse particles passing over the plates would lead to an undue enrichment of the circuit with metallic gold, which would not overflow the classifier until very finely divided in the tube mill. I agree with him that a finer mesh screen should be used on the pebble eliminators, but of stout wire. One of the difficulties of using slimy water for stamp milling is the rapidity with which mill service water pipes become choked, and open launders would be difficult to operate.

Coming to the contributions by members of the Institution of Mining and Metallurgy, I have to thank my friend, Mr. George T. Holloway, for introducing my paper and for his remarks thereon.

In reply to Mr. Fearnside Irvine, I would say that although the paper referred to by Mr. J. S. Curtis was written in 1894, this was before the success of the cyanide process had completely converted everyone to its use. Many engineers still adhered to concentration methods, as Mr. Curtis' paper indicated. His paper dealt entirely with concentration, re-grinding, and amalgamation methods, in spite of the fact that the cyanide process was introduced on the Rand four years earlier, in 1890. A reference to Mr. Curtis' contribution to my paper where he states that the Robinson and Langlaagte were crushing 3 oz. and 2 oz. ore respectively, and obtaining 60% recovery by amalgamation from oxidised ore, will show that Mr. Irvine's comparison of present day recovery from ore of 5 dwt. to 8 dwt. is hardly fair. Mr. Irvine gives a good summary of my paper until he comes to the part dealing with water ratios, where he states that pulp of the ratio of 1·1 water to 1 of ore is fed to the tube mill. Where 400 tons per day of coarse sand is fed to the tube mill it is found that the best moisture is 38% to 40% (or 0·66 of water to 1 of solid). This ratio is, of course, not suitable for the subsequent amalgamation, where 55% moisture (or 1·2 of water to 1 of solid) is found to be the minimum that will permit flow of pulp over a plate whose fall is 18%. The additional water must be drawn from some other source, or in the case of scheme (E) may be obtained from the liquid underflow of the safety cone. It is not necessary or desirable to operate these large classifying cones of 6 ft. diameter and 9 ft. depth with an uprising stream of water, nor does a pulp coming from the stamp mill at 4½ water to 1 of ore mixed with the tube mill returns affect their efficiency. In comparing pyramidal spitzlutten with diaphragm cones as classifiers, it was found by G. O. Smart that the latter did the best work besides having many other advantages.* Conical classifiers are everywhere taking the place of the old spitzlutten. In reply to Mr. Irvine's questions I would say that a stamp of 1,250 lb. running weight might be expected to crush about 13 tons of ore per day through a nine mesh screen at a water ratio of 4½ to 1 of ore, that a short apron plate needs a 10% fall, the launders somewhat less if well constructed, that no additional water need be added to the cones, that the common method of pulp elevation on the Rand is now by means of centrifugal pumps, and that the power necessary will depend on the height of elevation. Having determined the height of elevation, the power may be calculated.

There seems to be some doubt in Mr. Truscott's mind as to whether the percentage recoveries quoted in my paper referred to the original value of the ore. I may state that they did, and as this is the only possible value that they could be referred to it hardly seemed necessary to say so. I would say further that extractions and recoveries on the Rand are always quoted in fine gold and not in bullion as suggested by my critic. Bullion means nothing when the fineness is unknown. After quoting cases of the percentage recovery by amalgamation on different mines of the world, Mr. Truscott arrives at the conclusion that the methods of Waihi are not applicable to the Witwatersrand. It appears to the writer that that method which gives the highest recovery at the lowest cost is the method which should be adopted by any mine. What the relative recoveries by amalgamation and cyaniding may be is of no particular importance, provided the combination fulfil the conditions mentioned above of highest recovery at lowest cost. The writer made no mention of the regular use of lime in mill service water, because ever since slime treatment has been generally adopted on the Rand, the use of lime has become so universal that it did not seem necessary to mention the fact that the alkalinity maintained in the mill service water is sufficient for improving amalgamation as well as for ensuring the better settlement of slime.

Mr. Truscott's parting shot was "that it would be better if on the Rand there was a little more opposition to these present tendencies, and perhaps a little new blood would also be good." The more usual charge against the Rand is that metallurgical engineers are too conservative, and will not accept new ideas and methods. With reference to the necessity for the introduction of new blood, the mines here have been and are open to all competitors and competitive processes ; the unfit are eliminated and the fit survive.

Both Mr. Truscott and Mr. McCarthy's contributions might be read to advocate the improvement and refinement of amalgamation up to such a point that cyaniding becomes unnecessary. Even if on Rand ore the amalgamation recovery rose to 80%, which is possible, the mines would still make a profit on cyaniding the final tailing.

I am very gratified that such pioneers as Messrs. J. R. Williams, W. A. Caldecott, and J. S. Curtis should have contributed to the discussion on my paper. Mr. Williams took us back to the introduction of the tube mill on the Rand, and rightly points out the necessity for the shaking table when dealing with a concentrate as was then the practice. With the changes in tube milling methods and the separation of a larger proportion of the ore for tube milling and amalgamation, the necessity for making the plate shake has disappeared. Coming to the last contribution to discussion by Mr. J. S. Curtis, I wish to thank him for the useful information of the early days, of which there is not enough on record.

In conclusion, whilst discussion on the important subject dealt with in my paper has been somewhat scanty, it has been a source of satisfaction to me to observe that one of the main objects with which it was written has been achieved, namely, by leading to the reduction in numbers and dispensing with the shaking motion of amalgamated tube mill plates in the majority of plants on these fields, and that this change has been justified by the later experience of those responsible for the operation of these plants.

NOTES ON HIGH DUTY GRAVITY STAMP MILLS.

(*Read at October Meeting*, 1911.)

By Peter N. Nissen.

DISCUSSION.

Mr. A. M. Kratz (*Member*) : Mr. Nissen's paper is an exceptionally good one, and he has taken great pains to make it as thorough as possible. His tests are conclusive. He has a stamp that will crush more rock per 100 lb. falling weight for less power consumption than the ordinary stamp. It is only when he generalises and estimates that he lays himself open to criticism.

Let me make a prediction, that in a few years time nearly all half-inch product will be sent to the tube mills direct ; it will not even go through the crusher station. The Modderfontein "B" plant will soon be arranged so that all fines will be washed in trommels, and anything under $\frac{1}{4}$ in. will be pumped direct to the tube mills, passing by the crusher station and mill. Several of the deep level mines are now making over 60% of what is called fine, by fine I mean anything under 2 in.; of this product perhaps 50% is $-\frac{3}{8}$ in. I see no great difficulty in pumping this direct to the tube mills, even through several hundred feet of pipe if the pipe line is properly laid out.

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* See this *Journal*, Feb., 1910.

Now, what would be the effect of "by-passing" this product on the Nissen stamp. Looking at his paper you will see that the feed is:

2 in. - 36·9
1$\frac{1}{2}$ in. - 18·6
$\frac{1}{2}$ in. - 15·1

70·6 leaving approximately 30% of $-\frac{1}{2}$ in.

I do not imagine that "by-passing" this 30% of $-\frac{1}{2}$ in. product would decrease the tonnage per 24 hours 30%. The decrease would not be directly as the amount "by-passed," still there would be an appreciable effect, and I think the effect would be greater on the Nissen stamp than on the City Deep stamp. It is reasonable to suppose that the Nissen stamp, with its circular screen, eliminates fine more easily than the City Deep stamp with its parallel screen. What I am trying to point out is that at the City Deep tests there was a large percentage of fine (30% of $-\frac{1}{2}$) which was a more favourable condition to Nissen stamps than City Deep stamps if you grant the argument that all $-\frac{1}{2}$ should and will eventually "by-pass" the mill. Give the Nissen stamps and City Deep stamps a run on ore from which all $-\frac{1}{2}$ in. product has been eliminated, and then compare the tonnages. I think that the difference would not be anywhere near the 30% that the tests now show in favour of the Nissen stamp. Looking again at the test, the water ratio is always higher with the Nissen stamp than with the City Deep stamp. It is quite a consideration to keep the water ratio as low as possible to save pumping costs. All excess water generally has to be elevated three times; first and second tailing lifts and then returned to mill after settling. This high water ratio may account for a portion of the high duty, for it is reasonable to suppose that more water eliminates fine faster.

Admitting tonnages as shown by the tests, I will compare the length of bin required in three cases to show that, instead of there being a saving in capital cost by using Nissen stamps, there is a considerable loss. See diagram of lay-out: assumptions:— -9 mesh screen and 40,000 tons per month to be crushed:—

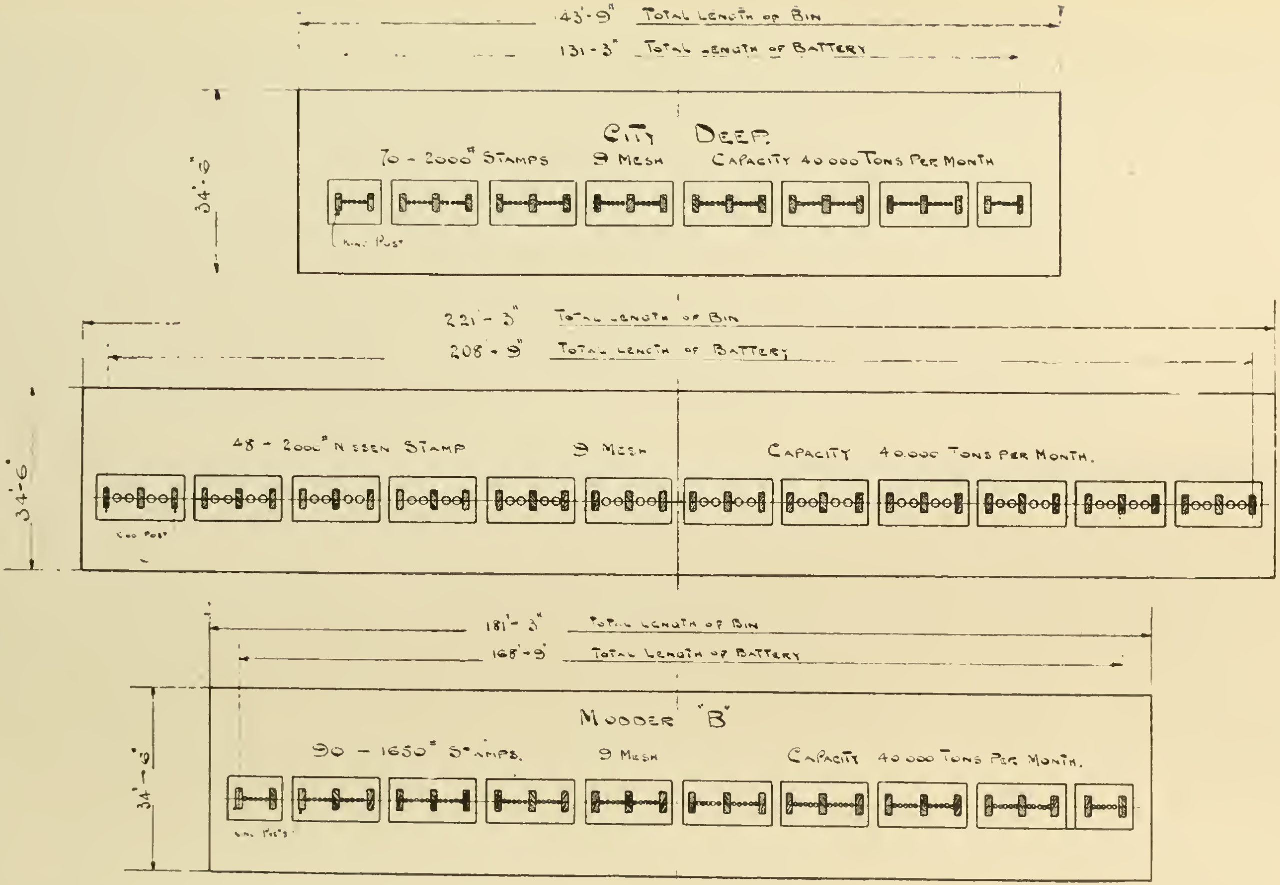
	Case I. Nissen.	Case II. Modder B.	Case III. City Deep.
No. of stamps ...	48	90	70
Weight ..	2,000 lb.	1,650 lb.	2,000 lb.
Total length over stamps ...	208 ft. 9 in.	168 ft. 9 in.	131 ft. 3 in.
Total length over bin	221 ft. 3 in.	181 ft. 3 in.	143 ft. 9 in.

It can be seen at a glance that assuming the saving in capital expenditure to be proportional to the length, there is a saving in the case of 2,000 lb. stamps of 35%, and in the case of 1,650 lb. stamps of 18% in favour of the ordinary stamp. This is due to the fact that a 1,400 lb. stamp takes up as much room as a 2,000 lb. stamp. Few are ready to say that a 2,000 lb. stamp is not a commercial success.

The excessive length of Nissen stamp plant is a drawback. To use Nissen stamps not only increases capital costs of bins and building, but it also adds to pumping costs. Length of Nissen batteries for 40,000 tons crushed equals 208·75 ft. (see sketch of 2,000 lb. stamp mill) – 70 stamps equals 131 ft. 3 in. The launder will be 38·75 ft. longer, causing an increased lift of 3·87 ft. on the pumps. Re-distributing to the tube mills double this making 7·74 ft. total extra lift due to extra length of mill.

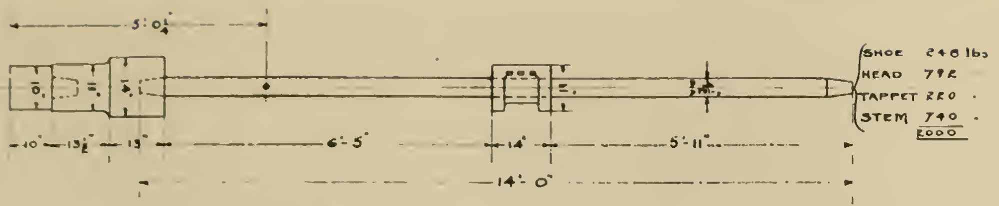
As the product to be lifted equals 8 tons per minute approximately and requires 1 h.p. per foot lift, then 7·74 ft. × 1 h.p. × £15 per annum equals £116 approximately, additional cost of power only, not inclusive of additional maintenance or increased depth of pump pit.

So far I have not touched on Mr. Nissen's claim of saving in height of building. There is little or nothing in this point. The height of bin is decided entirely by capacity and nothing else. As to the building, the saving here would be very small. Mr. Nissen lays great stress on the low centre of gravity of his stamp (see diagram). It will be noted that the centre of gravity of the Nissen stamp is 5 ft. $\frac{1}{4}$ in. from the bottom of the shoe while that for the City is 5 ft. 11$\frac{7}{8}$ in., a difference of only 11$\frac{5}{8}$ in. The over all length of the City Deep stamp is 17 ft. 9 in., the over all length of the Nissen stamp is 16 ft. 6$\frac{1}{2}$ in. I have also added a Modder "B" stamp, and a stamp which the Albu group use at the Roodepoort United and West Rand Consolidated. The other Nissen stamp is put in to show Mr. Nissen's latest stamp, which he expects to use in the future. It appears that the heavier the stamp the lower the centre of gravity, and if the City Deep stamp were made 9$\frac{7}{8}$ in. diameter as the Albu stamp, then its centre of gravity would be still lower. Thus it will be seen that the difference between the position of the centre of gravity in the two stamps


43'-9" Total Length of Bin
131-3" Total Length of Battery
34'-9"
City Deep.
70 - 2000# Stamps
9 Mesh
Capacity 40 000 Tons Per Month
King Post
221'-3" Total Length of Bin
208'-9" Total Length of Battery
34'-6"
48 - 2000# Nissen Stamp
9 Mesh
Capacity 40.000 Tons Per Month.
King Post
181'-3" Total Length of Bin
168'-9" Total Length of Battery
34'-9"
Modder "B"
90 - 1650# Stamps.
9 Mesh
Capacity 40 000 Tons Per Month.
King Posts

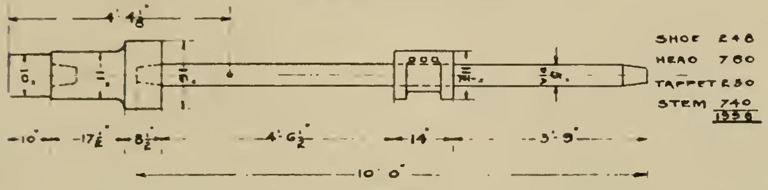
Length of Bin & Building for 48 - 2000# Nissen Stamps = 221 25'
" " " " " 70 - 2000# Stamps City Deep = 143 75'
= 35% Saving in Favour of City Deep

Length of Bin & Building for 48 - 2000# Nissen Stamps = 221 25'
" " " " " 90 - 1650# Stamps Modder "B" 181 25'
= 18% Saving in Favour of Modder "B"



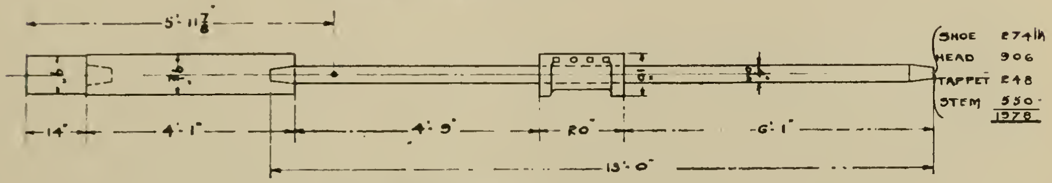
— NISSEN STAMP AT CITY DEEP —

2000 lbs FALLING WEIGHT



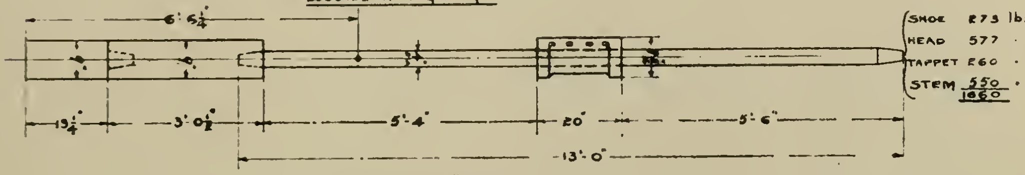
— NISSEN STAMP NEW DESIGN —

2000 lbs FALLING WEIGHT



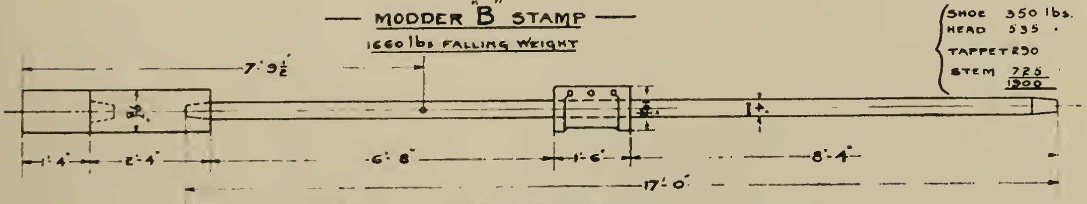
— CITY DEEP STAMP —

2000 lbs FALLING WEIGHT



— MODDER "B" STAMP —

1660 lbs FALLING WEIGHT



— WEST RAND CONSOLTD. & ROODEPOORT UNITED —

1900 lbs FALLING WEIGHT

under test is not much, and the total length is about the same. I certainly agree with the author that long stems with a heavy tappet constitutes bad practice. Keep the weight low, but this can be done with ordinary stamps, and, while exceptional opportunity is offered by his construction, still the centre of gravity of the ordinary stamp can be made lower, and the over all height of battery considerably reduced, if found advisable.

Mr. Nissen has a fine design of stamp which crushes with less power and effects some saving in iron losses per ton milled. It is a beautifully compact unit, and will make its way. The only argument I have brought up against it is that for large tonnages it will not crush as much rock per foot run in length of mill as our later stamps, thereby adding to capital costs, and cost of pumping. He will have to get units, two of which will equal five 2,000 lb. stamps; that is, he will have to get stamps of a capacity of $\frac{(5 \text{ stamps} \times 22 \text{ tons})}{2} = 55$ tons per 24 hours before he can compare capital and pumping costs.

Mr. M. Torrente (*Member*): In reading the very interesting paper of Mr. Nissen's I have come across a couple of details which require correction, viz.:—

1. The author states that it is not possible to keep a mortar box of the ordinary type rigid on its foundation. There are several mills at the present moment on the Rand where the mortar box is absolutely rigid on its foundations.

2. The diameter of shoes and dies in stamps from 1,250 lb. to 2,000 lb. is said to be the same *in all cases.* This also requires correction. As far back as 1908 stamps weighing 1900 lb. were designed where the diameter of the shoe was increased to 9⅞ in., and the distance between stem centres to 11 in. The mortar box was also lengthened accordingly. The duty of these stamps has been kept up without difficulty above 20 tons per 24 hours, using a screen of ·25 in. × ·6 in. aperture. It is thus evident that the relation between the shoe area and the falling weight had been taken into consideration at the time above mentioned.

Mr. W. H. Jane (*Member*): The author in his remarks on gravity stamps *versus* other crushing machines, remarks that it has been demonstrated on many fields that the cost of running stamp mills is lower than that of any other machine. Granting that this is so, I am still not satisfied that it is impossible to reduce our ores at less cost than with gravity stamps with their crude, unscientific lift by means of cams, probably the most crude piece of modern machinery in existence. I believe it would repay our Mines Trials Committee to experiment with a succession of breakers; first using a gyratory breaker, passing the product through a trommel to a knapper with the jaws swinging at the bottom, and again to a knapper with the jaws swinging at the top, The oversize product to be returned to the head of the circuit by a conveyor belt. This would require a tube mill with the largest internal diameter consistent with the point of highest efficiency, when we would probably not be limited to ⅜ in. size feed. With slow speed motors coupled direct to the pinion shafts similar to those at the City Deep, with good solid plummer blocks and bed plates, the cost of running tube mills should decrease considerably. With a modern mechanical pebble feeder like those in use on the Simmer Deep and Jupiter joint plant, with graphic meters recording to the load, hour by hour, and a tube mill of 8 ft. 6 in. internal diameter it should be an easy matter to get efficiency from tube mills. In regard to the uneven wear of crusher jaws, it has been already demonstrated that by installing devices in the chute feeding the crusher, it is possible to wear the jaws very evenly, simply by diverting the ore so that it does not all fall into the centre of the jaws.

The author's single mortar has undoubtedly an advantage over the multiple mortar in theory, but I am not satisfied as to its advantage in practice. It has been conclusively demonstrated that a double discharge mortar box gives no higher efficiency than the present single discharge mortar, because as long as there is sufficient area to admit of the discharge of the crushed product nothing is gained by increasing that area. I have experimented by increasing the volume of water flowing through the mortar and find that nothing is gained by excess, which goes to substantiate the fact that the present discharge area of boxes is sufficient. Of course the addition of water in excess sets up a reaction because the partly crushed product is washed against the screens thus increasing the height of discharge. This refers to the modern practice of introducing the water into the mortar by means of jets playing on to the dies.

The author's tables seem to prove that there is a much higher efficiency gained by using Nissen stamps, but were the tests carried out under the same conditions of supervision, etc.? Analysing the figures given by the author, let us take those given for the trial on June 23, 1911. We find the Nissen stamp of 1,932 lb. using only 4·09 h.p. per stamp, whilst the City Deep stamps of 1,863 lb. are given at 4·21 h.p. per stamp. It seems difficult to reconcile these figures if the feeds of both stamps were correct. Stamps properly fed recoil from the dies, and the only factor to show any alteration should be resistance of guides, which should not show anything like that difference, even allowing for change of centre of gravity, etc. The numbers of drops per minute are given at 103 and 100, 8½ in. drop. I venture to dispute the possibility of running 1,932 lb. stamps at 103 drops per minute with an 8½ in. drop under ordinary conditions. The City Deep stamps at 100 drops per minute, 8½ in. drop, were not running efficiently, that is, the breakages must have been abnormal, therefore loss on running time must have been high. The ratio of water to ore in the same tests gives, Nissen stamps 5·4 to 1 which is reasonable, City Deep stamps 2·7 to 1 which is unreasonable, certainly not enough to carry off the crushed ore. The same discrepancy as to power consumed applies up to September 16, but the water to ore ratios in the last tests are similar but much too low for practical work. No doubt the increased area of Nissen screens allowed more thick pulp to pass.

The author assumes that the ordinary stamp will only run 90% of its time, but the stamps on the Simmer Deep and Jupiter joint plant are averaging over 98% of actual running time since the mill plates have been removed. Again the stoppages for screen changing of ordinary stamps will be one-fifth of that for Nissen stamps, or taking the author's figures 100 to 48, $\frac{12}{25}$ of the time.

In regard to wear of shoes and dies the author attributes the smooth wearing of dies to the better distribution of the ore in the Nissen mortar. Experience has taught me that dies wear much smoother with a 9 mesh screen and high stamp duty than they do with a 700 mesh and a corresponding duty. Reason asserts that the cause of this is that the dies wear out quicker. With too low a ratio of water to ore, and an ordinary multiple mortar discharge, the dies will not wear evenly. Broken shoe shanks do not occur so frequently now that the manufacturers are beginning to realise our wants.

The author's figures of cost per ton milled of shoes and dies are high. Some maker's shoes at the Simmer Deep-Jupiter joint plant work out at 0·78d. per ton, and dies of other makes average 0·48d. per ton, which gives 0·63d. per ton milled. This cost is rapidly falling as the aperture of the screens increases in size.

The author's calculation of the saving in the height of the mill building is also wrong. The mill at the Simmer Deep Jupiter joint plant is about as low, but differs in construction from the author's suggested mill. The capacity of his bins is insufficient to run over Sunday or a respectable breakdown at the mine. As for the steel tank bin it would be eaten away by the sulphuric acid liberated from the ore by the addition of water. The item, "saving in mortar box liners" will be more than counterbalanced by the extra cost of screening.

The worst defect of the gravity stamp, the cam lift, is present in the mill under review. In regard to cam shafts, which the author improves by placing the bearings nearer the stamps, there is at present running at the Simmer Deep-Jupiter joint plant an improvement by Mr. H. C. Behr, of the Consolidated Goldfields, which reduces the length of the shaft by 18 in., the details of which I am not at liberty to give Most of the trouble with cam shaft breakages was due to experimenting with steel shafts, which have proved unsuitable for heavy stamps, too high speed of shafts and irregular voltage, these causing the speed to vary from 97 drops to 103. Most millmen will appreciate the result of the morning shift setting tappets at 97 drops per minute and the mill speeding up to 103 in the evening and continuing during the night at top speed.

Several makers have a D.F. iron shaft on the market which runs over 500 days and works out at 0·015d. per ton crushed. In conclusion, I would like to thank the author for his valuable paper, and hope he will be able to demonstrate more clearly in his reply the advantages of the Nissen stamp.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Influence of Copper on Corrosion of Iron.—"The unwarranted use which has in the last few years been made of the name of the American Society for Testing Materials in urging the validity of the so-called acid corrosion test, concerning which definite action has been taken in the report of the Committee on Corrosion of this Society at the present meeting, has made the Society unwittingly more or less responsible for two rather grave errors. These are first, that the rapidity of a given sample of iron or steel to dissolve in acid measures its tendency toward natural corrosion, and second, that a low acid test is an index of great purity of the specimen—that is, of an extraordinarily high content of iron. Enough data are now available to show the unreliability of the first assumption, and it is the purpose of this short article to show the fallacy of the second.

In investigating samples of iron and steel which had withstood corrosion for years, and which notwithstanding dissolved in acid very readily, together with samples which withstood solution in acid in a remarkable manner, and yet were rusting at the ordinary rate, it was found that the acid resistant specimens contained in every case a substantial amount of copper; that is, the presence of copper seemed to be the controlling factor in the resistance to solution in acid.

Upon examining the literature of the subject sufficient data to justify this conclusion was found. A number of investigators have studied the matter, and the results of all are fairly well presented in the Carnegie Research Report of Pierre Breuil on Copper Steels in the Journal of the Iron and Steel Institute of Great Britain for 1907. It is here found that small amounts of copper in iron or steel reduce enormously the tendency to pass into solution in acid.

In order to determine whether there is any foundation for the alleged fact so widely disseminated, that a highly acid-resisting iron or steel is also therefore a very pure product, the writer succeeded in having made a number of heats in a basic open-hearth steel furnace, under ordinary conditions, and using ordinary materials, into which metallic copper was added a little before or during tapping. The treatment of the product in the bar and sheet mills, and the annealing operation, was in every case as nearly identical as could be maintained. Although as the work developed many heats were made, the results were so uniform that but few will be here presented.

	C	Mn	S	P	Cu	Loss in weight in 20% sulphuric acid in 3 hrs.
A	·08	·50	·018	·017	—	·2235 grams
A_3	·08	·50	·018	·017	·21	·0075 ,,
B	·10	·41	·027	·026	·19	·0082 ,,
C	·09	·31	·031	·063	·19	·0095 ,,

The influence of small amounts of copper in these heats is truly remarkable and shows conclusively that this wonderfully slight tendency to dissolve in acid is not indicative of great purity.

Inasmuch as the presence of metallic copper in contact with iron *increases* enormously the rapidity with which such iron will dissolve in acid, it at first seems surprising that in this case copper should *retard* this action. But we have a direct analogy in the case of zinc and copper. Zinc free from iron dissolves in acid very slowly; mix with it some metallic copper, however, and solution is greatly accelerated. But make an alloy of the two, namely brass, and the solvent action of the acid is negligible. So long as copper is not added to iron or steel in excess of the amount which can remain homogeneously alloyed therewith, the writer can see no reason why there may not be an advantage in its use. But such advantage cannot be ascribed to purity, but rather to what we may call a ferro-brass of small copper content. Reasoning from analogy with zinc, pure iron should dissolve in acid very slowly, but the converse is here shown not to be true, namely, that a slow-dissolving steel or iron is on this account to be considered an exceptionally pure product."—W. H. Walker, Ph. D. *Metallurgical and Chemical Engineering*, Sept., 1911, p. 453. (H. A. W.)

The Compressibility of Molecules. — "On June 14 Prof. Theodore William Richards, of Harvard, delivered this year's Faraday lecture at the Royal Institute, and was afterwards presented with the Faraday medal. The following is an abstract of the lecture :

The lecture concerns the methods and results of some extended series of investigations designed to add to the foundations of human knowledge by careful experiment. Among quantities worthy of exact measurement the properties of the chemical elements are surely some of the most fundamental because the elements are the vehicles of all the manifold phenomena within the range of our perception. Weight is one of the most significant of these properties, and in the determination of atomic weights each substance employed must be free from all suspicion of impurity. Volume is also important. Twelve years ago the quantity *b* in Van der Waals' equation seemed to the author to be not really constant but subject to change under influence of temperature and pressure. This quantity is supposed to be dependent on the space occupied by the molecules, and if it be changeable are not the molecules themselves compressible ?

If changes in the bulk of molecules are to be inferred even from gases, may not expansion and contraction of solids and liquids afford a much better clue to the relative expansion and contraction of these molecules ? The ordinary conception of a solid has always seemed to me little short of an absurdity. A gas may be imagined with moving particles far apart ; but could a rigid body like steel have such an unstable structure ? The evidence seems to show that the interstices between atoms in solids and liquids must be small relatively to the size of the atoms themselves, if, indeed, there are any interstices at all. The compressibilities of 35 elements were studied with sufficient care to leave no doubt as to their relative values. It became manifest that the formation of a compound of a compressible element was attended with greater decrease of volume than was the formation of a similar compound of a less compressible element, other things being equal. This is just what the theory leads us to expect, and is inexplicable by any other hypothesis known to me. Then as to cohesion, if the pressure of chemical affinity causes compression of the atom, may not the pressure of cohesive affinity do the same ? The affinity which prevents solids and liquids from vaporising is generally admitted to set up great internal pressure, and must it not tend to compress the molecules ? Molecules of high cohesive affinity—those of substances hard to volatilize—should be much compressed and have a small volume, whilst molecules with little cohesive affinity should be larger. Moreover, those molecules already much compressed by their elf-affinity would be only slightly affected by additional pressure. Given two otherwise similar substances, the less volatile would be less compressible, denser, and have greater surface tension. These outcomes of the theory correspond with observed facts in most of the cases so far studied. Differences of structure and of chemical nature sometimes hide these relations, but the bulk of the evidence strongly indicates that cohesiveness, as well as chemical affinity exerts pressure, and that each plays a part in determining the volumes occupied by molecules.

Carried through to its logical conclusion, the idea that atoms are compressible gives one quite a new conception of the molecular mechanics of the universe. The influence of atomic compressibility may be perceived everywhere, and in most cases each fact seems to fit easily and without constraint into its place in the hypothesis. Even apparent exceptions, such as the abnormal bulk of ice, may be ascribed in a reasonable fashion to superimposed effects.

The lecture was received with marked appreciation by a full audience."—Prof. T. W. Richards.—*Metallurgical and Chemical Engineering*, Sept., 1911. p. 463. (H. A. W.)

Quantitative Estimation of Potassium Perchlorate in Potassium Chlorate.—"The method consists in distilling, into a solution of the mixed salts, SO_2 obtained from $NaHSO_3$ and H_2SO_4. The excess of SO_2 is gently boiled off ; while boiling, a solution of $AgNO_3$ is added, giving a precipitate of AgCl (1 mol. AgCl = 1 mol. $KClO_3$). $KClO_4$ is not attacked. The delicacy of the method was tested with carefully prepared mixtures and found satisfactory. In the samples of $KClO_3$ examined about 0·5% $KClO_4$ was found."—K. Scheringa-Nijkerk, *Pharm. Weekbl.* 48, 15-18, *Chem. Abstracts, Am. Chem. Soc.* vol. V., No. 17, Sept. 10, 1911, p. 2,791. (J.G.)

The Analysis of Tin Ores.—"The ore is roasted, extracted with HCl and the residue melted in a Hessian crucible with KCN. The heating is continued for a short time at a low temperature. The regulus is weighed, dissolved in HCl, the Sn is precipated with Al, and then dissolved in HCl in an atmosphere of CO_2. The solution is then titrated with $FeCl_3$ solution using KI and starch as an indicator."—Bayerlein-Essen, *Z. angew. Chem.* 23, p. 969 ; Chem. Abstracts, *Am. Chem. Soc.*, vol. V., No. 18, Sept. 20, 1911, p. 3,023. (J. G.)

Determination of Chromium in Chrome Iron Ore.—"The author takes 0·5 gm. of the powdered ore with 5 gm. of a fused and powdered mixture of two parts of anhydrous sodic carbonate with one part borax glass, and heats the mixed ore and fusion-mixture in a platinum crucible over a bunsen flame until all the ore is taken up—from an hour to two hours being required. When cool the mass is dissolved out with 300 cc. or 400 cc. of water containing one volume per hundred of sulphuric acid. Any residual unoxidized chromium is converted into chromate with 0·2 gm. of potassic persulphate, which precipitates, as peroxide, any manganese that may be present. Excess of persulphate is decomposed by boiling for half an hour or so, the MnO_2 is filtered off, and the chromium is estimated volumetrically by ferrous ammonium sulphate with a potassium ferricyanide indicator."—O. Nydegger, *Zeit. f. angew. Chem.*, 1911, 24, p. 1663.—*Met. and Chem. Eng.*, Oct. 1911. (J. A. W.)

Method for the Determination of the Total Sulphur in Coal.—"1 gram of the substance is intimately mixed in a porcelain crucible with 2 gm. CuO and 1 gm. anhydrous Na_2CO_3. The crucible is then placed in a hole in an asbestos pad and heated with a Bunsen: if stirred occasionally the action is complete in about 2 hours. When cool the crucible is transferred to a spacious beaker and moistened with Br water to oxidise any sulphites, etc., to sulphates: the residue is then treated with HCl and Br water, heated to effect solution and finally to expel Br, filtered and the S estimated in the filtrate in the usual way."—T. S. Warunis, *Z. anal. Chem.* 50, 270-2, Chem. Abstracts, *Am. Chem. Soc.*, vol. V., No. 18, 20th Sept., 1911, p. 3,024. (J. G.)

Potassium: Its Qualitative Detection as the Cobaltinitrite.—"Place 5 cc. of the potassium solution in a tall beaker of 50 cc. or 100 cc. size, set on a mirror beside it 5 cc. of pure water in a similar beaker, and add to each 2·5 cc. of sodium cobaltinitrite reagent and 5 cc. of 95% alcohol. If upon looking down through the solutions into the mirror an equally sharp image of the observer can be seen in each beaker there is no potassium present: if the test solution shows a dimmer image than the blank, potassium is present amounting to two or more parts per million. The only precautions necessary are to avoid making the test in an atmosphere dense with ammoniacal fumes, to be certain of the absence of ammonia from the solution and to be certain that the solution is neutral or slightly acid with acetic acid." Leon T. Bowser, *J. American Chemical Society*, vol, XXXIII., No. 10, Oct. 1911, pp. 1566-9. (J. G.)

METALLURGY.

The Liberty Bell Mill.—"The metallurgical practice here has frequently been described, and I will try not to dwell on well known features. The following are the units of equipment:—

1. Eighty 850 lb. stamps with suspended Challenge feeders.
2. Sixteen copper amalgamating tables, each 8 ft. × 4 ft., with three 1 in. drops.
3. Four Richards vortex (hindered settling), 3 spigot classifiers.
4. Eighteen Wiltley tables and 10 Deister No. 3 tables.
5. Three Abbé pattern, 5 × 22 ft. tube mills, the feed thickened by Dorr classifiers, or diaphragm cones.
6. Eight amalgamating tables, of the size given above.
7. Nine Dorr continuous settlers, 33 ft × 11 ft.
8. Six Hendryx type agitators, 17 ft. × 11 ft. above the 45° cone.
9. One equalizer vat, 20 ft. × 15 ft.
10. Moore filter plant, 7 vats, each 9 ft. × 27 ft. in area, and 8½ ft. to the coning.
11. Zinc shaving precipitation plant, capacity 1,200 cu. ft. of zinc.

A screen analysis of battery pulp shows: On 40 mesh, 25·4%; on 60 mesh, 10·9%; on 80 mesh, 5·9%; on 100 mesh, 6·3%; on 200 mesh, 9·6%; through 200 mesh, 42·9%. Some 38% of the battery pulp is flocculent. The power charged to the battery amounts to 160 horse-power.

The ore is stamped in cyanide solution. The recovery by amalgamation is materially less than in previous years of water amalgamation; it is more expensive in both labour and material and requires more skill, and the consumption of copper is considerable. Muntz metal, which has proved a satisfactory substitute elsewhere, has not been successfully adapted here, taking on a hard, glassy surface. The plates are kept rather wet, and any drip of quicksilver is caught in a trap. From a month's run the results of amalgamation were: from battery plates, 80% of all amalgam, yielding 29% bullion, 0·408 Au, 0·551 Ag; from tube mill plates, 29% of all amalgam, yielding 23% bullion, 0·153 Au, 0·825 Ag; the first plates have a grade of 2¼ in. per foot, and the second plates 1⅝ inches.

The concentration scheme is only now assuming definite form. Nine Willley tables take the underflow from the Richards classifiers, two take the middling, after removing the coarse on a Bunker Hill screen, and seven take the overflow, after thickening in six 6 ft. cones. The 10 Deister tables are to take the reground sand from the tube mills. Any oversize tailing from these latter tables is returned to the tube mills.

The great impediment to perfect work is the argillaceous slime. Coagulated by the alkaline solution, it is exceedingly buoyant and sustains coarse material, both sulphide and sand, until dilution is carried to extreme limits.

That amalgamating is still carried on has occasioned some adverse comment. It is recognised that the absence of this step would materially cheapen and simplify the milling. Sixty stamps, at most, would be required to crush the full tonnage through the coarser screens that could be used. It has been stated that the gold occurs irregularly in the ore, and hence is likely to be coarse. This gold would make an unwelcome element in the concentrate, which has not as yet been made amenable to local treatment. Were it feasible to concentrate successfully after regrinding, the battery plates might well be done away with and the coarse gold allowed to go into the tube mills with assurance that it would be ground and taken into solution; but that concentration after regrinding is not good practice here seems to have been amply demonstrated. The power charged to concentrating is 30 horse power.

Regrinding.—The tube mills are of the Abbé type, tyre mounted and with spiral feed. The tyres, both on the mill and on the supporting rollers, are of forged steel and promise indefinite life. The drive is by 50 h.-p. motors, bolted to counter shafts which in turn drive through heavy friction clutches to the tube mill shafts. The mills start readily with the use of the clutch, showing a maximum starting peak of 78 h.-p.; the running load varies from 45 to 48 h.-p. The lining is of 4 in. silex blocks, set in cement mortar, with the narrow edge to the wear. It gives a year's continuous service. The grinders are 4 in. imported flints, costing $33 per long ton delivered. The ends are lined with local cast iron, and the discharge is through a grating, which will probably give way to the Neal cone discharge. A typical screen test is as follows:—

Screen Mesh.		Feed. Per cent.	Discharge. Per cent.
On 40 …	…	47·1	0·7
On 80 …	…	29·4	11·4
On 100 …	…	8·2	9·2
On 200 …	…	6·4	22·3
Through 200 ..	…	8·9	55·4

As nearly as it is determined, half of the total tonnage crushed is reground. The efficiency of the regrinding varies with the excellence of the separation of the slime. Failure to remove this gives buoyancy to the pulp within and the grinding is poor. With this object in view, the above screen test is not satisfactory. The practice is to force a diaphragm cone and to care for the overflow of sand in a simple cone in series. The discharge from the simple cone carries slime out of all proportion. This is mixed with the discharge from the diaphragm cone to dilute it to 48% moisture. The result is to lose much of the benefit had from the diaphragm cone. It will undoubtedly prove better to combine the overflow from the three diaphragm cones in a single simple cone and confine the hammering effect of the fine material to one mill; or, better, a Dorr classifier at this point would be admirable. The feed to the other mills, being then diluted with clear solution, the grinding should be good,

The work of this diaphragm cone in preparing feed for a tube mill is remarkable, as the following screen test shows. The completeness of the elimination of fine suggests the benefits of hindered settling. This is a 6 ft. cone with 60° sides. The diaphragm is 13 in. from the point, with a 1½ in. annular space. The discharge is 1¼ in. diameter.

Screen Mesh.	Cone Feed. Per cent.	Discharge. Per cent.	Overflow Per cent.
On 40 ..	27·22	58·95	0·19
On 80 ...	24·82	30·66	8·97
On 100 ...	7·55	5·15	7·44
On 200 ...	8·89	3·42	17·00
Through 200 ...	31·52	1·82	66·40
Moisture ...	...	30·80	..

The absence of results from Dorr classifiers is due to the fact that the two machines installed here were the first, after Mr. Dorr's original installation at Terry. They conformed to his pattern neither in area nor in bottom slope, being built in what space was available, and were over-loaded. The results have been good through four years service, but not as good as would have come from the larger machine. The pebbles are fed during the day by the shift boss., being shovelled into the spiral feed; 135 lb. is the daily charge. The mills are entirely smooth running, and the cost of maintenance is at a minimum. The power consumed is 43 h.-p. per unit.

Dorr Settlers.—The latest step of great importance in improving the mill was the change to continuous from intermittent or charge settling in thickening the pulp for agitation. It is not possible to state definitely that certain measured results followed exclusively from this change. Certain definite improvements were shown by the experimental unit, and large improvements have followed the complete change, but at the same time other conditions were changed. The following are the principal advantages thus obtained: First, continuous extraction is obtained where before the solutions were inactive or re-precipitating; second, a given volume of settler space has 25% or 50% increased capacity, operated on the continuous basis; third, extraction going on in the settlers, the addition of a plant for settling has the added value of supplementing the deficient agitator-volume; and, fourth, labour is reduced one man on each of three 8 hour shifts.

This plant, originally of 5 vats, settling the pulp from the ratio of 5 : 1 to 2·5 : 1, has been increased to 9 vats, settling from the ratio of 9 : 1 to 2 : 1. The increase of solution has come with the addition after the battery of the concentrating plant, with its great volume of solution for washing and classifying. The four settlers recently installed have been placed out doors, with individual conical roofs and underneath shaft drive in a conduit, with great saving in cost as against providing the usual mill structure. The power consumed is one-fifth horse-power per unit.

Agitation.—The agitator capacity was designed with the expectation of using a low potential electric current to hasten extraction, following the results of extensive experimental work. The plan failed, and, without the current, the provision of space was inadequate. This has been remedied, in large measure, by the addition to the settler plant, already mentioned. The connecting of all agitators in series for continuous operation was a natural sequence of the adoption of continuous settling. The results appear better, but statistics on which to base exact conclusions as compared with previous charge agitation are lacking. Some saving in labour of operation and of maintenance is evident. These agitators operate steadily with little attention and very low cost of repairs, but the unit size is too small for a large tonnage plant, and the power consumption (6 to 7 h.-p.) is out of proportion, as compared with Pachuca tank practice or arm agitation, as carried on at El Oro. No benefit was obtained by spreading the pulp over distributors from the top of the central well, and it is allowed to plunge from the collar of the well. The power consumed is 50 horse-power.

Moore Filter Plant.—The equalizer, an integral part of the Moore filter plant, is a simple type of slow speed agitator, efficient for all depths of pulp in the vat and consuming a minimum of power. It has been lately patented and put upon the market as the Gordon agitator. The filter baskets of 66 leaves, each presenting two surfaces 8 by 6·5 ft. of free filtering surface, are carried on two 10 in. longitudinal I beams, which are in turn supported by transverse 6 in. beams which extend to the vat walls. The leaves are of No. 6 (20 oz.) canvas, reinforced on both sides at the bottom of the vertical stitching with a strip 3 in. wide of the same canvas. The vertical seams are on 2 in centres, and the wood strips between are ⅜ in. × ½ in. The frame is of ¾ in. iron pipe on ends and bottom, and the top is of strap and angle iron. No cocoa matting or other filter is used.

The leaves are made at the mill, the sewing (No. 4 linen thread, ¼ in. stitches) being by power machine (Singer 7·7), and cost complete $12; new canvas alone in place costing $8. The life of a filter is 18 months. All canvas requires hydrochloric acid treatment at intervals of three months. To do this a wash-water vat is brought to 1·25% HCl (18° B.) and to 140°F., the basket is immersed and the liquor circulated with a wet-vacuum pump. All the canvas in use can be treated within 30 hours. The cost of acid is 0·6 cent per ton of ore.

The lifting of baskets is by hydraulic cranes with cylinders 20 in. × 9 ft., the water 250 lb. pressure. Connection from the crane to water-main is by a specially reinforced coupling on a short length of metallic hose. The raised basket is held by a safety catch on the crane, and the transfer is effected by a 10 h.p., 3 phase, constant-speed motor, with a 3 armed trolley above. Transmission from motor to crane is through a Dodge multiple-disc friction clutch on the motor shaft. The service is severe, but the clutch does well. The vacuum connection from the basket is through a 3 in. hose to a pipe turning in a stuffing box, and is maintained throughout the transfer.

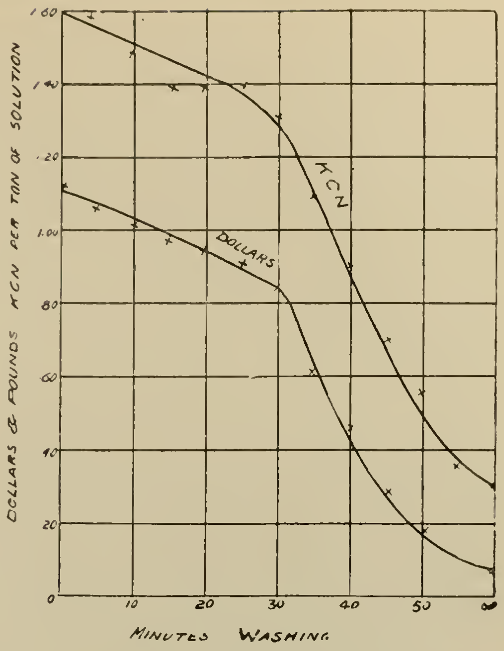
The greatest single step in perfecting the plant was the change from the common wet vacuum pumps to a combined dry vacuum and wet vacuum system; all entrained air is taken out at the upper end by a dry vacuum pump, an 8½ × 10 in. vertical duplex air compressor, and all solution at the lower end with centrifugal pumps in a sump 23 ft. below the top of the filters. Some leaks in canvas will occur, and sand sufficient in quantity to destroy a positive wet vacuum pump in a few hours is harmless to the centrifugal. Between the dry column at one end and the solution column at the other runs the main for strong solution. Parallel to this and leading from the same dry column runs the weak solution main, but to a different solution column and pump. Any basket may be connected with either vacuum main, or with the blow-off water main without disconnecting the hose. The vacuum never fails, being held at 19 to 20 in., near the maximum attainable at the altitude of the mill. To make this possible an excellent valve was designed. Its appearance is that of a globe valve, and its merit lies in seating an iron cone on a hard rubber ring of square section. It is quite impossible for sand to lodge on the ring so as to interfere with closing.

Filtration is accomplished in two groups of three vats each, the central one for loading and the two on either side for displacement in water. Basket No. 1 loads in the centre vat and is moved to the wash water vat at the right. Immediately thereafter basket No. 2 moves from wash water vat at left to loading vat. The cycle is thus: Loading, 50 min.; transferring and drying, 5 min.; displacing and discharging, 45 to 55 min.: transferring, 5 min. Each load is a ¾ in. cake, weighing 2·75 lb. dry per square foot, or 9 tons per basket load. This gives a capacity of 108 tons per basket per day, corresponding to 432 tons for the plant. Vertical uniformity of loading is secured by the use of three air lifts, which elevate pulp from the bottom of the vats and discharge it over the top.

The practice of displacing at once in water, without an intermediate wash of barren weak solution, can be approved ordinarily only on the assumption of good displacement and low strength solutions. Displacement is efficient, but the solution (1·6 to 1·75 lb. KCN at this point) is higher than was planned when the plant was designed. A factor that enters the special problem here is the 8% of moisture brought to the mill in the ore. Average results in washing are shown by the accompanying curve. The cake, partly dried, contains 33% moisture, not taking into account the solution in the pipes and channels, which is difficult to determine, but which must approximate one ton. The cake and passages thus retain a total of 5·5 tons of solution containing 8·8 lb. cyanide and $6·16 in gold and silver. The rate of displacement is 0·15 ton per minute.

In filtration there are two principal ends to serve: (a) to recover enough strong solution to restore the mill stock; (b) to recover the dissolved gold and silver, and that in solution of such strength as to ensure precipitation. Were the ore dry on entering the mill, 36 min. filtration would be sufficient. As the solution drawn in this period contains 7·72 lb. KCN, the efficiency of displacement is 87%, measured in cyanide. As it contains $5·08, the efficiency is 82%, measured in metals. It seems fair to accept 84 or 85% efficiency.

The mechanical loss of cyanide by dilution is that which cannot be restored to the mill stock. The fact that 35 tons of water is brought to the mill with the ore makes it impossible to secure the maximum theoretical efficiency given. From each basket-load the solution recoverable is (5·5 − 0·8 =) 4·7 tons, containing 6·48 lb. KCN. The combined mechanical loss is therefore $\frac{8{\cdot}8 - 6{\cdot}8}{9} = 0{\cdot}22$ lb. KCN, or 4·7c. per ton of dry ore.



WASHING CURVES, MOORE FILTERS.

In 55 min. washing, the recovery of gold and silver is $5·93, the apparent loss being 2½c. per ton of dry ore. This seems to be a maximum figure; washing to 70 min. showing almost complete removal. Regular sampling of the solution in washed cakes is not convenient, but, so far as done, it shows 1 to 2c. The loss in cyanide by dilution being so small and the recovery of dissolved metals being complete, the only remaining consideration is the low average strength, 0·75 lb. KCN, of the weak solution. Solution at 0·9 lb. KCN precipitates well. The use of a barren wash would insure this strength in the weak solution. On the other hand, 1c. per ton of ore will restore the few tons of weak solution to strength on the infrequent occasions of poor precipitation. It seems that added costs in depreciation, operation, and maintenance would offset any gains from an intermediate wash.

The weak solution after precipitation is used at low pressure, to force the cake from the filter, submerged in wash water. An advantageous change would probably be to perform this with air and thus return a more nearly dry basket to the loading vat. The removal of tailing is wholly automatic, by reason of the excess of wash-water available. The vat walls run down to three points, across which discharge single jets of water from ¼ in. nozzles, carrying the descending mud through ½ to ¾ in. orifices in the walls opposite the jets.

The operation of four baskets has been described. The fifth is used as a clarifying filter in the seventh vat. All solution, whether decanted or filtered, though apparently clear, requires clarification to insure clean zinc-boxes. In this service the canvas acquires a remarkably fine impervious and tenacious coating. To remove this the basket is returned to pulp filtration after 10 to 14 days. At times a coat of pulp has been gathered on the canvas before using it to clarify, but this seems an unnecessary refinement which reduces its capacity. The power consumed is 30 horse-power.

The Zinc Precipitation shows little that is unusual. The results are usually excellent with average heads of $1·25 and tailing of 1 to 2c. The flow of solution is 0·7 ton daily per cubic foot of zinc, and 2·4 tons per ton of ore milled, $\frac{3}{10}$ of the whole mill solution being precipitated.

Pumping Plant.—The handling of ore and solution necessitates much pumping. This is done with improved Byron Jackson slime pumps. The pumping units are in duplicates for pulp. The liners used are $\frac{1}{4}$ to $\frac{3}{8}$ in. cast iron. Manganese steel is to be tried. As it stands, the record shows this to be a good centrifugal pump. The life of the liners varies with the thickness of the pulp and the proportion of clay to sand The tube mill feed with nearly clean sand gives the shortest life, and the agitator discharge, very thick and with all the clay, gives the longest. Notwithstanding this good service from the centrifugal pumps, I have been for some years of the opinion that the proper pumping equipment for this mill would be a low pressure compressor with air-lifts for almost all transfers, and to replace the mechanical agitation. The displacement of motors, belting, and shafting, with their need of skilled supervision, would far outweigh the loss of efficiency in the air-lifts. The milling operations would be characterized by extreme simplicity.

Use of Chemicals.—The mill-sheets show this average in cyanide and protective alkalinity :

		Battery Head.	Second Plate Tailing.	Filter Heads.	Consumption.
KCN	...	1·75	1·64	1·50	1·48
P. A.	...	2·32	1·44	1·92	—
CaO	...	—	—	—	7·5
PbO	..	—	—	—	0·33

The mixed salt, 99% KCN, is used, no advantage being evident at first in trials of the 130% salt, but a recent concession by the makers has led to further trial with the 130% salt, with better results. The figures given are in pounds per ton of solution. Durango lime is used and the figures are in equivalents of caustic soda. Until September, 1909, lead acetate has been used, $\frac{1}{4}$ to $\frac{3}{10}$ lb. per ton of ore being added at the agitators. Since that time litharge, $\frac{1}{4}$ lb. per ton, has been added with the tube mill feed. An apparent improvement of 5% in silver extraction from the charge is based on inadequate tests."—CHARLES A. CHASE.—*Mining and Scientific Press*, June 24, 1911, p. 840. (H. A. W.)

SEGREGATION OF SOLIDS IN LIQUIDS.—"Drop a pebble in a pool of water and with it will follow a limited zone of liquid impelled by pressure in front, skin friction, and suction from vacuum behind. Two pebbles dropped in contact tend to keep together because in the wake of one is the line of least resistance and because of the difference of resistance the hindmost will press upon the one forward. If they have unequal and flat faces it is possible for them to be deflected into separate paths. A cluster of pebbles dropped together will tend to hover to the same path and are individually less exposed to deflecting influences and thus a zone of pebbles and liquid moves downward. If light material is enveloped within that zone every tendency is for it to remain there in the downward course, unless some exterior interruption occurs. If, instead of into a pool, a mixture of heavy and light grains is dropped into a body of water having an easy flow at near horizontal, the agglomeration will sag as it drifts, giving a lengthened path of travel and an exposure to disturbance of the zone aggregation by the flow of water across the gravitation path and a segregation of the light material from the heavier is affected. Because of these principles, settling and segregation are better accomplished with an easy lateral flow than when the flow is vertical, as with a cone where the inflow is central, either upward or downward. An easy rotary movement imparted to the content will facilitate segregation and better accomplish settling. Further than this, a cone centrally fed with an overflow of the lighter material, without lateral movement, must needs overflow evenly throughout its circumference, or efficiency will be curtailed in ratio to that part of the circumference which does not overflow, as that represents a segment of the cone not in action.

A liquid body at absolute rest holding in suspension very fine and light solids deposits those solids more slowly than if there is a mild movement, or changing of relative positions. It has not been rare to observe in a tank of exceedingly light slime, after it has stood for days, a top stratum of clear water overlying a slime still in suspension. It is probable that such a condition is due to confined air cells, the aeration having been effected in the crushing, though it is not possible with all methods of crushing.

The flow of liquid through a tank or within confines develops a current of unequal activity, naturally moving faster away from the walls and being the most quiet at the corners.

Interposed baffle plates, when there is a movement not too violent, have a marked effect favourable to precipitation. This is well exemplified in dust-laden gases seen in smelter or roaster settling-chambers. It would be futile to attempt to depict all the complex reactionary currents in a body of liquid through which a current passes, but it is well understood that a current passing a quiet pocket will create 'whirlpool eddy.' If a high-pressure nozzle of water be submerged in a quiet pond, it will set into movement a large mass, increasing in width and decreasing in force with increasing distance from the nozzle. This propelling of water from its place of rest would create a vacuum only that there is an inflow from behind which is compensated by the circular return of the outgoing flow, thus creating a whirlpool eddy on either side of the outflow from the nozzle. This principle of current compensation enters into all conditions of flow of slime or pulp laden liquid, from a Wilfley table top to a large settling vat, and what has here been called a 'whirlpool eddy' may have a vertical axis or its axis may be inclined or horizontal. If baffle plates are of the type called 'over and under,' that is, where the current passes over one baffle and under the next, the axes of the whirlpool eddies generated would be horizontal. On the other hand, if the baffles are staggered right and left with vertical openings between a plate and the opposite side, the eddy axis would be vertical. In one case the precipitating material would pass diametrically through or from a rotating zone, while in the other case precipitation would parallel the axes. It is easy to believe that the latter would be the most highly efficient for settling purposes or for classification."—M. P. BOSS, *Mining and Scientific Press*, Sept. 9, 1911, p. 326. (H. A. W.)

THE ROASTING OF COMPLEX ORES IN GOLD ASSAYING. "The 'roast before fusion' assay seems to be an endemic disease with students in assaying. The results are certainly interesting, but more from the metallurgical than the assaying point of view.

The method has been tried and thrown out with vigour at most smelting works' assay offices, and Mr. Hoare's results confirm the action. As he says, the errors are accentuated with rich ores.

Heavy silver losses were obtained in roasting the Broken Hill zinc concentrate, prior to zinc distillation at the Cockle Creek Smelting Works in New South Wales in 1901 and 1902. If my memory serves me correctly, the ores assayed 20 oz. to 30 oz. of silver per ton, and about 42% of zinc. I carried out roasting experiments on the concentrate, and found losses of silver corresponding to those found in the roasters.

Mr. H. A. Evans and myself, as assayers to the Cockle Creek Works, conducted some experiments on the assay of sulphide ores, and we found that the cleanest and most accurate method was to oxidise the sulphides, arsenides, etc., during the fusion by means of lead oxide, the use of iron nails being discarded: in other words, we used a combined pot roast and fusion. We were led on to the method by having to assay large shipments of Kalgoorlie telluride ores, which our company was purchasing. These ores assayed from 5 oz. to 20 oz. of gold per ton, and contained about 6% of sulpho-tellurides. The charge consisted of:

One assay-ton of ore,
150 gm. of litharge.
Borax,
Soda,
Flour, sufficient with the reducing matter in the ore to give a clean lead button of 60 gm. to 70 gm.
No nails were used, since the iron would have formed a speiss with the tellurides.

By dipping the solidified slag and button in water the lead separated perfectly clean, and required no hammering. This button was then scorified down to 10 gm. or 12 gm., and cupelled near the front of the muffle at the lowest possible temperature, our aim being the production of flowers of litharge on each cupel. As the silver contents were of no value, silver wire for parting was added in the fusion pot. A difference of 2 dwt. was all that was allowed between our assays and those of the West Australian representative on 20 oz. ore.

The Cockle Creek works purchased ores from all over Australia and New Zealand, and the assayer might have in one fire a sample of lead zinc ore or concentrate, another containing 30% of sulphur, another 30% of arsenic, another antimony, beside simpler ores, and by the method evolved we treated all on the same principle. By a preliminary examination, or by panning, we made an estimate of the amount of reducing agents (pyrite, etc.), or oxidising agents (sesquioxide of iron, etc.), and from this estimated the amount of flour, if any, required to produce from the litharge of the charge a button of 50 gm. to 60 gm. for scorification. With heavy ores (20% sulphur and upwards) we took half assay-ton charges and, if necessary, used a little nitre and sand. The lead button was scorified to 10 gm. or 12 gm. before cupellation. By the above method we abolished absolutely the use of iron nails and the accompanying dirty slags, mattes, and speisses. With antimonial and arsenical ores, an excess of sodium carbonate was necessary for fluxing.

We also applied this method as a check and substitute for the scorification assay for silver in the Broken Hill concentrate, containing 60% of lead, 11% zinc, and 30 oz. of silver per ton. Up to this time it had been customary to report the ordinary pot fusion and the scorification assay. With the former a charge of a quarter assay ton was melted with fluxes, using reducing agents and iron nails, and giving a 10 gm. or 12 gm. button for cupellation. In the scorification assay a quarter assay-ton of concentrate was scorified with 75 gm. of granulated lead, or with 50 gm. of granulated lead *plus* 25 gm. of litharge; a little borax was also used. It was recognised that the pot fusion result was about ½ oz. lower than the scorification on 30 oz. concentrate. We found that by our new pot fusion method we got as high results as by scorification. The charge consisted of a quarter assay-ton of concentrate, 30 gm. of red lead, borax, sand, and soda, the latter in slight excess. No iron nails were used. The resultant lead button weighed about 30 gm., and separated well from the slag. This first button was scorified to 10 gm. or 12 gm., and cupelled at a low heat. In this assay we aimed at leaving little or no lead in the slag as silicate, as our experience indicated that lead silicate in the slag caused low silver results, either by reason of it making the slag heavy or by dissolving the silver.

In using this method it is necessary to test the silver prill to see if it is pure."—J. MALCOLM NEWMAN, *Institution of Mining and Metallurgy,—Bulletin No. 84*, Sept. 20, 1911. (A. R.)

THE CONTINUOUS SYSTEM OF CYANIDING IN PACHUCA TANKS.—"The arrangement of a flow of cyanide-pulp through Pachuca vats in agitation, so as to permit a continuous process, instead of alternate filling, agitation, and emptying, has been proposed by various writers within the last two years, and more particularly by A. T. Grothe, agent for the Brown patents on Pachuca vats in Mexico. It was first put into practice, I believe, by M. H. Kuryla at the Esperanza mine in El Oro, Mexico.

The starting of agitation in Pachuca vats after filling may offer no serious difficulties with ores which do not settle rapidly in such tall vats; and the adaptation of the vats to continuous agitation under such conditions may give simply a somewhat more convenient method of treatment and greater agitation-capacity for a given number of vats, because of the saving of time lost in filling and discharging. But in the treatment of pulp which tends to settle rapidly, as in the cyaniding of concentrates or of the whole pulp of ores containing heavy sulphides, the packing of the slime at the bottom may cause much trouble. The use of the radial air-pipe attachments near the top of the cone and of the air-valve outside of the air-lift tube at the bottom may obviate the difficulty to some extent; but the action of the radial air-pipes on the cone-sides is that of a sand-blast, and their continuous use cuts through the vats. Moreover, even when these pipes are used, some pulps will pack tightly below them in the cones. Under such conditions the use of Pachuca vats with intermittent filling and discharging becomes a troublesome process. Time is lost in starting agitation; large quantities of compressed air are wasted; pulp is blown over the vat-tops; and not infrequently it may be necessary to dig out the bottoms of vats by hand. This trouble is the only important one occurring in the use of Pachuca vats; and, since it is caused by intermittent filling, the arrangement of a continuous flow of pulp from vat to vat, kept always full and in agitation, offers a means of avoiding such losses and of making the process much more satisfactory.

For a continuous flow of pulp from vat to vat, the outflow must be equal to the inflow in each vat, that the level may remain constant; and if the inflow is mixed thoroughly with the pulp already in agitation in the vat, as would take place in the central air-lift, then, roughly speaking, that part of the inflowing pulp which flows out of a vat in a short period of time will be to the whole inflow in that time as the quantity of inflowing pulp is to the whole charge. Thus, if a vat contain 100 tons of pulp, and 10 tons flow in during an hour, roughly, one-tenth of the latter, or 1 ton, will flow out to the next vat in the first hour; one-tenth of the ton which flows into the second vat will pass to the third, and so on through the series. The number of vats in the series will, therefore, determine the power to which the fraction is raised for a short period of time. As the process is continuous, obviously these figures are not exact; but for practical purposes we may assume that with a series of vats, the part of the pulp receiving a shorter period of agitation than the average will be balanced by the part receiving a longer period, and that in a series of six vats having a capacity of 600 tons in all, and with 10 tons an hour passing through the system, the pulp would receive 60 hours' agitation. The same vats, if filled, agitated, and discharged by the intermittent system, would give only 40 hours' agitation.

As the thorough mixing of the pulp in the vats takes place in the central air-lift tubes, the overflow connections from one vat to the following should be arranged so as to sample the over-flow of the air-lift. This sampling should make a cut of the whole thickness of the stream of pulp from the air-lift. Failure in this respect would lead to classification in the vat, which would prevent the consistency of the pulp remaining the same through the whole series, and cause a thickening or a thinning that would interfere with the smooth running of the process. Any arrangement for a continuous system should also be provided with by-pass connections, so that any vat or vats may be thrown out of the series when necessary, to allow for changes or repairs in the air valves or interior piping, which are subject to much wear, or for any accident which may occur, such as the dropping of a tool into a vat; otherwise, costly shut-downs and the emptying of the whole series must occur from time to time.

A. T. Grothe has proposed an arrangement for continuous agitation. The overflow connections consist of straight piping at an inclination of 60°, having the intake in each vat at a point midway from the central air lift tube to the vat side at two-thirds the height of the vat, and the discharge into the succeeding vat at the top of the cone. The pipe intake in one vat is joined to the discharge in the next by a piece of rubber hose By-pass arrangements do not seem to have been provided, and the pipe inlets are placed far below the pulp surfaces.

M. H. Kuryla installed continuous agitation in a roughly similar form at the Esperanza mine. The tanks are 45 ft. high and 14 ft. 10 in. in diameter. The pipe connections have their inlets 2 ft. from the 15 in. air-lift tubes and 7 ft. below the vat tops (5 ft. 3 in. to 3 ft. 3 in. below the pulp levels), and their discharges just below the tops of the cones in the succeeding vats. Valves and piping are provided for by-passing any vats in the series at half the height of the vats, and for compressed air to clear out the connecting pipes, in order to prevent their clogging with slime.

Both the above systems are arranged with the inflow openings of the pipe connections in the form of pipe-ends far below the pulp surfaces, which have the defect of not being so placed as to assure a good sampling of the contents of the vats.

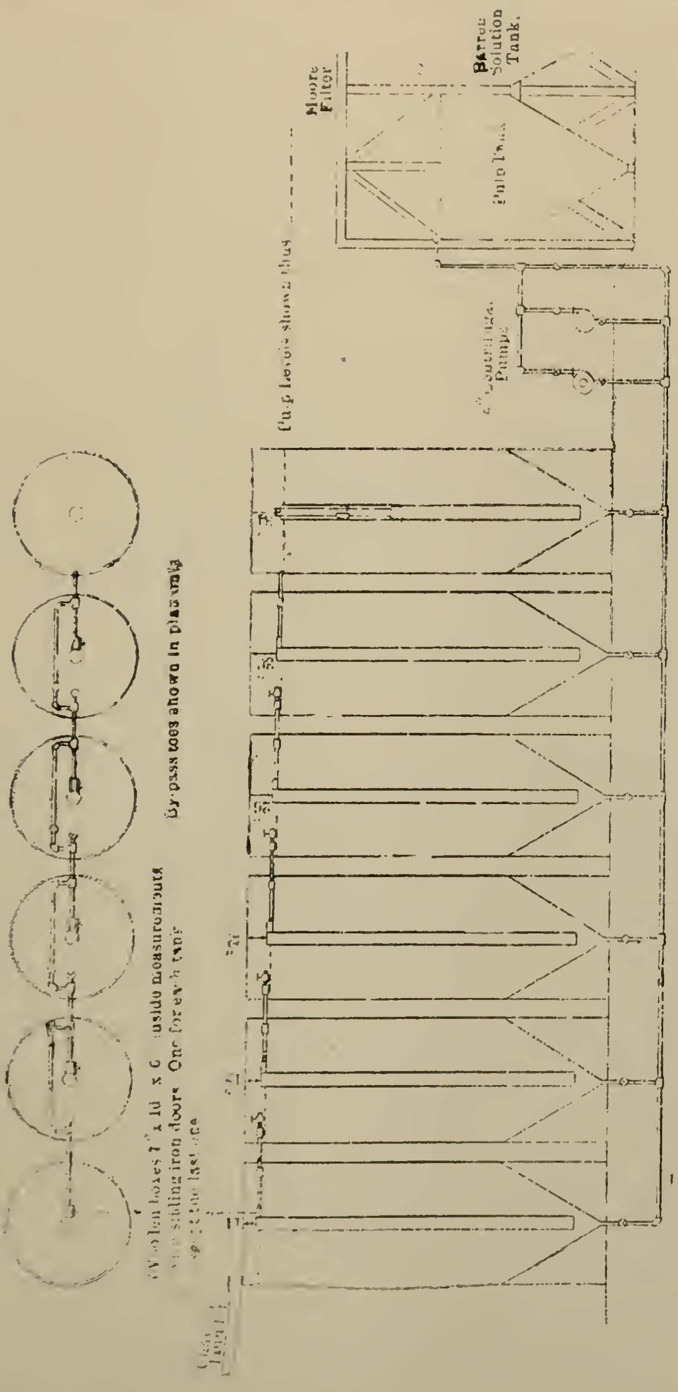
At the Natividad mine, Ixtlan, Oaxaca, Mexico, the 100 ton cyanide plant has been equipped with continuous agitation in a different form.

The ore contains from 5 to 8% of the sulphides pyrite, galena, and blende, and the value is chiefly in gold occurring in the pyrite. The low value of these sulphides when concentrated to 10% insoluble, and the high freight and treatment costs, make inadvisable the shipment of concentrates if a fair extraction can be made by cyaniding them. Tests on cyaniding the concentrates showed an extraction of from 92% to 93% of the gold, and 90% of the total value, if ground fine enough. The mill, as first put into commission in January, 1910, contained no concentration. The ore was as nearly all slimed as practicable in the ordinary way with tube mills, and the overflow from the Dorr classifiers (of which 90 to 95%, passed a 200 mesh sieve) went to Pachuca agitators after thickening in Dorr thickeners. Recently, Johnston vanners have been added, to concentrate the slime overflow from the Dorr classifiers before the pulp passes to the agitation; and the concentrates (90% through 200 mesh) are now returned to the tube mills for regrinding, and circulate from the tables through the tube mills and classifiers back to the tables, until so fine as not to be caught among the concentrates,—after the method of F. C. Brown.

While the pulp from the above dressing, even before the addition of the vanners, was probably as fine as any usually agitated in Pachuca vats, nevertheless, during the filling of the vats, a part of the pulp settled rapidly to the bottom, while the lighter part, containing schistose gangue, showed very little clear solution above it, if left to settle quietly for 6 hours. Though the Pachuca vats used are smaller than those commonly installed in Mexico, being 12 ft. in diameter and 35 ft. in height, the starting of agitation after filling commonly gave such difficulty as to cause several hours' delay, during which, at intervals, the compressed air had to be shut off from other tanks in agitation, in order to raise the pressure to 60 or 100 lb. so as to flow out the settled slime at the bottom of the vat to be started. Frequently, it was necessary to make use of hydraulic force from a pipe line of 500 ft. head (installed originally for another purpose) to force an opening through the bottom of the settled pulp. But for the help of this latter force, the intermittent agitation would have involved the frequent digging out of vats by hand.

In Kuryla's installation at the Esperanza mine, although he used roughly diagonal pipe connections from vat to vat, the last vat was arranged to discharge from the overflow of the central air-lift tube into a box a little below its top, in order to gain head in passing to thickening tanks before filtering. Since the object of the connections should be to sample the stream overflowing from the air-lifts, the box arrangement at Esperanza is nearer the desired form than the submerged pipes, and naturally suggested a similar arrangement for the whole series of vats.

This arrangement was carried out at Natividad for the series of vats, with the exception of the last, as shown in Fig. 1. A drop of 4 in. is used from vat to vat, and the central air-lift tubes are cut down or added to. in order to give this drop. Wooden boxes 7 in. wide, 10 in. long, and 6 in. deep (inside measurement) are fixed against the 15 in. central tubes, with their tops flush with those of the tubes, and 4 in. pipes, placed horizontally, pass from the bottom of each box to the next tank in the series. By-pass pipes, fitted with valves, join each pipe connection with the next in the series, as shown in the plan, and the 4 in. drop from tank to tank is made in the by-passes. Since the level of the inflow pipe in each vat is 2 in. below the tops of the air-lift tube and of the overflow box connected with the outflow pipe, but is slightly above the pulp level in the tank, no part of the pulp entering can pass out of the tank without having first gravitated to the bottom, and risen through the air-lift tube, thoroughly mixed with the whole content of the vat. On the tops of the overflow boxes are sliding iron covers, which open at right angles to the direction of the overflowing stream. The regulation of the flow from vat to vat is done entirely by means of these covers, and



Moore Filter
Barren Solution Tank.
Pulp Tank
4" Centrifugal Pumps
Pulp Levels shown thus
By-pass toes shown in plan only
Wooden boxes 1' x 10 x 6 inside measurements
with sliding iron doors. One for each tank
except the last one

FIG. 1.—Pachuca Tanks arranged for Continuous Operation at the Natividad Mine. Ixtlan, Oaxaca, Mexico. Plan, and Longitudinal Vertical Section.

the valves are used only when it is desired to by-pass vats. The boxes are sufficiently large for the tonnage passing through agitation, so that the covers need be opened only an inch or two, in order to give the required flow. The openings which sample the pulp stream are thus rectangular, with their long axes parallel to the radial overflow at those points; and, while theoretically this is not as correct a shape to sample the stream as would be a sector of a circular ring, in practice it has been found to offer no difficulties, while it is simpler to install and to keep in order in the pulp stream.

As the last vat in the series discharges to the pulp vat of a Moore filter, where an intermittent feed is important, it is arranged with two pairs of sliding doors on the central air-lift tube, so as to permit agitation at various levels. The pulp is drawn off intermittently to the filter through the bottom discharge-opening of the vat, either by hydrostatic pressure in the vat, if full enough, or by 4 in. Butters centrifugal slime pumps.

All the vats retain their bottom discharge connections to the centrifugal pumps, which lift to the filter, so that whenever it is necessary to empty any of them, they may be cut out of the series, to prevent fresh pulp entering, and after sufficient agitation, may be discharged to the filter.

In practice, the continuous system of agitation has removed completely the former difficulties in starting vats, and gives greater capacity or longer agitation for the same tonnage. This is probably the cause of the better extraction noted by Kuryla at Esperanza with continuous agitation. While a better extraction is probably gained at Natividad for the same reason, it has not been readily measureable in practice, because slime concentration for regrinding of the sulphides, as described above, was commenced simultaneously with the continuous agitation and caused a gain in extraction, which obscures the slight gain there might be because of the change in the agitation arrangement. The continuous system of agitation has been in constant service since September, 1910, without having shown appreciable classification, though slight daily differences in the proportions of the pulp from vat to vat are caused by changes in the feed. The flow from vat to vat gives no difficulty, and is regulated by vat-boys at a cost of 62 cents American currency per 24 hr. The by-pass arrangements are very satisfactory, and any one or several vats can be thrown out of the series whenever necessary.

The advantages of the arrangement for continuous agitation at Natividad over those of Grothe and Kuryla seem to be:—

Greater simplicity of installation (the whole change from the intermittent to the continuous system was made in four days); greater accessibility for handling and supervision (all connections are above the pulp levels of the vats and within reach from the main deck on top of the vats); no plugging of the pipe connections can occur, and no compressed air connections are necessary to free them; and as the boxes, by which the flow from vat to vat is regulated, make a good sampling cut of the thoroughly mixed pulp overflowing from the air-lift tubes, no classification is liable to occur, and the proportion of solution to slime remains the same throughout the whole series of agitators." —H. ADAMS, *Transactions of the American Institute of Mining Engineers*, August, 1911, p. 595. (J. A. W.)

THE PHASE RULE IN CYANIDING.—"It is not the purpose of this article to enter upon a scientific discussion of the phase rule as applied to the reactions occurring in the cyanide process, but rather to point in a general way to its bearing upon the conduct of operations in practical work. The object will be served by a consideration of the reactions between gold and an alkali cyanide if it is borne in mind that the influences at work in the treatment of silver ores by cyanide are governed by the same laws.

Gold in ores when in contact with cyanide solution dissolves at a rate which decreases as the solution increases in precious metal content and becomes reduced in free cyanide. This decreasing rate of dissolution, analogous to the diminishing rate at which a salt dissolves in water as the point of saturation is approached, is well known, but if viewed in the light of the phase rule the reason for it is made clearer and the application of cyanide solutions to ores might in many cases be made more efficient.

Bearing in mind that dissolution will proceed until an equilibrium is reached between the undissolved metal, the metal in the solution and the free cyanide, and that as this equilibrium is approached the velocity of the reaction becomes reduced, it is easy to account for the relatively slight increase in extraction secured by prolonging treatment, as well as by the addition of fresh cyanide to solutions after being for some time in contact with the ores; the latter by reason of the fact that although the free cyanide is increased thereby, the alkali aurocyanide remaining in the solution continues to exert an important influence on the velocity of dissolution. Were the alkali aurocyanide removed from the field of the reaction, the velocity of the dissolution would be increased by reason of the removal of that phase which, from its tendency to undergo the reverse reaction, exerts a detrimental effect by keeping the whole system near the point of equilibrium. An immeasurably small concentration of aurous cyanide would suffice to bring the dissolution to a standstill were it not removed as rapidly as formed, by dissolution in the excess of alkali cyanide present with formation of the complex alkali aurocyanide. Although this removal of aurous cyanide is what renders the cyanide process possible, the fact should not be lost sight of that we have in the formation of alkali aurocyanide only a system in which the permissible concentration is relatively large, and that while the tendency to the reverse reaction is, from a commercial standpoint, often negligible, there are nevertheless instances in which it should be given serious consideration.

A discussion of all the influences affecting these reactions, including those brought about as a result of working with such impure substances as ores, is unnecessary, and nothing would be gained by introducing complications which would only befog a clear view of the subject. The application of the phase rule points to the separation of ore from solution when the velocity of dissolution becomes low and the application of fresh solution in which the concentration of alkali aurocyanide is small, and to the eventual attainment of a continuous cyanide process based on the principle of counter currents, in which the richest solution is in contact with the richest ore and the solution most free from alkali aurocyanide is in contact with the ore after the dissolution of the greater portion of the values." —H. B. LOWDEN, *Metallurgical and Chemical Engineering*, July, 1911. —*Mining and Metallurgical Journal*, Aug., 1911, p. 35. (H. A. W.)

VACUUM FILTER PRACTICE IN WESTERN AUSTRALIA.—"The stationary-frame vacuum filter process was introduced into Western Australia four years ago, and has since been developed and improved until it now forms one of the chief methods of slime treatment on these fields. The object of reproducing this article, is to point out, particularly, the novel features of the plants in operation in that country, so as to make it comparable with American practice and methods.

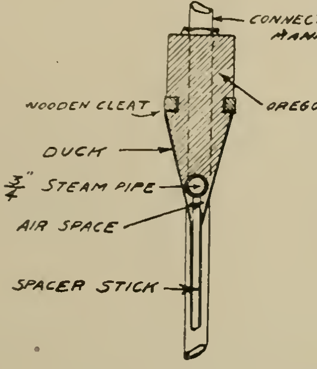
The material treated in vacuum filter plants varies, from the finest oxidized slime to an all-slimed product from sulphide ore. No slime of any grade has yet been tried that could not be economically handled.

The worst filtering slime yet tested was an oxidized clay containing no fine sand, obtained from the Never Never mine (near Southern Cross). This slime gave a cake only ⅜ in. thick, containing 3·46 lb. of dry slime per square foot after 40 minutes of vacuum. It could be successfully washed and discharged. Average oxidized slime will form in 25 minutes a cake 1¼ in. thick, carrying 8 lb. of dry slime per square foot. An all-slimed sulphide product will, in the same time, form a cake 2½ in. thick, carrying 18 lb. of dry slime per square foot.

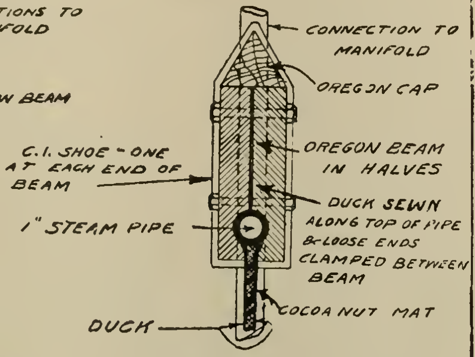
It is always advisable to make experiments with a single filter-frame before a plant is erected. The appearance of slime as regards its cake forming power is often deceptive; thus, Rand slime is considered a perfect slime free from sand and is practically a pure clay, but with this slime a 3 in. cake can be readily formed. The slime that contains most sand is not always the most easily formed into a cake. It appears that a well-sized slime, *i.e.*, one containing particles of approximately equal size, best forms a cake, the reason being that the interstitial space through which the solution can pass, is greater than in a mass of closely packed particles of variable sizes. This subject has been carefully studied in connection with sand leaching.

The cakes from oxidized slime will, as a rule, contain 30 to 33% of moisture, while an all-slimed sulphide product will contain from 23 to 26%.

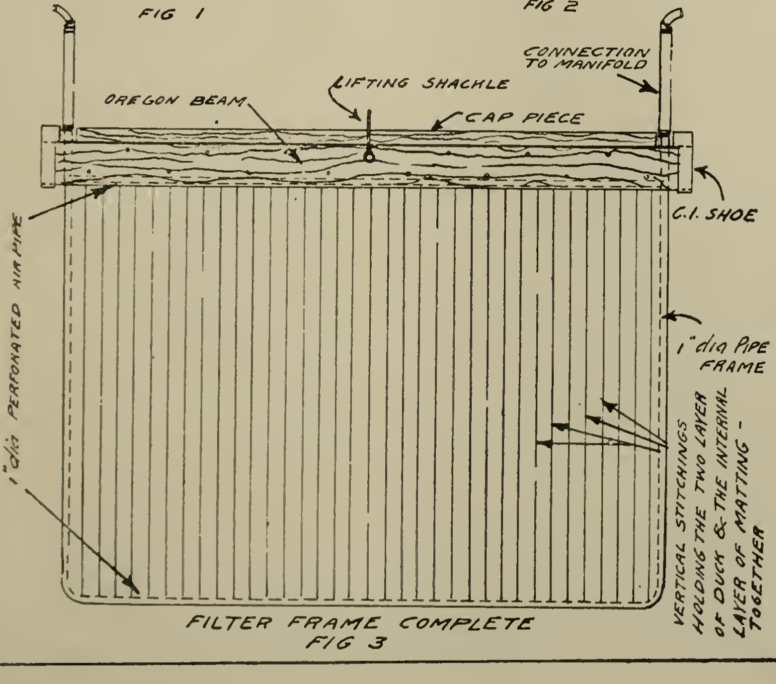
The Filter Frames.—The present filter-frame is the result of considerable experimental work. The



ORIGINAL FILTER BEAM
FIG 1


CONNECTION TO MANIFOLD
OREGON CAP
C.I. SHOE - ONE AT EACH END OF BEAM
OREGON BEAM IN HALVES
DUCK SEWN ALONG TOP OF PIPE & LOOSE ENDS CLAMPED BETWEEN BEAM
1" STEAM PIPE
COCOA NUT MAT
DUCK

PRESENT FILTER BEAM
FIG 2


CONNECTION TO MANIFOLD
LIFTING SHACKLE
OREGON BEAM
CAP PIECE
C.I. SHOE
1" dia PERFORATED AIR PIPE
1" dia PIPE FRAME
VERTICAL STITCHINGS HOLDING THE TWO LAYER OF DUCK & THE INTERNAL LAYER OF MATTING - TOGETHER

FILTER FRAME COMPLETE
FIG 3

trouble with all the earlier frames was that the cakes cracked on exposure and consequently fell off. Practically all frames are now built of 1 in. steam pipe, and are covered with 16 oz. duck, with cocoa-mat between.

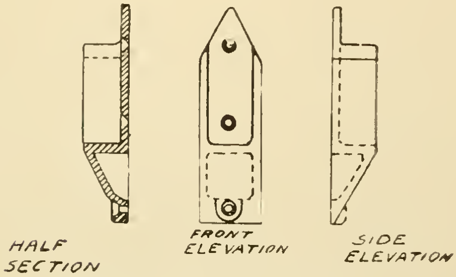
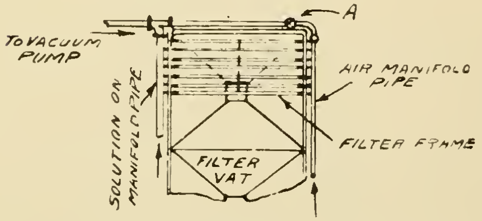


FIG 4 CAST IRON SUPPORT FOR FILTER FRAMES



PLAN
FIG 5

The filter-cloths are sewn with vertical seams 3 in. apart, this spacing having been found to be the best. Undoubtedly a great part of the filtered solution passes through the stitch holes instead of the cloth, and it has been found that bags with seams 6 in. apart do not take cake so quickly as those with 3 in. spacing. Bags with seams only 1 in. apart have been tried but these did not form the cake so quickly as the 3 in., the reason being that the close stitching drew the cloth so tightly to and compressed the coco mat so that very little space was left in the mat for the solution to pass through.

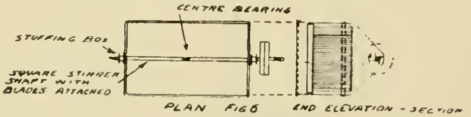


FIG 6

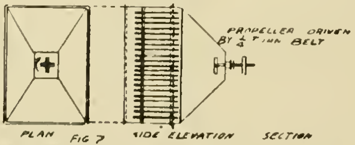


FIG 7

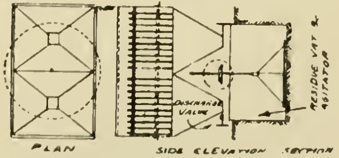


FIG 8

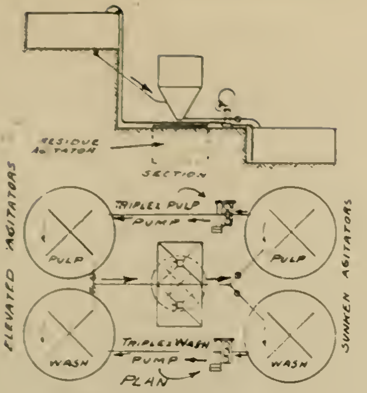
The weight of the pipe, frame, and cake is hung from two supporting beams of special section. The detail of these beams is important. The beam at first used (Fig. 1) had many disadvantages. The wooden cleats were not strong enough to hold the cloth and the weight of cake. The cloth also had no support where it joined the beam, thus causing cracks to form there on exposure. It is also obvious that if the cloth were allowed to stretch, the cake would crack at the top. Another difficulty was that the cake formed on the cloth where it was stretched over the beam and could not afterwards be dislodged by the air discharge, thus necessitating constant hand cleanings. The air pressure also caused rapid wear of the cloths, for when the air was turned on, a great strain was put on the vertical seams at the top of the bag, and it was only a matter of a few changes before holes appeared, allowing the slime to leak into the filtered solutions.

All these disadvantages have been overcome in the improved frame. (Fig. 2). In this frame the support of the cloth is satisfactory, as it is clamped securely on the split beam. No cloth is exposed on the outside of the beam, consequently no cake can form on it. The shape of the edge "A" is designed so that it just presses on the cloth, but not so tightly as to prevent the passage of compressed air. The vertical seams come just up to this ledge, and it is, therefore, impossible for them to rip.

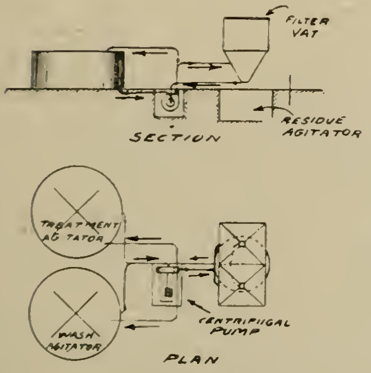
Frames are now usually made of a uniform size (Fig. 3) of 9 ft. wide × 4 ft. 9 in. deep; this permits the use of a covering of duck, 72 in. wide. The capacity of a plant is adjusted not by the size but by the number of the frames employed. Taking into account the handling of the frames for renewals, and the time occupied in filling and discharging the filter vats, this size of frame has been proved, from practical experience, to be the most efficient; deeper frames take considerably longer to discharge, owing to the choking of the cakes.

If a plant is working on slime that will form a cake less than 1½ in. thick, piping of smaller size than 1 in. may be used for building the frames; but with cakes above this weight nothing less than 1 in. steam pipe will stand the shock of the falling cakes. Pipes are liable to break where they join the beam, owing to the leverage exerted on them when the falling cakes spread the frames at the bottom. It is advisable not to fix the frames in the filter vat too securely, as a slight motion of the beams decreases the shock to the frames. Figures 2, 3 and 4 show the latest design for holding the frames in position. The cast-iron shoes are securely bolted to the frames and have a good footing in the cast-iron sockets bolted to the filter vat sides. This arrangement allows for slight rocking of the frames; it will withstand considerable wear and tear, and the frames will always hang plumb.

Various devices have been tried for obtaining the necessary spaces for the flow of the solution between the cloths of the frame. Wooden laths were first used, one being placed between each 3 in. seam; these were fairly satisfactory, but they do not give the support to the cloths that the coco mat does and the cakes are, therefore, more liable to crack during exposure. Ripple iron, *i.e.*, galvanized iron with small corrugations, as used at the Waihi mine, New Zealand, was also tried, but as this had to be used without vertical seams, trouble was experienced in discharging the cakes: when the air was turned on the cake was stretched but not discharged. With very free filtering slime, *i.e.*, slime that will form a cake of 2 in. or more in 20 minutes, it has been found that the cakes form thicker at the bottom than the top. The reason for this has been traced to the fact that the filter mat is not sufficiently porous to allow

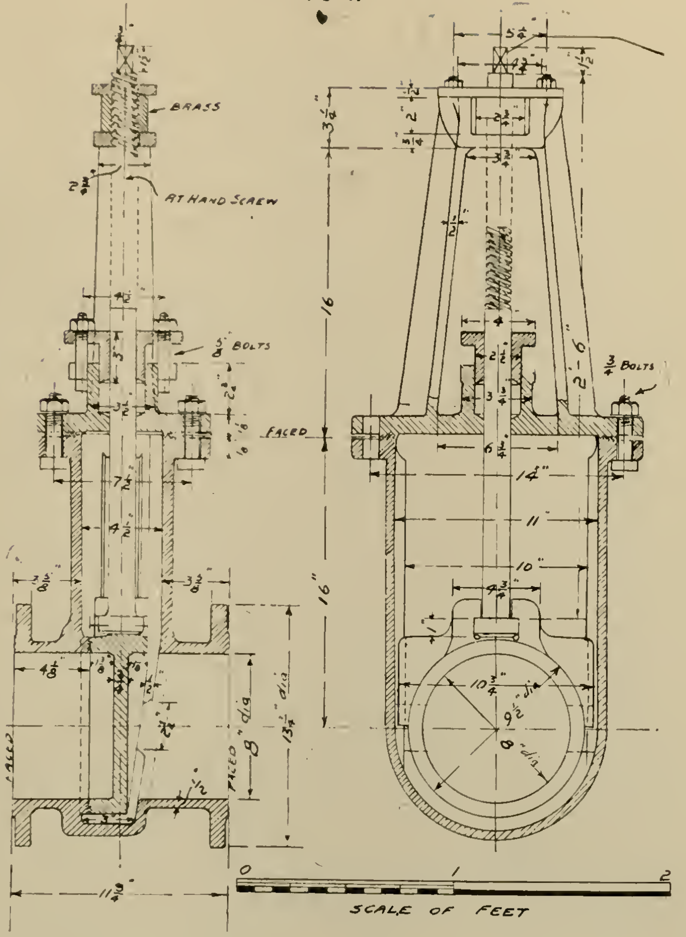


GRAVITY SYSTEM
FIG 9



PUMPING SYSTEM
FIG 10

FIG 11



the solution to be drawn off as quickly as it passes through the cloth. This difficulty has been obviated by using the top pipe of the frame, as well as the bottom one, for drawing off the solution. Fig. 5 illustrates this method, which may be described briefly, as follows: In taking cake or washing, both the solution and air-cocks on the frames, as well as the valve "A" are left open, thus enabling each cock to draw from approximately half a frame. When the cakes are ready to discharge, valve "A" and the air valves on the frames are closed; compressed air is then admitted to the other end of the air manifold, and the air cocks are opened one at a time to discharge the cakes. The same result could be obtained by using larger pipes for the frames, and a larger solution manifold, but this would add needless weight to the frames. This arrangement is also useful as an addition to an old plant, where by reason of the internal deposition of lime, the pipes become too small for the flow of solution.

Filter Vat Construction.—The chief point to be considered in the design of filter vats is the method of discharge. It was obvious from the beginning that in this country sufficient water could not be obtained to discharge them by sluicing with water pressure, as practiced in America. It was required to discharge the residue into the residue dam in the form of a pulp as thick as could be handled. This has been effected by mixing devices of different designs placed in the bottom of the filter vats.

Fig. 6 shows the first type that was used, consisting of a horizontal shaft with attached blades driven through two stuffing boxes. This arrangement was found to mix the residue but slowly, requiring over half an hour for discharge; the wear in the stuffing boxes was excessive, and the V-shaped bottom retained a quantity of dead pulp and wash, which increased pumping expenses.

Fig. 7 shows an arrangemant suitable for small plants where a vat with a single hopper is of sufficient capacity. The mixer blades take the form of an impeller, which is driven by a quarter-turn belt.

Fig. 8 shows the most successful type, in which the cakes are dumped through discharge doors into a separate mixing agitator below. This allows of very quick discharge—less than ten minutes—as the mixing can be completed after the discharge doors are shut, and the filter vat is at work on the next charge. This type is always adopted where the water from the Goldfields water supply is used for sluicing purposes, since water used exclusively for sluicing and not entering the treatment plant, is supplied at a cheaper rate than that used for general purposes.

Filling Filter-Vats.—It is essential that the filter vats should be filled and emptied quickly. Two methods have been employed :

1. Filling by a small pump and gravitation (Fig. 9). In this arrangement the pulp and wash are elevated to storage vats by small continuously-running pumps, and the filter vats are rapidly filled from these through large pipes, as required. The excess pulp and wash from the filter vats is run out into two storage vats situated below, and then elevated to the upper vats.

2. Filling by a large pump running intermittently (Fig. 10). In this arrangement only one set of storage vats is required. These are placed on the same level as the filter vat, from and to which the pulp and wash is pumped as required.

Both methods have their advantages. The capital cost in each case is much the same, for while with the gravitation plant extra storage vats are necessary, the pumping plant requires a more powerful pump, and larger piping and valves. As regards power cost, there appears to be little difference, since the gravitation plant requires power for running two small pumps continuously on a high lift, and for stirring two extra storage agitators, whereas the pumping plant requires only sufficient power to drive a large pump on a low lift.

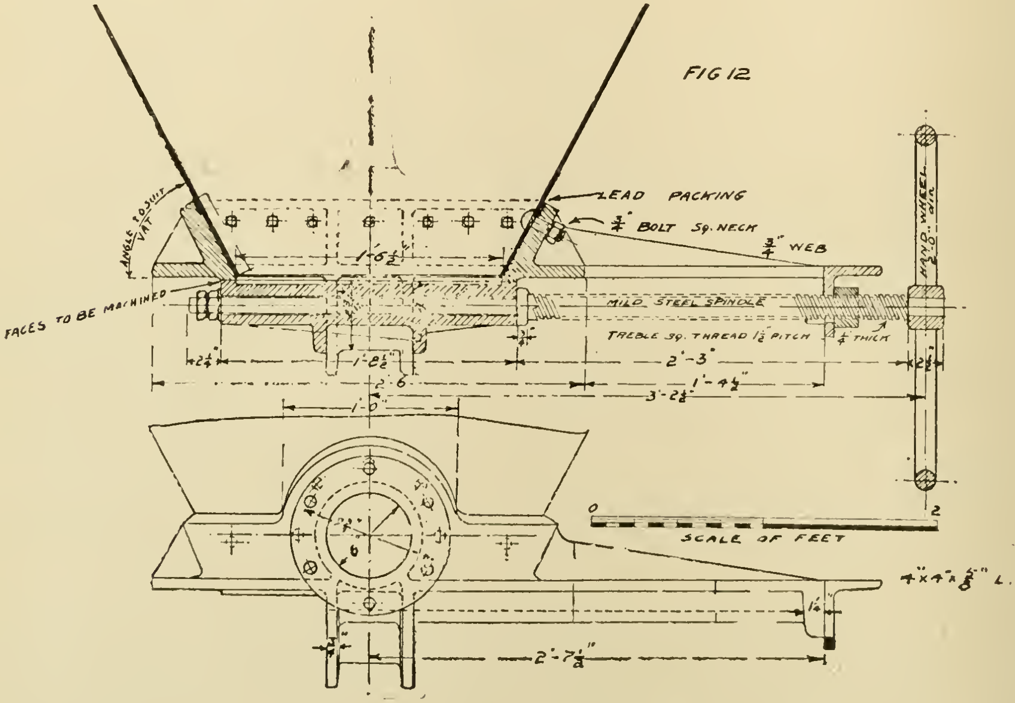


FIG 12

For a mine with a large power plant, or one buying power from a central generating station, the sudden addition of a big load, such as a large pump, is of little account ; but for a mine with a small power plant of its own, the smaller but continuous load is the more convenient. As a rule it has been found that where only one filter vat is required, the gravitation filling is good, but where two or more filter vats are necessary the pumping plant is undoubtedly better.

Style of Pumps used.—For a high lift, triplex plunger pumps are most efficient, and these call for no remark. For a large capacity on a low lift the centrifugal pump is best ; the pumps most in use are of the Kershaw type. The casings of these pumps are made in halves with interval hard liners, which, with other working parts, are easily renewable. The cast-steel ported runner is fitted with cast-steel renewable shoes, and is carried on a heavy mild-steel shaft. The shaft is run in a specially sealed four-part bearing with wedge adjustable brasses, inside wearing-ring and nipple. A stuffing box and gland for grease packing are also provided. The runner shaft is connected by a muff coupling to a short shaft carrying the fast and loose pulleys between two outer standards having heavily flanged gunmetal bearings and loose thrust collars. All bearings are provided with water circulating pipes and cocks to supply clean water under pressure to keep the bearings free from sand and grit. Stauffer lubricators are also fitted. The loose pully is brass-bushed, and fitted with a Stauffer lubricator ; it is run on a cast-iron sleeve, which is secured to the spindle, and which is the full width of the pulley face.

Wet vacuum-pumps without intermediate receiver have been found quite satisfactory and capable of maintaining a vacuum up to 27 in. These pumps are usually of 8½ in. diameter, single cylinder, 10-in. stroke, double-acting up to 40 revolutions per minute, according to the number of frames to which they are connected. They are fitted with vacuum and air vessels to promote uniform action. The pistons are of the trunk type, ordinary square packing being used.

Valves and Doors.—For filling and emptying the filter vats 6 in., 8 in. and 10 in. pipes are used; special and inexpensive valves have been designed as shown in Fig. 11. Where it is possible to do so, these quick-opening valves are arranged so that they may be operated from the working platform by means of extension spindles and hand wheels.

Fig. 12 shows detailed elevations of the type of discharge door shown diagrammatically in Fig. 8. It will be noticed that incorporated in the bottom casting is a flanged discharge for emptying the vat.

Clarifying Solutions.—In most cases it has been possible to make use of old filter presses for clarifying solutions: the solutions are, as a rule, very clear and a by-pass is arranged so that in the case of a leaky bag, the delivery of the vacuum-pump can be turned back into the filter-vat. Where filter presses have not been available, filter-bag frames, placed in an elevated gold-solution vat, form an inexpensive and effective clarifier. The frames are similar to those used in the filter vats, but are long and narrow. The suction is obtained by the fall to the zinc boxes.

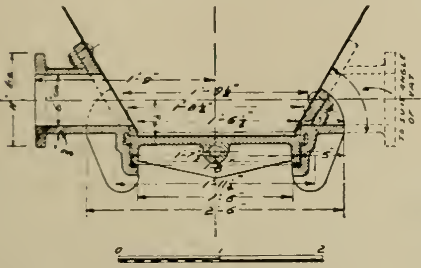


FIG 12a

The frames are usually cleaned once in each shift by closing the delivery to the extractor boxes, and inflating the bags with air; this causes any deposited slime (which is very tough) to peel off into the bottom of the vat. After the frames have been cleaned, the delivery is kept shut off as long as possible to allow the slime to settle. The settled slime in the bottom of the vat is sluiced out once a week into one of the treatment agitators.

Washing of the Cakes.—The time of economical washing varies greatly with the nature of the slime, and can only be definitely determined by taking progress assays from the discharge of the vacuum pump. This subject has been already discussed in an earlier number of the *Journal.* The time occupied in washing is always greater than that required for taking cake. It has been found that a low-grade slime (under 8s. per ton) obtains a good wash when the amount of solution sent to the extractor boxes is equal to about one-tenth more than that contained in the original pulp. As in all other filter processes, it is essential that the wash should be kept low in gold; to do this the filter vat must be quite free from pulp when the wash is put in, and this can be easily effected by spraying down the sides of the vat with spent solution as the pulp is emptied out. If this be done, the wash will not gain more in gold by contact with the vat and cakes, than can be counteracted by dilution with the make up of barren solution and fresh water."—T. B. Stevens and W. R. Degenhardt, *Journal of Chamber of Mines, Kalgoorlie Mines and Methods*, June 1911, pp. 237-240. (W. R. D.)

Improved Method of Securing Battery Posts.—"An arrangement for securing battery posts to the concrete foundation block is described by F. W. Sewell, in the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 3. The arrangement is used at the mills of the Guanajuato Cons. M. & M. Company and the Zacatecas M. & M. Company.

A cast-iron shoe is bolted to the concrete block, and the shoe, in turn, receives the battery post and holds it in place by suitable bolts and brackets. The shoe weighs 860 lb. and has a bearing surface on the concrete of 4 ft. 6 in. × 1 ft. It has a cavity cast in it 2 ft. long × 10 in. wide and 6 in. deep, into which the battery post fits.

The shoe is secured to the concrete block by two 2½ in. rods, each 5 ft. long, threaded at both ends. The threaded portions are of enlarged cross-section, so that the diameter of the base of the thread is not less than that of the remainder of the rod. The lower ends of the rods pass through broad, square, cast-iron washers embedded in the concrete, and are secured thereto by square nuts. Hexagonal nuts or lock nuts are used on the top end to hold down the shoe.

The battery post is held securely in the shoe by means of 1½ in. bolts 2 ft. 7 in. long, connecting brackets on the side of the post with the cast-iron shoe. At each end of the shoe are cast suitable projections which are bored vertically to receive the long bolts mentioned. The two side brackets, which weigh about 100 lb. each, are recessed into the 12 in. faces of the post and held in place by five ⅞ in. rods passing through the post from one bracket to the other. These rods are threaded at both ends, and lock nuts hold the brackets firmly in position. Passing vertically downward from each bracket to its respective end of the shoe are two rods, mentioned above, with square nuts to attach them to the shoe and hexagonal nuts to attach them to the bracket. The lowest part of the bracket is 1 ft. 2 in. above the top of the shoe.

In erecting the battery, the cavity in the shoe is half filled with oil, and the foot of the post is cut slightly larger than the cavity, the point being cut so that it will just enter the cavity. When the post is forced into place by means of tightening the connecting rods, aided by blows from heavy hammers on top, the oil is forced into the end of the post. This insures protection from rotting due to water. The entire post is then oiled occasionally until no more oil is absorbed, when it is painted."—F. W. Sewell.—*Metallurgical and Chemical Engineering*, Sept., 1911, p. 475. (H. A. W.)

Calculations in Ore Concentration.—"The size of American concentrating plants, especially of those treating copper ore, has very greatly increased in recent years. When we consider the enormous sums of money represented by the ore treated in these plants and how great might be the financial loss due to slight errors in mill operation, the necessity of keeping close watch on the work becomes apparent.

It is not possible to so thoroughly control a plant of this kind as it is to regulate, say, an electric central station, where a number of ammeters and voltmeters will indicate precisely the operation of the plant. The best indicators which the concentrating man has at his command are screen tests and assays, but a great deal also depends on his ability properly to interpret the readings of these indicators. To do this successfully requires the application of some mathematics, and it is my desire to show in this paper how much can be done in this direction by simple algebra. I do not claim that the relations that I shall develop are new, for others have already applied algebra to problems of ore dressing; but the methods used and results obtained do not seem to have become well known, probably because the text books give but little, if anything, on the subject. It is my hope that by communicating the information to this Association, I may save others the trouble of working out such problems for themselves.

The Value of the Assays.—Let us consider in the first place the information which the assays of head, concentrate and tailing will give us regarding the saving which a concentrator is making. Let x equal the weight of the ore put into the mill, with y and z the respective weights of concentrate and tailing produced from the x quantity of ore. The assays of ore, concentrate, and tailing may be designated by a, b and c, respectively.

Then the two facts, that the weight of ore equals the combined weights of concentrate and tailing, and the amount of copper, or other valuable constituent, in the ore equals the combined amount of copper in concentrate and tailing, are expressed by the two equations,

$$x=y+z \qquad (1)$$

$$ax=by+cz \qquad (2)$$

The absolute quantity of ore milled is immaterial to the calculation of the saving, inasmuch as these equations apply equally well to large or small amounts of ore passing through the mill. Therefore we may consider the treatment of 1 ton of ore, and make $x=1$. Our equations then become,

$$1=y+z \qquad (1a)$$

$$a=by+cz \qquad (2a)$$

If we eliminate z between these equations we get,

$$y=\frac{a-c}{b-c} \qquad (3)$$

Now y, according to our definition, is the weight of concentrate produced from 1 ton of ore. We will get the saving by dividing the amount of copper in the concentrate by the amount of copper in the 1 ton of ore, or the saving which may be represented by S, is expressed by the equation,

$$S=\frac{by}{a}$$

Substituting this deduced value of y in equation (3) we get the following equation for the saving,

$$S=\frac{b(a-c)}{a(b-c)} \qquad (4)$$

This formula contains no weights and it is clear, therefore, that the saving of a concentrating plant may be calculated from the assays of the ore, concentrate and tailing without knowing anything about the quantities used or obtained. For example, a concentrator operating on 3% copper ore may produce concentrate carrying 17% *Cu* and tailing with ·7% *Cu*. What saving has been made? Substituting the values in formula (4) we find approximately 80%.

In plants where ore and concentrate are weighed (the weight of the tailing being taken by difference), the results of this formula can be checked against the saving calculated by other methods. There are three more possibilities, which in our terms are as follows:

$$S=\frac{by}{ax}=\frac{\text{copper in concentrate}}{\text{copper in ore}} \qquad (5)$$

$$S=\frac{by}{by+cz}=\frac{\text{copper in concentrate}}{\text{copper in concentrate and tail}} \qquad (6)$$

$$S=\frac{ax-cz}{ax}=\frac{\text{copper in ore}-\text{copper in tail}}{\text{copper in ore}} \qquad (7)$$

If the weights and assays are all correct, these four methods should check. In practice, of course, they do not check exactly. An example of the results obtained by the four methods in a certain copper concentrator for several consecutive months is as follows:—

Method of Calculation.	Saving.		
(1) Assays only	77·48	78·59	80·33
(2) Weights, and assays of ore and concentrate ...	78·09	80·73	79·00
(3) Weights, and assays of tailing and concentrate ...	77·64	79·11	80·02
(4) Weights, and assays of ore and tailing	77·52	78·69	80·27

These four methods of calculation are not independent of each other. If the amount of copper in the ore as calculated from weight and assays equals exactly the amount of copper in concentrate and tailing as calculated from their weights and assays, all four methods will give the same results; but if there is a discrepancy, the methods will give discordant results. In the latter case it will depend entirely on specific conditions which method is the most trustworthy; whether, for example, one sample is more reliable than another, or whether the weights can be relied upon, etc.

In the derivation of the formula for the saving calculated from assays only, equation (3) has special interest.

If we recall that y means that fraction of the ore which makes up the concentrate, we see that it is equivalent to the 'rate of concentration.' For instance, in the example of an ore carrying 3% copper, and giving concentrate of 17% and tailing of ·7% copper, we would find for y the value ·141, or expressed as a common fraction $\frac{1}{7{\cdot}09}$, which is equivalent to the statement that 7·09 tons of ore have produced 1 ton of concentrate, or that the ore has been concentrated 7·09 into one.

This equation is interesting from another standpoint. The rate of concentration is expressed by the assays of the ore, concentrate, and tailing; therefore it will hold true for copper, gold, or silver assay, or any other set of chemical determinations. All of these determinations cannot be made with equal exactness, and when we have a practical problem of determining the rate of concentration from a set of assays, such determinations will be chosen as are most reliable.

On the other hand, the fact that the rate of concentration can be calculated in so many ways, permits us to eliminate y and compare the different assays directly. If, for example, a, b, c, represent assays for gold; a_2, b_2, c_2, assays for sulphur; and a_3, b_3, c_3, assays for iron or any other constituents in ore, concentrate, and tailing, respectively, the relations will hold true, thus:

$$\frac{a-c}{b-c}=\frac{a_2-c_2}{b_2-c_2}=\frac{a_3-c_3}{b_3-c_3}, \text{ etc.} \qquad (5)$$

I have not at hand any assays to show the application of this equation, but it is clear that it will allow the calculation if a complete analysis of the ore and concentrate and only one assay of the tailing be given.

Value of Screen Tests.—Thus far we have been considering only the informatien which the assays give in solving problems that come up in ore concentration. Let us now introduce the sizing screen and see what additional information we can gain by its use.

Take for example a problem of this kind: A certain concentrator disposes of its tailing in the following manner: After classification in cone classifiers, the spigot discharge contains the coarse sand with a little water, and is used to build a dam across the mouth of a gulch adjoining the tailing end of the plant. The overflow is conducted into the basin thus formed, where the suspended solid matter settles, liberating clear water which is again used in concentrating. The question arises, what percentage of the tailing goes to the dam and what percentage to the pond. This problem can easily be solved by making some sizing tests on the general tailing and the two products of classification mentioned.

Let the total quantity of the tailing be represented by x: that part going to the dam by y, and the portion going to the pond by z. Then we have the equation,

$$x=y+z \qquad (6)$$

If we screen the samples of each material on, say, a 100-mesh screen, we will have a certain fraction of each sample remaining on the screen which we will call α_{100}, β_{100}, and γ_{100}. If we screen a general tailing sample we will evidently get the same percentage of 100-mesh material as if we split this sample into the part which builds the dam and the part which feeds the pond, screened these two samples separately and combined the corresponding sizes. This fact is expressed by the equation,

$$\alpha_{100}x=\beta_{100}y+\gamma_{100}z \qquad (7)$$

Since these two equations in no way depend on the actual quantity of tailing represented by x, we can make $x=1$. Then,

$$1=y+z \qquad (6a)$$

$$\alpha_{100}=\beta_{100}y+\gamma_{100}z \qquad (7a)$$

or by the elimination of z,

$$y=\frac{\alpha_{100}-\gamma_{100}}{\beta_{100}-\gamma_{100}} \qquad (8)$$

This equation naturally holds true for any other screen size, and if we screened on a 200-mesh screen we would have,

$$y=\frac{\alpha_{200}-\gamma_{200}}{\beta_{200}-\gamma_{200}} \qquad (8a)$$

The screen to be used in a given case depends on the material to be screened, the principal point to be observed being the selection of the screen which will give the most nearly correct results. It is to be understood that screening does not give exact results, for no screen is absolutely uniform; but those who have made wet screen tests often enough to become proficient will admit that screening on fine sizes, say from 100 mesh down, is fairly accurate. Although the error in screening expressed in percentages may be greater than the error in assaying, there are cases in which the results based on screen tests are much more reliable.

In the case under consideration the screen tests were as follows: Percentage on 100 mesh, general tailing, 44·4; tailing to dam, 75·2; tailing to pond, 3·1.

Substituting the screen results on 100 mesh screen in our formula, we find that 57·3% went to the dam and the balance of 46·7% to the pond.

This problem could have been solved from the assays only. The same set of samples used for the screen test was assayed with the following results:

		Cu Per Cent.
General tailing	... ...	·85
Tailing to dam	... ..	·72
Tailing to pond	... ...	1·16

Our formula (3) will solve this problem. Although we have deduced this formula for relations existing between ore, concentrate, and tailing, it holds for any case in which one product is split into two other products, or where two different products are combined into one. I must emphasise the fact that the two products into which one is split, or which are combined into one, must be different, for if they are the same the problem cannot be solved and if they are nearly alike the error in the result will be great. Therefore if there is but little difference in the assays of the products, screening will give better results, but if the material is of practically the same size the assays should be the basis of the calculation. Returning now to dam problem, let a represent the assay of the general tailing, b the assay of the coarse tailing, and c the assay of the fine tailing; y will represent the fraction of the tailing going to the dam. Using these assays in the formula we get as a result,

$$y=70{\cdot}5\%$$

This is decidedly different from the figure 57·3% which we obtained from the screen tests, and is evidently not reliable. This is due to the fact that the difference between the assays of the three products are so small that they are not a safe basis of calculation. This, then, is a case where screen tests give much more reliable results than assays.

That the assays are really conformable with the screen tests can be shown by calculating the assay of the general tailing from the assays of the coarse tailing (dam) and the fine tailing (pond) making use of the proportion between the two found by screening. The calculated general tailing assay, according to this, would be,

$$a=bx+cy$$

$$a={\cdot}91\%\ Cu$$

This checks the actual assay within the limits of the ordinary cyanide assay for copper, and shows that the results by assaying have not contradicted the results obtained by screening.

But we are able to solve this problem in still another way, by considering the percentage of water carried by each one of the materials in question. We have the equation for the dry pulp,

$$a=b+c \qquad (9)$$

If we call the weights of the wet pulps A, B, and C, respectively, the following equation is also true,

$$A=B+C \qquad (10)$$

If α, β, γ, represent the percentages of solid matter in A, B, and C, respectively, we will have the following equations:

$$a=A\alpha \qquad A=\frac{a}{\alpha} \qquad (11)$$

$$b=B\beta \qquad B=\frac{b}{\beta} \qquad (12)$$

$$c=C\gamma \qquad C=\frac{c}{\gamma} \qquad (13)$$

Substituting these values in equation (10) it follows,

$$\frac{a}{\alpha}=\frac{b}{\beta}+\frac{c}{\gamma} \qquad (14)$$

Since it is again unessential what quantities we consider, make $a=1$. Then,

$$1=b+c \quad c=1-b \qquad (15)$$

$$\frac{1}{\alpha}=\frac{b}{\beta}+\frac{c}{\gamma} \qquad (16)$$

By eliminating c in the two equations

$$b=\frac{\beta(\alpha-\gamma)}{\alpha(\beta-\gamma)} \qquad (17)$$

The moisture samples taken gave the following results,

$\alpha=13{\cdot}1\%$ solids

$\beta=42{\cdot}5\%$ solids

$\gamma=\ 6{\cdot}9\%$ solids

By substituting these figures in our equation

$$y=56{\cdot}5\%$$

which represents the percentage of dry tailing that would go to the dam. This checks the result obtained from the screen tests, which was 57·3%.

It seldom happens, however, that problems in the distribution of pulp are as easy as this one. If three products are combined into one, or one is split into three, the calculations are more complicated. For example, a concentrator produces three final tailing products, by the following treatment: the coarsest ore is treated on Wilfley tables, the next finer on one set of vanners, and the finest on still another group of vanners. The tailings from all three sets of machines are combined into general tailing. The question arises, what percentage of the general tailing originates from the Wilfley tables, and what percentage from each of the vanners. This question could be solved by screening the different tailing samples. Let us call,

a_{30} the fraction of general tailing remaining on 30 mesh screen.

b_{30} the fraction of Wilfey tailing remaining on 30 mesh screen.

c_{30} the fraction of 1st vanner tailing remaining on 30 mesh screen.

d_{30} the fraction of 2nd vanner tailing remaining on 30 mesh screen.

a_{200} the fraction of general tailing on 200 mesh screen, passing 30 mesh.

b_{200} the fraction of Wilfley tailing on 200 mesh screen, passing 30 mesh.

c_{200} the fraction of 1st vanner tailing on 200 mesh screen, passing 30 mesh.

d_{200} the fraction of 2nd vanner tailing on 200 mesh screen, passing 30 mesh.

β the fraction of Wilfley tailing in general tailing.

γ the fraction of 1st vanner tailing in general tailing.

δ the fraction of 2nd vanner tailing in general tailing.

Actual screening showed that only a portion of the general and Willey tailing remained on the 30 mesh screen, and the vanner tailing contained only negligible portions of such material. This furnishes the equation,

$$\beta b_{30}=a_{30} \qquad \beta=\frac{a_{30}}{b_{30}}$$

from which β can be calculated.

For the screenings passing through 30 mesh and remaining on 200 mesh the following equation will hold true,

$$\beta b_{200}+\gamma c_{200}+\delta d_{200}=a_{200}$$

But according to our assumption,

$$\beta+\gamma+\delta=1$$

and by eliminating δ we get

$$\gamma=\frac{a_{200}-d_{200}-\beta(b_{200}-d_{200})}{c_{200}-d_{200}}$$

The actual screen tests for one day were as follows:

$a_{30}=17{\cdot}7$ $\qquad a_{200}=36{\cdot}8$

$b_{30}=40{\cdot}7$ $\qquad b_{200}=53{\cdot}9$

$c_{200}=44{\cdot}0$

$d_{200}=\ 9{\cdot}3$

The calculation gives the following results,

$\beta=43{\cdot}5$

$\gamma=23{\cdot}3$

$\beta b_{200}+\gamma c_{200}+\delta d_{200}=a_{200}$

$\beta+\gamma+\delta=1.$

The elimination of γ furnishes the equation,

$$\delta=\frac{a_{200}-c_{200}-\beta(b_{200}-c_{200})}{a_{200}-c_{200}}$$

$\delta=23{\cdot}3$

As a check on this calculation $a+\beta+\gamma$ may be formed which would add up to 100%.

This may be sufficient to show how similar problems may be solved. If it is necessary frequently to determine what percentage of the total tailing originates from the different machines in a concentrator it may be wise to install mechanical appliances for this purpose. At my suggestion the Detroit Copper Co. has done this in the following way: The tailing launders from the different groups of machines are conducted separately to the bottom of the mill. There separate sample slots are provided for the discharge from each launder. These slots are of equal width and are operated by the same mechanism, consequently the samples cut out by them are proportional to the total amount of tailing. I may mention, by the way, that the three streams of tailing are combined after passing their respective sampling slots and that a sample of this general tailing product is taken through a slot which is operated by the same mechanism that actuates the three sampling devices mentioned above.

Of course the assay of the general tailing can be calculated from the weights and assays of the three separate tailing samples. The comparison between the calculated and assayed general tailing sample furnishes a check on the work of sampling and assaying.

Distribution of Pulp.—The question may be asked whether it is of so great importance to know just how the pulp is being distributed in a mill, and whether the mill man cannot tell from the appearance of his machines whether the distribution is correct. A comparison between actual determination and estimates from appearances of the machine has shown that such estimates are in no way trustworthy. There will be one distribution of pulp which will give the minimum loss, and the only question is whether the results obtained by that distribution are sufficiently better than any others to warrant exact determination of the proper conditions. I will try to illustrate this by again referring to a practical case. In a certain concentrating plant two vanner feeds are made. The separation of the two feeds is made in a V-shaped box classifier in which, however, no hydraulic water is used, so that only a rough classification is made, throwing the greater part of the sand and the smaller part of the slime to the first set, and the rest of the sand and the bulk of the slime to the second. The question now arises, how much feed should each set of vanners carry?

Some tests showed that if the best possible saving of copper was desired, only a very small feed should be carried per machine, and that increasing this load increased the loss of metal. But such a policy could not be carried out in a commercial plant, as the question of operating cost would enter. An extremely light load would be impossible, for only a relatively small profit would remain after deducting operating costs, which in the case of a small load would form a serious item.

Let us call metallurgical saving the percentage of valuable metal recovered. If we deduct from this figure as many per cent. as would pay for the operation of the machine, we will get what we may call commercial saving. Now the commercial saving is the best, or expressed mathematically, has a maximum for a greater load. Assuming, for example, a copper value of 15 cents per pound, curves may be drawn which it is assumed will represent commercial saving for the coarser vanner feed, as 5½ tons, and the finer vanner feed, as 8 tons.

This would mean that the feed should be distributed between the first and second sets of vanners in the proportion of 11 to 16 in order to get the most economical load, and it shows the value of the method in guiding the concentrator foreman in the distribution of pulp. In arriving at this conclusion the assumption has been made that by changing the size of the classifier spigot discharge, the nature of the two feeds is not changed. This assumption will be safe when the existing distribution is not very different from the one found best by the method.

The result indicated, viz., that a vanner treating a coarse feed should receive a smaller load than a vanner treating a slime feed, may seem surprising in view of the fact that a slime table should receive a very small load in order to make a good saving. I think also the fact that a slime vanner carrying its best load will make high tailing and a vanner treating coarse material will produce low tailing would induce some mill managers to take the feed away from the slime machines and increase the load on the coarse.

Nevertheless, considered from the standpoint of the curves this would be a mistake. The explanation lies in the fact that this slime feed contains such a large percentage of what has been called absolute slime, on which concentrating machines cannot effect a saving by any means, that the proportion of copper that can be saved is very small and would not pay for the operation of the large number of machines necessary to save it. A coarser feed, however, contains only a small percentage of this extremely fine slime, and for this reason the recovery will be high in every case. But even though it may be high already, it may pay to try to increase it by distributing the pulp over more machines.

There is one point that I would like to emphasize in this connection; that the tailing assay gives no information regarding the work which a machine or plant is doing. The only criterion is, does the tailing contain mineral in a form that can be saved by wet concentration. It is not easy to determine this point, and the only place where its determination is attempted, as far as I am aware, is in the concentrator of the Green Cananea Co., Cananea, Mexico. Although the methods in use for this purpose may not be perfect, it seems to me that by adopting some such plan we shall be able to distinguish between avoidable and unavoidable losses and learn to eliminate the former."—RUDOLPH GAHL, *Western Chemist, Mines and Minerals*, Oct. 1911, p. 188. (A. R.)

REGENERATION OF CYANIDE SOLUTION.—"The recovery of the cyanogen present in solution in combination with the various metals it may have dissolved during the treatment of an ore is the object of a process patented by Mr. Raymond P. Wheelock, of Searchlight, Nev. Generally speaking the results of the process are first the precipitation and removal of such base metals as copper, lead and zinc which may have been dissolved during the treatment of the ore; second, the precipitation of any gold and silver which has escaped regular precipitation by the regular methods; third, the recovery of the cyanogen which was combined with these metals as soluble compounds.

The first step is to add sufficient sulphuric acid to the spent solution to precipitate as simple cyanides the metals held in solution as double cyanides of the metals and the alkali or alkaline earths present. This will liberate one-half of the cyanogen combined as double cyanides, which will be held in solution as hydrocyanic acid if care is taken not to agitate the solution. Taking copper as a typical case the following equation will show the reaction:

$$H_2SO_4 + K_2Cu_2(CN)_4 = K_2SO_4 + 2HCN + 2CuCN.$$

The precipitate is removed and the solution is rendered alkaline by the addition of a soluble alkali or alkaline earth, such as calcium hydrate, thus regenerating potassium cyanide as follows:

$$K_2SO_4 + 2HCN + Ca(OH)_2 = CaSO_4 + 2KCN + 2H_2O.$$

The precipitate resulting from this step is filtered off and the resulting clear solution used again as a solvent for gold and silver.

The precipitate resulting from the addition of sulphuric acid may be further treated with acid and heated to evolve hydrocyanic acid which can be combined with a solution of an alkali for further use. The metals remaining from this treatment can be recovered if their value warrants the operation."—R. P. WHEELOCH.—*Metallurgical and Chemical Engineering*, Oct., 1911., p. 521. (J. A. W.)

SUGGESTIONS ON CYANIDE PRACTICE.—"Several writers have from time to time called attention to the advisability of separate treatment for natural slime. I think that much may be done in this direction to develop the highest efficiency in many plants. E. M. Hamilton has pointed out that at El Rayo mine a higher net profit per day was obtained by treating very fine sand separate from the slime. The essential reason for this treatment at El Rayo was that whereas the extraction obtained from the sand and slime when collectively treated was just as high as when separately treated, still the time of treatment required for such an extraction reduced the capacity of the plant a great deal.

By treating separately, a shorter treatment for the slime gave the maximum extraction, and only a trifle longer treatment was required on the sands. The result was more tons treated per 24 hours with the same equipment and a higher ultimate return with no increase of costs.

In the same way I believe that the natural slime separated before final regrinding begins, would yield the maximum extraction in a much shorter period than that required for the final product of regrinding. In this direction I look to see much advance made in reducing the vat capacity required as well as the volume of solution to be handled per ton of dry slime treated.

The success reported for the Just process must call for considerable argument in the near future, regarding the relative merits of two theories which seek to explain the best conditions for rapid dissolution of silver values in a slime charge. Is it necessary to circulate solution faster or slower than the slime, to promote a perfect diffusion of the dissolved metals, from the immediate surface of the slime particles to the general mass of solution? Or can the same rate of extraction, accomplished by the above method, be obtained by merely keeping the slime particles suspended in the solution, depending upon a purely physicochemical diffusion of the dissolved metals rather than mechanically assisting such an action?

Touching on the subject of the economic limit for moisture in the feed for a tube mill. It is only necessary to note the difference between the consistency of sand and slime, each containing 25% moisture, to realise that whereas the sand might grind all right at that dilution, certainly the slime would not do so. As the surface of the material increases, due to reduction in size of its grains the amount of moisture required to keep it mobile necessarily increases. This points to the advantage of stage grinding in which, for instance, ¼ mesh product could be introduced at 20% moisture in the first stage, and after classifying the discharge, the second stage could be conducted with 39% moisture as suggested by Walter Neal's experience at Dos Estrellas.

Next, as to stage grinding, I wonder that the advantages of this refinement have not made greater headway in general practice and I also wonder why the high speed Chilian mill has come into favour instead of a short length tube mill as the intermediate grinding unit, in several large companies which have already adopted stage grinding. Certainly the operating costs for the Chilians must be much greater than it would be for 15 ft. tube mills. In fact I should consider the tube mill more efficient in power duty, labour and supplies than the Chilians.

In conjunction with stage grinding as already suggested, it would seem preferable to use the Callow screen to separate the coarse feed for the primary grinders and Dorr classifiers for the secondary mills. Whereas there can be no doubt of the efficiency of the cones provided with diaphragms, as described by Mr. Neal, for dewatering a tube mill feed, still I doubt whether they lend themselves so well as the Dorr classifiers do, to rehandling the entire discharge of the tube mill. The Dorr machine is a classifier and a dewatering machine as well, with no rival for compactness and saving in grade or elevation about the plant.

On the other hand the dewatering cones using baffles must be preceded by other cones for separating the greater bulk of slime, and this entails much more difference in elevation.

I would recommend the practice of uniform charges for the vats so that by simple arithmetic it may be possible to obtain useful data as to the working, instead of having to analyse the usual mass of documents forming the reports.—J. B. STEWART, *Mining and Engineering World*, Sept. 23, 1911, p. 588. (A. R.)

MINING.

FILLING MINE STOPES WITH SAND.—"Sand filling of stopes, though not new in some of the coal-mining regions of Europe, in Western Australia, and more recently on the Rand, has not been extensively attempted, if at all, in the metal mines of the western United States. There is nothing impracticable in sand filling, and where the conditions are such that it can be economically applied, it seems to be the best method of filling mine workings.

The technical journals and transactions of some of the mining societies have of late contained a number of interesting contributions on sand filling. The art seems still to be, to some extent, in the experimental stage, particularly as regards its application in metal-mining practice. Important factors in the successful employment of sand filling are; the dip of the vein, the method of mining and the situation of the available sand with reference to the shaft or other channel through which the sand may be introduced into the mine workings. By successful is meant economically, as well as mechanically successful, for without doubt mine workings could be easily filled with mill tailing by any one of a dozen different methods, if the expense of the operation were of no moment. To be successful, therefore, the method of sand filling must be lower in first cost than any other method, and must be free from the danger of disaster in the future.

The scheme may be considered scarcely to have advanced beyond the experimental stage in the metal-mining regions outside of Western Australia, and, therefore, it is not known whether the risks inherent to this method of filling stopes are really greater in wet mines than in those that are dry, or comparatively so, like those of Western Australia and of the Rand.

The Practice in European Collieries.—A brief résumé of what has been accomplished is interesting, as it indicates the possibilities of the system in its application to metal mines everywhere, as well as to coal mines in which latter sand filling has thus far found its greatest employment. The workings of what is known as the Shamrock mine, in Westphalia, are filled with sand. These workings are about 1,900 ft. deep, and the sand is carried 1,500 ft. horizontally from the shaft underground and deposited; at another mine it is carried a horizontal distance of over 3,300 ft. At the Myslowitz colliery, 3,000 tons of sand were daily sent down into the mine for stope filling. In this mine one coal seam is 8 ft. to 21 ft. thick, and another 28 ft. to 37 ft., indicating that sand filling is applicable to large as well as small stopes. The sand is obtained from a bank 20 ft. thick and 1,500 ft. long, steam shovels being used to load cars, which are hauled to the point of discharge by steam locomotives. There are two bore-holes, one 780 ft. deep, the other 1,100 ft. From the foot of these holes the sand is distributed to the stopes. In European practice one of the difficulties of sand filling is the rapid destruction of conduits, which are cut by the passing sand. In one instance wood linings passed 45,000 tons of sand before being worn out. In another case, in Germany, a white porcelain lining passed 90,000 tons, at which time it appeared to be about half worn out. A lining of glass passed only 60,000 tons, and one of earthenware about the same, before they became useless. At one mine iron pipes without lining are used, the holes cut by the sand being repaired as they occur, by means of the oxyhydrogen flame.

At a colliery near Liège, Belgium, the sand is sluiced down with water through 6 in. pipes. In one instance the pipe was 460 ft. long, with 90 lb. pressure at the bottom. The sand and water were carried by this head 2,100 ft. along a level, at which point the pressure was practically nil, due to friction of the pipe. From here the sand was carried by gravity down the slope in 5 in. pipes to the place of distribution in the stopes. Another 6 in. pipe column was 1,100 ft. in length, with 50 lb. pressure at the bottom. The sand was carried along a level 650 ft., down an incline 400 ft., and then along a drift 1,350 ft., where the friction was found to have equalized the pressure. From this point smaller pipes were used, and a total horizontal distance from the borehole of 3,600 ft. was reached. The drifts had a gradient of 1 in 100. At one coal mine in Silesia 7,500 tons of sand and débris are sent down daily to fill the stopes.

In some of the European collieries crushed rock, fine and coarse together, is used for filling instead of sand. This, in America, would probably be economically impossible, except, perhaps, in a district where such material is the result of mining and metallurgical methods, as in some cases in the Cripple Creek district, where there are large dumps of waste in which the largest pieces are about ½ in. diameter. There the rock from the mine passes through a sampler, and this crushed rock accumulates in large quantity. Where rock is not thus obtainable, as a sort of by-product, it would be an expensive filling.

Practice in Australia and South Africa.—In Western Australia, mill tailing is sent down into the mines through shafts from the surface. The tailing may contain up to 25% moisture: if more, it packs in the passes and gives considerable trouble. When the sand is wetter than this it is sometimes allowed to remain on the surface for a few days to dry out. The sand is distributed by means of a belt conveyer placed on an old level running over the stopes to be filled below. Sand filling has been practised in the Western Australia mines for the last 14 years, so may be said to have there passed the stage of experiment. The cost is stated to be about 20c. per ton of ore extracted, which certainly compares favourably with the most economical employment of timber in American mines.

There appears to have been more difficulty in applying the method on the Rand than elsewhere, due to conditions there, and to the character of the sand. It should be understood that clean, sharp sand does not constitute an ideal filling material by any means. It is compact, and possesses considerable cohesion when wet, but when dry it runs readily, then becoming an element of no little danger if a bulkhead breaks, or if it gets an opportunity to run from any cause. Most miners would as soon be drowned as smothered beneath tons of dry sand. The mill tailing of the Rand contains little beside silica, and therefore, does not become consolidated, so that if a bulkhead breaks, because of decay, or in case it becomes desirable to remove it, the sand will run freely, if dry, and this it is necessary to avoid.

At the Silesian coal mine above referred to, the material used in filling consists of ferruginous sand, clay, rock, slag and other débris. These are handled with a steam shovel. To this mixture are added bricks, slate and earthy coal, all of which is run through a rock breaker. In addition to this, hot furnace slag is run by launders to the mixing machinery. Much of this material is hauled 11 miles by railroad to the mines. The mixture is sent down into the mine through porcelain-lined pipes which pass through the shaft. Wooden bulkheads are constructed in the mine to hold the filling. Owing to its composition—a mixture of sand, clay and iron, besides considerable coarse material—it has sufficient porosity to permit the water to escape, and later becomes a solid mass which can only be broken by a hammer: this is quite in contrast with a sand filling that will run through a knothole at every opportunity.

Retaining the Filling.—It is evident that under pressure the sand can be carried at least as far as 3,600 ft. horizontally from the point of discharge at the shaft, winze or borehole. The size of the pipe used will naturally affect this distance, as there will be less friction in a large than in a small pipe. If, in any case, the pressure is too great, owing to the great depth of the shaft, the pipe may be broken into two or more units, with a hopper at the top of each unit. As friction appears to be so important it is improbable that the pressure will ever be found to be too great. The principal difficulty in the application of the method to the mines of the Rand seems to have been the construction of long bulkheads at the bottom of extensive stopes which would safely hold the load of sand and water poured in upon it from above. Mr. Johnson estimates the total pressure upon a bulkhead, 400 ft. long, 6 ft. high, in a stope lying at 30%, with sand filling piled to the height of 200 ft. above it, at about 10,800 tons, or 9,000 lb. per sq. ft. These figures evidently contemplate 200 ft. on the slope, not vertical measurement, and that the sand is perfectly dry. Thoroughly wet sand will weigh about 120 lb. per cub. ft., which would greatly increase the weight on the bulkhead, if the total weight of the sand and water rest upon it. This, however, is improbable, in the case of a stope only 6 ft. high, as at so low an angle as 30% much of the weight of the sand would be on the foot wall, whereas the pressure of a column of water on the bulkhead would be directly proportional to its height.

It is quite evident that the construction of long bulkheads, such as described, at or near the bottom of a stope, is not the safest way of holding back the sand. If it is run in wet, the sand will settle and some water will rise to the surface of the sand, and this may be drained or pumped away, while some of the water will surely find its way out through holes and interstitial spaces in the bulkhead, and even in the surrounding rocks themselves, if the pressure be great. Anything that could become a tributary cause to a mud rush must be carefully avoided, and to me it seems that a long bulkhead, which may possibly give way under pressure, is one of the things that might result in such a catastrophe. For this reason it appears desirable that some other form of support than a wooden, or even a concrete bulkhead, or 'barrier,' as they call it on the Rand, is desirable. I would suggest that the better plan were to leave a solid back of ore, from 10 ft. to 15 ft. thick over the gangway, and to cut through it at only such places as are necessary for manways, for ventilation and for chutes. This, together with shrinkage stoping (drawing away only sufficient broken ore to make room for the miners at the top of the stope), should make sand filling possible anywhere that this mining method can be applied. When the stope is finished all of the broken ore may be drawn out, the chute holes and manways sealed by heavy timber bulkheads, in which provision has been made for the seepage of water through filters, and then running the mill pulp into the abandoned stope. Later the solid backs left over the gangways could be mined, if the filling became solid and coherent; if not, these horizontal pillars could be robbed as far as might be considered safe. True, some loss of ore might be entailed by this method, and this would have to be charged to mining expense. Mining cannot be carried on without cost, and the constant aim of the mining engineer is to mine safely at the lowest possible cost. If the ore that must be lost is so high grade as to increase the expense greatly, then the mine is rich enough to mine by some other method which will permit the removal of all the ore.

Manipulation of the Filling Process.—If the sand, with sufficient water to carry it, be poured in at the top of one end of a stope, the solids will settle to the bottom at the end beneath the entry pipe and gradually pile up there, sloping at a low angle toward the other end of the stope where fairly clear water will collect. At least, that is the result when tailing from a mill flows into a pond, and it is only reasonable to expect that it will act in the same manner as a stope, or in any sort of reservoir. It is desirable, even necessary, to get rid of this superfluous water as speedily as possible. It would, in time, doubtless drain out more or less completely, but it should be removed as fast as it enters, leaving only that water that saturates the sand to percolate and drain out through the filters previously provided in the bottom, in the same manner, that it does in cyanide tanks.

A. R. Hughes* has described a device for draining water from a sand-filled stope in the Simmer and Jack mine on the Rand. The stope was completely inclosed, either by solid rock, by hand-built rock walls, or by timber bulkheads. The necessary filters were provided at the bottom of the stope for the drainage of the sand. A small inclined raise was driven from a cross-cut beneath up into the stope, and in this was placed a wooden box 12 in. square, having on its upper side a series of frames covered with cocoanut matting. A 6 in. pipe was fitted into the lower end of this box, the pipe being then surrounded by a carefully placed, dry rock wall. The top of this filter box extended above the surface of the water, and was carried up in 12 ft. telescoping lengths, as the sand accumulated in the bottom of the stope. The superfluous water passed freely through the cocoanut matting and ran down the box into the pipe and out on the level below, where it was conducted to a tank and hoisted to the surface.

This method of handling the water is similar to that which I have heretofore suggested, and contemplates that the unfilled portion of the stope above the top of the filter box is always accessible to the men whose duty it is to place the additional sections of the box in position as the depth of the sand in the stope increases. Where this is not possible, or where it is too dangerous to venture, owing to impending rock falls, the filter box should be completed from top to bottom of the stope before filling begins, and it may even be deemed advisable to protect the filter box with plank, a crib of timber, or in particularly bad places, with loose rock where the stope is flat enough for such cover to remain in place. As the filter box is gradually submerged in the accumulating sand, it is essential that some means of preventing the filter from caving in under the increasing pressure be provided. To this end I would suggest that the matting be placed upon a supporting frame within the box, and that the upper side of the box be of 2 in. plank perforated with 1 in. auger holes.

Surface Conveyance of Filling.—One more matter to be considered is the conveying of the sand from the accumulated pile on the dump, or from the tailing launder of the mill, to the shaft, winze, or other passage leading into the mine. In nearly every instance the mill, and consequently the tailing pile, are at a lower level than the collar of the shaft. For this reason the sand must be elevated or else introduced into the mine through a shaft, winze or borehole sunk at some lower point on the surface, and connecting with the stopes below. In many places a tunnel can be driven at a lower level than the mill, which can be connected with the workings and thus facilitate the handling of the material. Where this is resorted to it would be better to make a flat incline of it, one in which the tailing will run freely, either in a flume or on the rock floor. Mill tailing usually carries an excess of water, but it is an easy matter to dewater it to almost any desired extent by any one of the several dewatering devices on the market. A mixture of about two parts of sand to one part of water by weight, will flow on a gradient of 1 in 5. A mixture containing a greater proportion of water will run on a lighter grade."—WILLIAM H. STORMS, *Engineering and Mining Journal*, Sept. 23, 1911, p. 585. (A. R.)

AURIFEROUS TERMS.—"A meeting of the members of the Ballarat and District Mine Owners' Association has adopted the recommendations of the Chamber of Mines (Melbourne and Bendigo Chamber) in regard to the following terms in reporting the values of ores: 'Payable gold' shall mean that it will pay working expenses; 'fair gold' shall mean that it will give a profit after paying working expenses; 'good gold' shall mean that it will give 1 oz. and upwards to the ton in quartz mines, and 1 oz. and upwards to the fathom in alluvial mines."—*London Mining Journal*, Oct. 14, 1911, p. 993. (A. R.)

MISCELLANEOUS.

THE ELECTRIC WEIGHER.—"In most industrial establishments where material is handled in bulk, the conveyor is rapidly superseding other methods of transportation. This has created the necessity for a new method of weighing at least as accurate as the scales used with the older methods of handling in barrows, cars, or other receptacles. The design of the electric weigher is the direct outcome of this necessity.

Principles of the Electric Weigher.—While material is in motion on a conveyor, the former method of leisurely direct comparison of its weight with standard weights is impracticable, and interruption of its motion for the purpose of weighing is distinctly disadvantageous. In order to weigh a continuous stream of material in motion, it is necessary to take into account the instantaneous weight of a definite part of the length of the stream and the speed at which that part of the stream is moving. While the problem was being studied with a view to adopting old methods of weighing and old methods of integrating to the new conditions, it was one which presented many serious difficulties. At a certain point, however, the idea of utilizing electricity was hit upon and the difficulties soon vanished.

The measurement of electrical quantity has been brought to a degree of precision not excelled by that attained in any other field. Ohm's law says that the quantity of electricity which flows in any circuit in a given time is proportional to the product of the electromotive force, or voltage, and the conductivity of the circuit. As soon as the problem of weighing a moving stream began to be studied, it was seen that the total weight passing a point in a unit of time is proportional to the product of the average weight of a unit of length of the stream and the speed of motion.

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*See this *Journal*, Vol. XI., Oct. 1910, pp. 173-174.

This led to the thought that if an electromotive force could be produced proportional to the speed of the stream and the conductivity of a circuit be made to vary with the weight of a part of the stream, or *vice versa*, the quantity of electricity which would flow in a given time would be truly proportional to the weight of material discharged by the stream in the same time.

Thanks to the wonderful progress already made in electricity, this can be done with an accuracy that leaves nothing to be desired.

It will be seen, therefore, that the electric weigher must consist of three elements, one corresponding to each of the three terms in Ohm's law. The first, to produce the electromotive force, is a dynamo: the second, to vary the conductivity of the circuit, is a rheostat: and the third, to measure the current, is a meter.

Constant-field Dynamo.—The dynamo is subject to the rigorous law that the e.m.f. produced is proportional to the strength of the field and the number of conductors cutting across it per second. With a constant field and constant number of conductors in the armature the e.m.f. is directly proportional to the speed. The field is made constant by the use of permanent magnets. The demand for accurate electrical-measuring instruments has resulted in the development of special steels for permanent magnets and special methods of "ageing" after magnetization, so that it is now possible to procure magnets which, when properly mounted, show no deterioration whatever over indefinite periods of time. The rest of the dynamo, or magneto, consists of the usual parts, pole pieces, armature, commutator and brushes. The design of the dynamo and the range of current used are such that armature reaction is not allowed to enter as a disturbing factor; and multiple brushes of a highly perfected material developed in Europe, maintain indefinitely a perfect condition of the commutator, and bring the brush resistance under absolute control.

The Rheostat.—The second element, the rheostat, is controlled by scale beams of the usual lever and knife-edge type. Leaving for the moment the important question of the proper suspension of the conveyor, the construction of the rheostat will be described. The counterpoise of the scale is a cylindrical plunger, which is partly floated in mercury. With every change in the weight on the conveyor the plunger instantly adjusts itself to a new position, thereby at the same time changing the level of the mercury.

The rheostat proper consists of a large number of loops of wire placed in narrow vertical spaces at the two sides of the weigher. Each loop is connected to a platinum-tipped contact point on the inside of the mercury vessel. The current from the dynamo is led into the lower part of the mercury and leaves it by way of the highest contact point, which is below the surface of the mercury. In this way different portions of the total resistance are short-circuited in accordance with the weight.

The plunger constitutes a self-adjusting counterpoise, which operates without friction. The mercury vessel is also a dash pot which quells all vibration. The upper part of the vessel is widened out to form a reservoir, which is partly filled with oil. This protects the mercury from the air and forms a trap to catch any fine dust which may pass the other safe-guards mentioned below.

The Recording Mechanism.—The meter is a form of ampere-hour meter, which has been used for years. It has a single rotating member floated in mercury and revolving in proportion to the flow of current. Friction and wear are eliminated, so that no change can be detected in the meter after years of use.

The method of calibrating the weigher is such, that it is only necessary to have a meter which will always go at the same speed when conveying a given current. This meter fulfils this requirement absolutely. The dial of the meter is, of course, graduated to read in units of weight.

The variation in the specific gravity of mercury, due to changes in temperature, is compensated for by making the mercury vessel of an insulating material whose coefficient of expansion just offsets that of the mercury. To illustrate the importance of this it may be noted that the difference between the expansion of iron and of mercury would cause a difference of 1% in the recorded weight between winter and summer, if plunger and mercury vessel were made of iron.

The conductivity of most conductors of electricity, of course, varies with temperature, increasing in some materials while decreasing in others. Effective means for balancing these variations have long been in use, and in the electric weigher the materials are so chosen that the meter has the same rate for a given weight and dynamo speed regardless of the temperature.

In connection with the above, the calibration of the weigher will explain its remarkable accuracy. Each resistance loop is provided with an adjusting rider astride of its two wires. The rider short-circuits the part of the loop beyond it. By moving the rider, an extremely fine adjustment of the resistance of the loop is made. Standard weights are suspended from the scale and the dynamo is run at a known speed and each loop is adjusted until the meter is exactly correct for the corresponding weight. The riders are then soldered permanently in position and the weigher is ready for service.

Empty Belt Tares the Loaded Portion.—The method of suspending the conveyor is, as pointed out above, a matter of paramount importance. This is because the weight of the empty belt, which is the tare, is not constant. Wear, moisture and sticky material cause continual changes. To provide for this, a corresponding length of the unloaded part of the conveyor is suspended from the scale on the opposite side of the fulcrum from the loaded part, so that its weight acts counter to the weight of the loaded part. In this way the tare weight is automatically deducted and the scale is self-balancing.

There is often much dust in the air in the neighbourhood of conveyors and the protection of the weigher from it has received considerable thought. The dynamo itself is tightly inclosed, and is further protected by a tight casing which surrounds it and the gears through which power is transmitted to it from the conveyor. The meter is hermetically sealed and the only other thing requiring protection is the mercury.

Dust Protection of Delicate Parts.—Looked at from above, the two vertical spaces containing the wire loops and the transverse space containing the mercury vessel form a U-shaped chamber. Except for the narrow clearance around the plunger where it passes through the top, this chamber is tightly closed. The heat generated in the resistance wires is utilized to create an upward flow of air through the chamber. The air is admitted at the bottom through a double cotton filter, and must all escape through the narrow clearance around the plunger. The relatively high upward velocity here prevents the descent of dust, which may get past the double roof above. In addition to this the trap formed by the oil reservoir with its depressed bottom for sediment, is expected to protect the mercury surface for years under the worst conditions.

The electrical system lends itself readily to the attachment of distant indicating and recording instruments of the utmost value, without complication. By simply running two small wires from the totalizing instrument, referred to above as the meter, instruments may be placed in the office of the superintendent or elsewhere, which give a complete record of the tonnage carried by the conveyors. They also show immediately any slowing down or interruption in the feed and make a record of such interruptions, giving the exact time at which they occurred. These distant instruments are operated by an auxiliary circuit from a battery or other source.

Importance of Weight Returns.—Experience has shown that means for instantly detecting variations in any part of a process and of recording them, have a far-reaching, beneficial effect on the capacity and efficiency of plants. This is true enough when applied in the form of pressure gauges and the like to subsidiary processes, but is more especially true when applied to the movement of the material itself that the plant is built to handle. Not only should the movement of incoming and outgoing material be watched every minute of the day, but the amount of material transferred in course of treatment from one process to another, or into and out of storage, should be known from hour to hour. Samples, analyses and assays acquire a new value when the tonnage to which they apply is always accurately known.

In general, the weigher will serve the purpose much better than railroad scales, because it does not interrupt the movement of the material, saves switch-engine service and requires no attention other than to read the dial. The electric weigher is manufactured by the Electric Weighing Company, 180 Thirteenth Avenue, New York "—E. H. Messiter — *Engineering and Mining Journal*, Sept. 16, 1911, p. 551. (H. A. W.)

Electrical Resistance of Pure Metals at Very Low Temperatures.—" For several years Prof. H. Kamerlingh Onnes, of Holland, has carried out exceedingly interesting experiments on the enormous decrease of electrical resistivity which pure metals undergo at very low temperatures. He has now perfected his experimental facilities for the use of liquid helium (which by the way was found to be an excellent insulator) and the results of his determinations of the resistivity of pure gold and pure mercury at liquid helium temperatures are given in a recent issue of the Proceedings of the Koninklijke Akademie van Wetenschappen, Amsterdam, of which a translation in abstract appears in the London *Electrician* of August 4. The author's recent experiments with gold have greatly strengthened his former conclusion that the resistance of pure gold vanishes at helium temperatures. Even more interesting are the experiments with mercury. The measurements were made with the differential galvanometer by the method of overlapping shunts (Kohlrausch) and also by the method of the measurement of current strength and of potential difference. The value of the mercury resistance used was 172·7 ohms in the liquid condition at 0° C; extrapolation from the melting point to 0° C by means of the temperature co-efficient of solid mercury gives a resistance corresponding to this of 39·7 ohms in the solid state. At 43° K (4·3° on the krypton thermometer) this had sunk to 0·084 ohms, that is, 0·0021 times the resistance which the solid mercury would have at 0° C. At 3° K, the resistance was found to have fallen below 3×10^{-6} ohm, that is, to one-tenth-millionth of the value which it would have at 0° C. As the temperature sank further to 1·5° K, this value remained the upper limit of the resistance. The next step was to look for the point at which the resistance first becomes measurable as the temperature is raised. The temperature of this point was found to be slightly more than 4·2° K, at which the resistance was found to be 230 microhms or one-hundred-thousandth of the resistance (solid) at 0° C." — Prof. H. K. Onnes. — *Metallurgical and Chemical Engineering*, Sept., 1911, p. 469. (H. A. W.)

The Use of Dynamite in Agriculture.—" For some time past dynamite has been used in America for agriculture. Not only has it been used to break up big stones and uproot trees, but has also been put into daily use for making ditches, ploughing and planting of trees. The making of ditches by use of dynamite is much quicker and 5% cheaper than any machinery. The farmer digs holes 3 ft. deep and 2 ft. apart, at the bottom of this he puts a small quantity of dynamite: although this method may not make such a tidy hole as by the ordinary way, yet the direction will be quite correct. The U.S.A. possess 70 millions of acres of marsh land; here dynamite has proved particularly advantageous for making ditches. Dynamite can be used advantageously for the same in rocky ground and in primeval forests. In the case of ploughing, dynamite is used to break up the ground below the surface. This renders the ground far more fruitful. It can also be used for planting fruit trees, for it not only makes the holes to plant the young fruit trees in, but breaks up the ground as well. It has been found that the fruit tree will bear fruit sooner than they otherwise would have done. This was proved by planting three trees by means of dynamite and three by the ordinary method."—*Zeitschrift fur das Gesamte Schiesmund Sprengstoffwesen*, July 1, 1911, p. 257. (R. V. C.)

Loss through Bunkering of Coal.—" A report was issued by the Mining Bureau of the United States of America, dealing with experiments carried out during the last three years, in order to discover the loss of calorific power and the physical change of coal due to storage. The experiments were carried out for 18 months with New River coal in the Naval Arsenal at Portsmouth N.H., Norfolk V.A., Key West Fla. and Pittsburg the home of the Mining Bureau. The coal was stored in the open, and also under water in order to find out the saving due to under water storage. Pocahontas coal was stored in the open at Panama for 9 months in order to find out the effects of a tropical climate. Experiments were made also with half bitumous coal of Sheridan, Wyoming, and Pittsburg gas coal. They found out that in the open air the mass of the coal from the Appalachen mountains was very little altered, whereas the coal from the West which was geologically younger, chemically different, and took up more oxygen was considerably weathered. The New River coal lost in the open air 1% of its calorific power during a year, while the coal stored under water showed no loss. This saving, however, is done away with by the diminished calorific intensity of the coal. The only advantage is that spontaneous combustion is impossible. Pocahontas coal lost ·3% of the calorific power during 9 months storage in the open air. The half bituminous coal from Sheridan lost 3% to 5% of its calorific power in three years storage in the open, and the weathering contracted the upper surface one foot. There was no loss of the calorific power of the Pittsburg gas coal during six months storage in the open."—H. C. PORTER & S. K. OVITZ, *Chemiker Zeitung* (paper read at the general meeting of the American Chemical Society), Sept. 14, 1911, p. 1,008. (R. V. C.)

PROCESS FAKES.—"A curiously large number of gold bricks are gilded by chemical methods, invariably applied by amateurs, who have nevertheless utilized most ingeniously some few scraps of chemical information. The surprising thing about the industry is the character and quality of its clientele and the psychology of the selling arguments. Hard-headed business men, enriched by dearly won successes, who have learned to trust their judgment, and who have demonstrated their capacity for affairs, men who turn a box of strawberries upside down, and require a pastor's certificate of character from the office boy, these are the best prospects. They listen, they witness a demonstration, they calculate profits, and they are lost.

After all, the psychology of the transaction is not so obscure. It is because they have learned to trust their own judgment in the things they know about that they are led to regard it as equally trustworthy in case of something about which they only think they know. They have so long ignored expert assistance in their ordinary affairs that when the extraordinary occasion arises they feel quite competent to cope with it alone.

There is, too, unfortunately, still a certain atmosphere of mystery around the processes of chemistry as viewed by the average mind, which clouds deduction and seems to justify the otherwise unreasonable. 'Chemistry accomplishes so many extraordinary things, why not this one which I have seen with my own eyes?'

As examples of successful 'process' swindles, Mr. Little sketches briefly the Jernegan 'sea-water gold' process and other fakes, including that for converting water into burning fluid.

Dangerous as these more grossly fraudulent schemes have proved to would-be investors in the past, there is often even greater danger in propositions put forward with the best intentions by half-informed inventors and promoters. While, therefore, it can be amply demonstrated that no class of investments can be counted on for larger and more regular returns than those in well-considered enterprises based on chemistry, no one untrained in metallurgy should consider such investments without expert advice."—*Mines and Minerals*, Sept. 1911, p. 128. (A. R.)

OFFICIAL VALUATION OF SHARES.—"From your columns I gather that you and many of your contributors regard precious metal mining as a legitimate business and that you consider mining engineers as honoured members of society. My own observations are at variance with these conclusions, for I find that precious metal mining is regarded by many financiers as less dignified than horse-racing, and that mining engineers are given a lower social rating than horse-trainers.

I do not consider that this disrepute need be permanent and I propose to point out a powerful means of bringing mining into better repute. My proposal is that shareholders in all precious metal mines be furnished with an official valuation of their shares at least once a year, or oftener, if necessary.

Mr. M. H. Burnham has recently studied this question and has suggested that fuller information be furnished to the shareholders of mining companies. His suggestion is admirable, but it does not go far enough. Even if the information he asks is given, the position of the ordinary buyer of mining shares is but little improved, for it is impossible for the ordinary mind properly to interpret the mass of statistics included in a complete mining report. The prospective buyer or shareholder asks: 'What are the shares worth?' Every annual report should answer this question explicitly. The filling up of the following form would give all the essentials:

A fair valuation of the shares of the Hypothetical Mining Company on June 30, 1911, is:

Preference shares	£——
Ordinary shares	£——
This value is deduced from:	
Present value of the plant ...	£——
Value of ore developed	£——
Value of probable ore	£——
Value of other assets	£——
Total value of property ...	£——
Number of preference shares ..	£——
Number of ordinary shares ...	£——
Value per share	£——

This valuation should be made by the manager of the company, and a full explanation should be given as to how it is obtained. In other words, instead of having a hodge-podge of statistics, which not one man in fifty can interpret correctly, I would give the shareholder the final opinion of the management as to the value of his shares.

I do not contend that this valuation will be absolutely accurate, but it will be definite and will represent the best judgment of experts most familiar with the property. The chief merit of the valuation is that it tends toward openness and straightforwardness; it works against crookedness and concealment. When generally adopted it will be a great safeguard against the wiles of the predatory promoter, both for the engineer and the public.

It does not follow that the market price of the shares will be identical with the valuation, but there will certainly be a close connection between the two.

I feel quite certain that if a system of managerial valuation of mining shares is put into practice it will greatly benefit all honestly managed mining companies; and all persons interested will wonder why business was done without it.

Possibly some of your readers may cavil at my calling the buying of precious-metal mining shares 'gambling' and may retort by remarking that 'Everything is a gamble.' In the misty domain of metaphysics this dictum may be true, but in the actual world of finance it is easy enough to draw a sharp distinction between investing and gambling.

Everyone familar with the subject can recall mining shares that have doubled or trebled in price within a year, and greater gains are not uncommon; for instance: the shares of the Mohawk mine in Nevada advanced from 20 cents to $20 within a year. Equally familiar are the sudden collapses in the price of mining shares: witness the recent tumble in the price of Waihi. Thus the buying of precious-metal mining shares brings prizes and blanks, and dealing in them is practically a lottery.

The gambling spirit is inherent in human nature and men will always pay more for a gambling chance than its strict mathematical value as calculated by the theory of probabilities: consequently the market price of mining shares, as a whole, will always exceed their real value. Thus, from my point of view, the case of the buyer of mining shares is always desperate, and to handicap him further by incomplete and misleading reports is an outrage. He should be put in possession of all the essential facts relating to the company in which he has an interest and the final deduction from these facts—that is, the valuation of the shares—should be made for him by his servants, the directors and manager of the company, who alone have all the data. And it is because I want shareholders to be fairly treated, and to be in a position to act intelligently, that I advocate the managerial valuation of mining shares in the annual reports of companies.

The giving of detailed information as to ore reserves, costs, and other factors controlling the values of the shares is becoming more common; some of the best reports furnish almost all of the information needed, but even these exceptional reports would be greatly enhanced in utility if they contained a direct valuation of the shares."—*The Mining Magazine*, Sept. 1911, p. 220. (A. R.)

REPRODUCTION OF SURVEY PLANS.—"The means of reproducing plans has been limited hitherto, so far as I am aware, to one of three processes, viz.:—

(1) Photolithography.
(2) Sun-printing.
(3) Tracing.

The first would be used where a reduction in scale and a large number of copies are required. The second when only a few copies are necessary on the same scale; and the third for one copy only.

On the establishment of six District Survey Offices in Western Australia, at the latter end of last year (in pursuance of the policy of decentralization inaugurated by the Hon. J. Mitchell, Minister of Lands) it became necessary to provide the country offices with copies of the Standard Compilation Plans on the quarter-mile scale at the earliest possible date.

Photo-lithography was tried with more or less indifferent results, as the plans had not been drawn with that object in view: it was found that each litho. required a large amount of touching up by a draughtman before it could be issued. As only one copy was required, the cost, which amounted to over £2 per plan, was prohibitive.

To make tracings on cloth of the large number of plans required would have necessitated the employment of an army of draughtsmen and checkers.

Sun-printing, of course, was out of the question, as the compilation plans are plotted on mounted drawing paper and it would have been necessary to specially prepare tracings from which to print the duplicates.

It may be of interest to other survey departments to know the manner in which the difficulty was successfully overcome.

The plans that had to be duplicated are double elephant size (40in. × 27in.), and show all the survey marks, survey data, topography, etc., and are, in fact, copies of the surveyors' diagrams. Three colours are used: black, red and blue, and hills are shown with a wash of neutral tint or Indian ink.

After many experiments on sun-printing lines, it was discovered that if a mounted plan was placed in a sun printing frame with its back to the glass, and a sheet of bromide photo-paper placed over it, and a strong electric light thrown through the back of the plan, that a good sharp paper negative was obtained. From this negative a positive could be printed off in exactly the same manner. If the sensitised film on the bromide paper is not in absolute contact with the ink on the plan, the work will be foggy and illegible. In order to ensure a perfect contact, a sun-printing frame provided with a rubber blanket and vacuum pump is necessary. With the frame in use in the Lands Department in Perth, a pressure of about 3 lb. to the square inch is obtained, which is found to be ample for the purpose, and will flatten out a badly broken plan. The pneumatic frame is provided with hinged flaps which are screwed down on to the edges of the rubber blanket, and the air between the blanket and the glass exhausted by the vacuum pump. The frame is mounted on a stand in such a way that the whole thing can be reversed for the exposure to the light, in a manner similar to that adopted for sun-printing. Of course, a dark room is necessary, conveniences for developing the sheets and an electric light of about 100 candle-power for the exposures. The paper at present used to obtain the negative is a rapid bromide, 'Platino-Argentic,' and for the positive 'Nepera gas-light,' as manufactured by the Kodak Co. in rolls 25in. × 10ft. For the negative an exposure of 15 sec. is given and 6 min. for the positive for a plan on mounted drawing paper. The 'Nepera' paper is far too slow and will probably be abandoned in favour of the rapid paper for both.

For developer the best results appear to be obtained with ferrous oxalate or hydroquinone.

The process gives a wonderfully truthful reproduction of the original drawing pencil, finger-marks and the dirt both on the front and the back of the plan all appear on the print. If the plan is very dirty the result is not pleasing, although everything except the blue figures is quite legible. With a clean, new, well-drawn plan the print appears to possess advantages over a photo-litho. reproduced to scale. With mounted drawing paper, a slight discoloration of the ground work of the print will always appear to be present. This is due to the grain in the calico backing and the paste. With unmounted drawing paper, sharp, jet black lines will be obtained on an ivory ground.

This process has many advantages, which may be briefly summarised as follows:—

(1) *Cost.*—The cost of the first copy of a double elephant sheet would be about 8s., labour and materials; second and subsequent copies, 4s. each. It is possible that a cheaper paper could be manufactured to suit the process.

(2) When a print has been prepared no checking of the work is necessary. Anyone who has had experience in the laborious checking of tracings of large survey plans will know how easy it is for errors to remain undiscovered.

(3) An operator can turn out from 6 to 8 d.e. plans per day, single-handed. When it is borne in mind that to prepare a tracing of one of these plans with an average amount of work on would take a draughtsman from two to three weeks, the saving of time and money is at once realised.

At the present time the process is in the experimental stages, and I have little doubt that, with more experience, improvements will suggest themselves which will give better results at a lower cost.

This method has been brought into the region of 'practical politics' through the skill and efforts of Mr. A. W. Bassett, Compiling Branch."—N. S. BARTLETT, *The Survey Journal*, Sept. 1, 1911, p. 10. (H. A. W.)

Antidote for Cyanide Poisoning.—"In a recent issue of the *Journal*, I noticed some correspondence regarding antidotes in cases of cyanide poisoning. As it is an absolute necessity that everyone interested in cyanide workings should be acquainted with all fresh knowledge gained on the subject, I would like to call attention to my paper on the use of adrenalin as an emergency treatment in cyanide, strychnine and other forms of non-corrosive poisoning.

The usually accepted and best known chemical antidote for cyanide taken by the mouth is an alkaline solution of ferrous sulphate (which combines with the cyanide and forms the innocuous prussian blue), which is conveniently kept in the form recommended by Prof. W. Martin and Dr. J. E. O'Brien. As Martin and O'Brien showed, if the antidote is not administered within five minutes of a fatal dose of poison, there is no hope for the patient. Even when this antidote is kept in its most convenient form, it takes at least two or three minutes to prepare it for administration, so that if two or three minutes had been lost before the patient had discovered that he had taken cyanide, his chances of survival would be extremely poor.

From theoretical considerations I was led to investigate whether the administration of adrenalin would delay the absorption of the poison into the system and so give time for the application of the usual measures and antidotes. The results of my experiments have shown that if the adrenalin solution (which can be kept ready on a shelf and needs no further preparation, although it deteriorates on standing) can be administered within five minutes of the poisoning, there is gained a respite of a further four minutes or so for the administration of the antidote. Moreover, my experiments showed that if adrenalin were given within five minutes of the poison, the animal would recover if its stomach were completely emptied by a thorough stomach wash, even though no chemical antidote had been given.

The recommendations which I laid down were:

(1) Adrenalin to be immediately given—12 cc. of the 1 : 1000 solution diluted, or better, 120 c.c. of the 1 : 10,000 solution.

(2) Use Martin's antidote, 30 cc. of a 23% ferrous sulphate solution; 30 cc. of a 6% caustic potash solution; 2 gm. of powdered magnesium oxide (light). The ferrous sulphate solution should be prepared freshly or kept in a hermetically sealed bottle.

(3) The stomach to be thoroughly washed out.

(4) A further small dose of adrenalin to be given to follow up any cyanide which the antidote or wash has failed to reach. About one-half the dose recommended in direction (1).

(5). A good saline purge to be administered and the patient put to bed under the care of a medical man, if possible. If Martin's antidote is not at hand, then 1, 3, 4 and 5, as above. The adrenalin administered thus as an antidote will not deleteriously affect the individual. I hope these notes may be of some value to cyanide workers."—J. L. Jona, *Engineering and Mining Journal*, Sept. 30th. 1911, p. 625. (J. G.)

Reviews and New Books.

Metallurgy of Tin. By Henry Louis. Cloth, octavo, pp. 140, with many illustrations. (New York: McGraw Hill Book Co.) Price 8s. 6d.

"Of all branches of metallurgy the smelting of tin ores is at the present time the most secret. Those who by labour and experience have mastered the intricacies of the art are still able to adhere to the ancient order of things and keep their knowledge to themselves. At present the smelters in the Straits, Queensland, Cornwall, and Hamburg alone have the knowledge how to combat the affinity of tin for silica, and how to deal with the multitudinous impurities usually found in association, such as iron, wolfram, titanium, etc. The straight tin-smelting reaction is simple enough, and if ore and labour is plentiful a crude furnace and process will be profitable though much metal is lost in the slag. Such operations are conducted in China, the Malay peninsula, and in South America, but they cannot be ranked as metallurgy in its modern meaning, nor are they of great help to the student who desires to equip himself for a practical campaign.

The author has also collected together nearly every item of published information. There are indeed many parts where additional or corroborative information is desirable, and could be given by people with special knowledge without any detriment to their own interests."—*The Mining Magazine*, Sept. 1911. (A. R.)

The Mining Industry During 1910. Vol. XIX. Edited by Albert Hill Fay. (New York: McGraw-Hill Publishing Co. pp. 904; illustrated. $10.

"This well known and excellent compendium of information relative to the mineral industry, embracing statistics of production of various minerals and mineral products as well as reviews of technical progress, has again appeared, being in its nineteenth volume. *The Mineral Industry* is the only work of its kind. Its statistical matter has always been as accurate as possible under the conditions of publication. The statistics for 1910 are no doubt close approximations to the truth—at least close enough to be of considerable value in the commercial world. Where available, information is given as to market conditions and prices. A glance at the list of contributors to this volume—acknowledged authorities in their respective branches—leaves no grounds to question the accuracy of the technical and scientific information contained."—*Mining and Engineering World*, Sept. 1911. (A. R.)

Allen's Commercial Organic Analysis. Vol. 5 4th Ed., entirely re-written. 8vo., pp. 614. *Churchill* Net 21s.

Baily, G. H., and Bausor, H. W., Senior Chemistry. Cr. 8vo., pp. 520. *Clive.* Net 4s. 6d.

Bickford, J. S. N., Faults and How to Find Them. Motor Car Failures and their remedies fully explained. Cr. 8vo., *Iliffe.* Net. 2s. 6d.

Barton, Edwin H. Analytical Mechanics, Comprising Kinetics and Statics of Solids and Fluids. 8vo., pp. 556. *Longmans.* Net 10s. 6d.

Bauer, G. and others. Marine Steam Turbines. Forming the Supplementary Vol. to "Marine Engines and Boilers." 8vo., pp. 230. *C. Lockwood.* Net 10s. 6d.

Bruce, C. R., Marine Engineering Estimates and Costs, for those engaged in the Technical and Commercial sides of Engineering and Shipbuilding. 12mo., pp. 126. *Fraser Asher, Glasgow.* Net 4s. 6d.

Cantell, M. T., Reinforced Concrete Construction: Elementary Course. Cr. 8vo., pp. 144. *Spon.* Net 4s. 6d.

Cattelle, W. R., The Diamond. 8vo., pp. 434. *Lane.* Net 7s. 6d.

Colvin, F. H., Machine Shop Mechanics; the Why of Things in the Shop. 12mo. *Hill Publishing Co.,* Net 4s. 6d.

Corbin, Herbert E. and Stewart, A. M., A Handbook of Physics and Chemistry. 4th Ed. Cr. 8vo., pp. 528. *Churchill.* Net 7s. 6d.

Coste, J. H., The Calorific Power of Gas. Cr. 8vo., pp. 326. *C. Griffin.* Net 6s.

Costello, L. W. J., The Law Relating to Engineering. A Course of Six Lectures Delivered at Caxton Hall, Westminster, in 1910-11, before the Society of Engineers and the Junior Institution of Engineers, With an Introduction by the Rt. Hon. Lord Justice Fletcher Moulton. Demy 8vo , pp. 170. *Society of Engineers.* Net 5s.

Darling, Chas. R. Pyrometry : a Practical Treatise on the Measurement of High Temperatures. Cr. 8vo., pp 22. *Spon.* Net 5s.

David, W. T., Radiation in Explosions of Coal Gas and Air. 4to., sd., pp. 24. *Dulau.* Net 2s.

Factory Form No. 957, Regulations etc. for the Smelting of Materials containing Lead, the Manufacture of Red Lead, etc., and of Flaked Litharge. 1d.

French, T. E., A Manual of Engineering Drawing for Students and Draughtsmen. 8vo. *Hill Publishing Co.* Net 8s. 6d.

Freudemacher, P. W., Electrical Mining Installations. 12mo., pp. 194. *Constable.* Net 2s.

Friend, J. Newton. Elementary Domestic Chemistry. With 55 Diagrams. Cr. 8vo., pp x., 182. *Allman.* Net 2s. 6d.

Gammon, John C. Reinforced Concrete Design Simplified. 4to. pp. 124. *Crosby Lockwood.* Net 10s. 6d.

Haeder, Herman. A Handbook on the Gas Engine, comprising a Practical Treatise on Internal Combustion Engines Roy. 8vo., pp. 330. *C. Lockwood.* Net 18s.

Heineman, Paul G. A Laboratory Guide in Bacteriology. 2nd Ed. Enlarged and Revised. 12mo., pp. 216. *Cambridge University Press.* Net 6s.

Hewlett, R. Tanner. A Manual of Bacteriology, Clinical and Applied. 4th Ed. 8vo., pp. 716. *Churchill.* Net 10s. 6d.

Hurst, Charles. Valves and Valve-Gearing. 6th Ed. Revised and Enlarged. 8vo., pp. 302. *C. Griffin.* Net 9s.

Husband, Joseph, and Harby, William. Structural Engineering. 8vo., pp. 408. *Longmans.* Net 7s. 6d.

Iddings, J. P., Rock Minerals. 2nd Ed. Revised and Enlarged 8vo., *Chapman & Hall.* Net 21s.

Investigation of self-contained Breathing Apparatus, for use in Mines, and The Report of a Committee of the South Midland Coal Owners, etc. Roy. 8vo., pp. 102. *Cornish Bros.* Net 7s. 6d.

Jamieson, A. P., Isometric Drawing; a Treatise on Mechanical Illustrating etc. 8vo., *Hill Publishing Co.,* Net 4s. 6d-

Jamieson, Andrew. Elementary Manual on Heat Engines : Steam Gas and Oil. 13th Ed. Revised and Enlarged. Cr. 8vo., pp. 442. *C. Griffin.* Net 3s. 6d.

Jones, T. and T. Gilbert. Machine Drawing for the use of Engineering Students. Book 2, 2nd Ed. Revised and Enlarged, Book 3, 3rd Ed. 4to., limp. *J. Heywood.* Net 3s.

King, W. R., The Elements of the Mechanics of Materials and of Power Transmission. 8vo., *Chapman & H.* Net 10s. 6d.

Lodge, R. W. Notes on Assaying and Metallurgical Laboratory Experiments. 3rd Edition. Roy. 8vo. *Chapman & Hall.* Net 12s. 6d.

Louis, H. Metallurgy of Tin. 8vo., *Hill Publishing Co.,* Net 8s. 6d.

Lunge, George. Technical Methods of Chemical Analysis. English Translation. Edited by Charles Alexander Keane, Vol. 2, Pts. Roy. 8vo., pp. 1292. *Gurney & Jackson.* Net 63s.

Magie, W. F., Principles of Physics. 8vo., *Bell.* Net 7s. 6d.

Martin, L. A., Text-Book of Mechanics. Vol. 3 Mechanics of Materials. Cr. 8vo. *Chapman & Hall.* Net 6s. 6d.

Maxwell, William H. Drainage Work and Sanitary Fittings. 3rd Ed. Revised and Enlarged. Cr. 8vo., pp. 127. *St. Bride's Press.*

McIntosh, John Geddes. The Manufacture of Varnishes and Kindred Industries. Based on and including the "Drying Oils and Varnishes" of Ach Livache. 2nd. Greatly Enlarged Ed. Vol. 3 8vo., pp. 492. *Scott Greenwood.* Net 12s. 6d.

Moore, F. J., Experiments in Organic Chemistry. a Laboratory Manual. Cr. 8vo., *Chapman and H.* Net 2s. 6d.

Morrow, John. Steam Turbine Design, with special reference to the Reaction Type, including Chapters on Condensers and Propeller Design. Roy. 8vo., pp. 480. *E. Arnold.* Net 16s.

National Association for the Prevention of Consumption and other forms of Tuberculosis. Transactions of the Annual Conference at Caxton Hall, Westminster, July 19-21, 1911. 8vo., pp. 220. *Adlard & Son.* Net 2s. 6d.

Neave, G. B., and Heilbron, I. M., The Identification of Organic Compounds. Cr. 8vo., pp. viii., 103. *Constable.* Net 4s.

Parsons, Sir Charles. The Steam Turbine,—the Rede Lecture 1911. Cr. 8vo., pp. 64. *Cambridge University Press.* Net 1s. 6d.

Pearson, K. The Fight against Tuberculosis and the Death rate from Phthisis. Questions of the Day and of the Fray. 8vo., sd. *Dulau.* Net 1s.

Pertwee, Ernest. The Art of Public Speaking. Cr. 8vo., pp. 278. *Routledge.* Net 3s. 6d.

Pullen, W. W. F., The Testings of Engines, Boilers and Auxiliary Machinery. 2nd Ed. Cr. 8vo., pp. 744. *Scientific Publishing Co.,* Net 12s. 6d.

Rankine, W. J. M., A Manual of Civil Engineering. 24th Ed. Thoroughly Revised by W. J. Millar, C. E. Cr. 8vo., pp. 838. *C. Griffin.* Net 16s.

Royds, R. The Testing of Motive Power Engines 8vo., pp. 408. *Longmans.* Net 9s.

Senter, George. A Text-book of Inorganic Chemistry. Cr 8vo., pp. 596. *Methuen.* 5s. 6d.

Seymour, Charles. Speaking in Public, How to Produce Ideas and How to Acquire Fluency. 2nd Ed. Cr. 8vo., pp. xii., 192. *Speaker's Library.* Net 3s.

Spencer, James Frederick. An Experimental Course of Physical Chemistry, Pt. I. Statical Experiments. Cr. 8vo., pp. 242. *Bell.* Net 3s. 6d.

Spencer, Leonard J. The World's Minerals. 8vo , pp. 224. *Chambers.* Net 5s.

Tyrrell, H. G., A Treatise on the Design and Construction of Mill Buildings and other Industrial Plant. 8vo , *Spon.* Net 17s.

Wegmann, E. The Design and Construction of Dams. 6th Ed. Revised and Enlarged. 4to., *Chapman & Hall.* Net. 25s. 6d.

White, P. T., White's Engineering Pocket Book. Mechanical and Electrical, with Notes, Tables, and Formulae in General Use. 12mo. pp. 93. *Wall & Sons, Wigan.* Net 1s.

Abstracts of Patent Applications.

(C.) 548/10. John Fraser Price. Improvements in apparatus for treating crushed ore products with cyanide solution or other liquid. 10.11.10.

This application relates to apparatus for agitating and aerating crushed ore and solution, and consists of a hollow rotatable cylinder provided with vanes and transverse partitions forming pockets, which elevate and drop the pulp on its passage through the cylinder. At the discharge end, similar vanes are provided which lift the pulp into a launder which conveys it wherever desired. A pipe is also provided to convey compressed air or steam into the cylinder to facilitate the aeration. The apparatus as described is similar in appearance to a tube mill without the grinding load, and is discharged by buckets similar to those in a tailing wheel which lift the pulp to fall into the launder placed just within the discharge end.

(C.) 585/10. George Anderson Robertson (1), Charles Thomas Stephens (2). Improvements appertaining to the mortar boxes of stamp mills. 20.11.10.

This application refers to a mortar box screen comprising only vertical wires in combination with a deflector plate which serves to carry the upper ends of the wires. A plate is secured to the bottom of the screen for retaining the lower ends of the wires. The wires are stretched by means of keys somewhat like those of a piano, and this arrangement will give to the wire a good deal of elasticity and keep it in a vertical position. The arrangement admits of the angle at which the screen can be placed from the box being considerably increased.

(C.) 595/10. Henry Crawford Boyd (1), Robert Henderson (2) Improved concrete vibrating moulding machine. 2.12.10.

This invention relates to a machine for the efficient packing together of the broken stone used in the making of artificial stone from broken stone, sand and cement.

This invention consists of a horizontal vibrating table to which a mould of a shape to suit the article to be made can be bolted. The table is held in place by a horizontal arm projecting beyond the end of the table and so pivoted to an upright that it permits of both vertical and horizontal movements being imparted to the table. A vertical vibratory movement is imparted to the table by means of toothed wheels revolving in contact with smooth wheels fixed to the table, whilst a horizontal movement can also be imparted to the table by setting the latter wheels eccentrically to each other.

(C.) 643/10. John Collins Clancy. The treatment of ores bearing precious metals. 30.12.10.

This application applies to a new process for treating ores containing precious metals, and is based upon the reaction of cyanamide and a cyanogen bearing compound or prussiate. The applicant claims to have discovered these mixtures to be very active solvents of the precious metals contained in ores. The claims of the inventor include:

1st. The use of an amide or amidine solution containing a cyanogen bearing compound.

2nd. The use of a cyanamide solution containing a cyanogen bearing compound.

3rd. The process of treating ore containing precious metals by subjecting them to the action of a cyanamide solution containing a ferro or ferricyanide compound.

4th. The process of treating ore which already contains or is in association with soluble or insoluble prussiates to the action of a cyanamide solution.

(C.) 468/11. Theodore Fitzroy Fitzsimons. Venting and discharging valve for vessels containing liquids. 17.8.11.

This invention relates to a device for venting vessels containing liquids and for discharging the liquid contents thereof, and has been specially devised for facilitating the discharging of portable rectangular cans containing benzine, kerosene, spirits and like liquids.

The said device consists of a plug valve with air by-pass, fitted in a valve casing which terminates at one end in a delivering nozzle and at the other in a washered neck adapted to set into a hole in the can head, and a venting pipe in communication with the air by-pass port; this valve is carried by trunnion pins on a forked bracket which springs from a collar slidable on a rod; the lower end of the rod is fitted with a piercing tool and with claws which are engageable with the rim of the can bottom, and the upper end of it is fitted with a clamping nut which acts against the slidable collar to force the valve foot firmly down on to the can head.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 535/11. Asplan Beldam. Improvements in or connected with vehicle tyres. 29.9.11.

(C.) 537/11. Charles Dickie (1), Fred Lambert (2), Richard Stable (3). A safety apparatus for suspending mine cages and the like. 30.9.11.

(C.) 538/11. William Hart (1), Joseph Mark Deschamps (2). Improvements in retorts for burning charcoal or other analogous substance. 30.9.11.

(C.) 540/11. Arnold Frinyi. Improvements in heating devices for liquid fuel. 4.10.11.

(P.) 541/11. Walter Hall. Improvements in lubricating compositions. 4.10.11.

(P.) 544/11. Benjamin Waites. Improvements in means for reducing, classifying and concentrating ores, minerals and the like. 6.10.11.

(C.) 545/11. Bela Dorner. Improvements in or relating to a process for the economic and industrial utilization of maize straw. 6.10.11.

(C.) 546/11. Hugh Mackenzie Leslie. Improvements connected with the treatment of ores by the cyanide process. 6.10.11.

(P.) 547/11. James Snodgrass. Improvements in the manufacture of carbonate of soda. 7.10.11.

(P.) 548/11. Albert Ernest Webster (1), Robert Jackson Webster (2). Regulator for the better separation of sand and slime. 7.10.11.

(P.) 549/11, Ambrose Kowastch (1), Carl Alexander Baldus (2). Improvements in blasting methods and apparatus. 11.10.11.

(P.) 551/11. James Solley Reah. Improvements in means for transporting and loading jumpers or drills and other material into mine skips and the like. 12.10.11.

(P.) 552/11. Nicholas Francois Haarhoff. A new and improved dam scoop, land leveller or the like. 12.10.11.

(C.) 553/11. John Alston Hart. Improvements in and apparatus for effecting the compression of gases. 13.10.11.

(P.) 554/11. James West (1), William West (2). Improvements in means of and apparatus for comminuting mineral substances and the like. 13.10.11.

(C.) 555/11. Dr. Werner Siebert. Process for the electrolytic production of ammonium nitrate from hydrated nitric acid. 13.10.11.

(P.) 556/11. Edwin Ralph. A new or improved remedy or preparation for resuscitating persons who have been "gassed" or are under the influence of mine gases or the like. 13.10.11.

(C.) 557/11. Moses Arthur Knapp. Improvements in or relating to rock drills. 13.10.11.

(C.) 559/11. Valentin Stasch. Improvements in safety apparatus for miners trucks. 13.10.11.

(P.) 560/11. Victor Hartog. Process for extraction of platinum. 16.10.11.

(P.) 561/11. Charles Legward. Improvements in treating crushed ore with cyanide or other treatment liquid. 16.10.11.

(P.) 565/11. Frederich Patterson (1), Thomas Evans James (2). An improved spark-arrester. 18.10.11.

(P.) 567/11. David Hamilton Evans. Improvements in means for aerating slime, cyanide solutions and the like. 18.10.11.

(P.) 568/11. Georg Ludwig Kustner. Improvements relating to lubricators. 19.10.11.

(C.) 569/11. August George Liebmann. Improvements in and relating to railway ties or sleepers. 20.10.11.

(C.) 570/11. Max Adolf Bleichert (1), Max Paul Bleichert (2). An improved hauling rope grip for rope railways. 20.10.11.

(C.) 572/11. Vincent John O'Brien (1), Romeo Chalmers Awkerman (2), Edward Wynne Williams (3). Improvements in valve mechanism for pneumatic tools. 20.10.11.

(C.) 573/11. Charles Hoyle (1), Harry Adelbert Guess (2). Improvements in classifiers. 20.10.11.

(P.) 574/11. Allen Thomas Cocking (1), Kynoch, Limited (2). Improvements in and relating to explosives. 20.10.11.

(P.) 575/11. Allan Thomas Cocking (1), Kynoch, Limited (2). Improvements in and relating to the nitration of sugars and to explosives. 20.10.11.

(P.) 576/11. Harry August Stockman. A new or improved corrugated expanded metal sheeting for building and like purposes. 23.10.11.

(P.) 577/11. William T. Norris. Improvements in or relating to rock drilling machines. 23.10.11.

(P.) 579/11. Thomas Henry Chad (1), Joseph Barril (2). An improvement in the means of lighting fuses for blasting and analogous purposes. 25.10.11.

(P.) 580/11. Charles Edward McCann (1), Samuel Robert Skellorn (2), Sidney Cooper (3). Improvements in incandescent electric lamps and the like. 25.10.11.

(P.) 581/11. William George Manners. An improved centrifugal grinding or sliming machine. 26.10.11.

Changes of Address.

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Horwood, C. D., *to* P. O. Box 26, Roodepoort.
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Irwin, R. W., *to* P. O. Box 108, Germiston.
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Phillips, F. D., *l/o* Germiston; Robinson Deep G. M. Co., Ltd., P. O. Box 1,392, Johannesburg.
Richardson, R., *to* Horo Mine, Louws Creek, Barberton.
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Smith, W. J. W., *l/o* Selukwe ; *c/o* G. F. Smith, Esq., Shangani Siding, Rhodesia.
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THE JOURNAL
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Vol. XII. DECEMBER, 1911. No. 6.

Proceedings
AT
Ordinary General Meeting,
December 16, 1911.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, December 16th, Mr. W. R. Dowling (Vice-President) in the chair. There were also present :—

35 Members : Messrs. R. G. Bevington, W. Cullen, A. F. Crosse, E. J. Laschinger, Dr. J. Moir, A. Richardson, Prof. G. H. Stanley, H. A. White, J R. Williams (Members of Council), W. Beaver, F. T. Chapman, J. Chilton, W. M. Coulter, R. C. Evans, R. Gascoyne, H. L. Harland, A. M. Kratz, G. A. Lawson, H. Lea, J. Lea, R. Lindsay, Dr. D. Macaulay, M.L.A., J. D. Marquard, T. G. Martyn, S. Newton, P. N. Nissen, F. S. Pearce, F. D. Phillips, E. Roberts, A. Schwarz, J. A. Smith, S. H. Steels, J. A. Taylor, O. Tonnesen and C. Toombs.

11 Associates and Students : Messrs. John Cronin, A. King, G. F. Mathews, W. H. Pickburn, F. J. Pooler, H. B. Powter, S. de Smidt, H. Stadler, A. M Thomas, P. A. Tucker and W. F. Zipp.

12 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as published in the November *Journal*, were confirmed.

NEW MEMBERS.

The Secretary notified that since the last meeting of the Society the following Associates had been admitted by the Council:

Fyfe, Alexander, A.I.M.M., M.Am.I.M.E., Grand Junction Gold Co. Ltd., Waihi, New Zealand. Metallurgist.

Quinan, Kenneth B., Cape Explosives Works Ltd., Dynamite Factory, Somerset West, C.P. General Manager.

The Chairman: I have to announce that to-night for the first time for some years we have no members up for election. I also wish to intimate that commencing with the new year, the subscription for new members will be payable only for the half year ending June.

General Business.

The Secretary : The results of the Evening Class Examinations at the School of Mines have just been issued, and the following gentlemen have been successful in obtaining the prizes offered by the Society :—

Chemistry—		
Stage I., Theoretical,	£1 10s. :	T. L. Thorne
Stage I., Practical	£1 10s. :	W. Anderson T.J.R. Richards
Stage II., Theoretical,	£3 :	G. F. B. Ellis
Stage II., Practical	£3 :	Joseph Cohen
Stage III.		No award
Metallurgy—		
Stage I.	£1 10s. :	J. P. Symons
Stage II. (Gold)		No award
Assaying—		
Stage I.	£1 10s. :	Robt. Murdoch
Stage II.	£3 :	D. F. Scott
Metallography—No award		

DEATH OF MR. G. O. SMART.

The Chairman : I regret to have to announce the death of one of our most respected members and a member of our Council, Mr. G. O. Smart. He was one of the oldest and most experienced millmen on the Rand, and trained many men, amongst whom was myself. This Society, his employers and indeed the whole mining industry will lament his loss, as they owe him much. We therefore tender to his family our sincerest sympathy and condolence.

I have pleasure in informing you that Prof. Wilkinson, who has been laid up with enteric, is now well on the road to recovery.

DISCUSSION ON PAPERS.

Mr. E. J. Laschinger (*Vice-President*) : I should like to bring up a matter which I hope will be considered by the Council, but upon which I thought we might obtain an expression of opinion from the members. It has often been noted that there has been a great difficulty in obtaining discussion on some of the papers which have been read before this Society, and I would point out that matter for verbal discussion has often been prepared by members, but owing to the reading of the paper or papers during the evening having taken up an undue amount of time, these discussions we have not been able to include in the proceedings. Then one has to consider that after the coffee interval some members have to leave, so I would like to suggest to the members for reference to the Council the advisability of discussions, at certain meetings, taking precedence over the reading of papers.

Mr. Andrew F. Crosse (*Past-President*): I think that there is a good deal in what Mr. Laschinger has said, and I also consider that when the papers are read first, the discussions on previous papers are apt to be curtailed.

Mr. W. Cullen (*Past-President*): I think a difficulty of the Council will be not knowing what amount of discussion there might be, and therefore in determining on what occasions discussions should take precedence over papers.

Mr. T. G. Martyn (*Member*): I suggest that the papers be printed and circulated before the meeting, they being taken as read on the evening, and the author being allowed a few minutes in which to present a summary by way of opening the discussion.

Mr. C. Toombs (*Member*): I think that a solution of the difficulty would be found in curtailing the discussion on papers the same night they are read. Impromptu discussion, such as must be the case when delivered immediately after the reading of a paper, must of necessity be less valuable than that given after careful consideration.

The Chairman: This matter will be brought before the Council at its next meeting, and the various points which have been brought forward in the discussion, will receive its careful consideration.

THE EXHIBITION.

Mr. W. Cullen (*Past-President*): I should like to draw the attention of members to the notice which has appeared in the pages of the *Journal* in connection with the proposed exhibition. So far as the commercial houses are concerned we have had a most satisfactory response, but there has been very little response indeed from the Society's members in regard to the exhibition of metallurgical and other appliances. It was hoped that this would be the feature of the exhibition, the success of which will undoubtedly depend to a great degree on the efforts of the members themselves. I trust, therefore, that all members who have any models or apparatus suitable for exhibition will communicate with the Secretary as soon as possible.

NOTES ON ABSORPTION OF GOLD BY AMALGAMATED COPPER PLATES, CAST IRON RETORTS AND AMALGAM TRAYS.

By Prof. G. H. Stanley, A.R.S.M., F.I.C. and M. Thornton Murray, M.Sc.

Notwithstanding published work which tends to prove the contrary, it is still occasionally stated that considerable amounts of gold are absorbed by the amalgamated plates used in gold reduction plants.

In order to obtain additional light on the question the authors procured a sample of plate and carried out an investigation with the object of ascertaining whether, in the specimen under examination, such absorption had taken place and if so to what extent, and further, in what manner the gold, if any, had penetrated the plate. The specimen examined had been in use in one of the mills of the Crown Mines for twelve years and was ·13 to ·14 in. in thickness; it had been sweated and scraped, but not scaled, before being discarded, and the under surface was also amalgamated. One portion, on assay, showed the presence of 1·69% gold and 1·31% mercury, and in order to ascertain how this was distributed in the plate another portion was carefully flattened, secured to a wooden support, and successive layers of as nearly as possible ·02 in. in thickness planed off in a shaping machine; particular care being taken to free the edges by filing from any contamination due to the possibility of "salting" by some of the top scale of amalgam being spread over the edge during cutting by a cold chisel. These successive portions were then assayed for gold, mercury and lead, and the results obtained are given below in tabular form, No. 1 being obviously the surface sample.

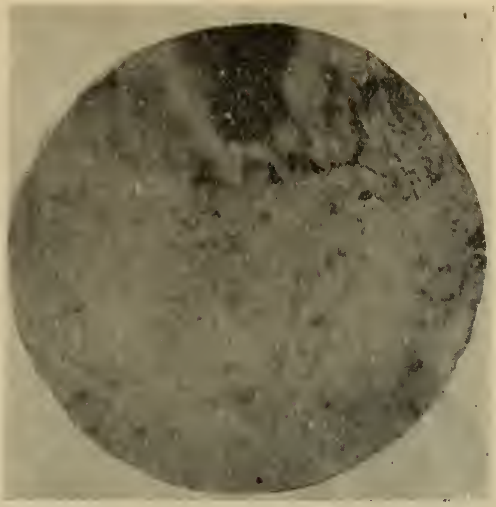
Layer No.	Gold %.	Mercury %.	Lead %.	Ratio : $\frac{\text{Gold}}{\text{Mercury}}$.
1	10·48	9·39	·085	1·12
2	·0166	present in all cases but in amounts too small for determination.	·084	?
3	·00505		·080	—
4	·0084		·053	—
5	·0041		·097	—
6	·0145		·089	—
7	·0018	1·89	·104	·00105
Tl.	1·21			

It will be seen that over 99 % of the total gold is contained in the first layer of ·02 in., the remainder being distributed irregularly throughout the plate, and though it is, of course, quite probable that the plate contained a small amount of gold before use, it seems unlikely that this original gold would be present in such quantity or in such uneven distribution. The lead is probably an original constituent, and being liable to segregation, its uneven distribution need not occasion surprise; it was determined because lead amalgams are soft and pasty and it was thought might give some indication of value, indeed, a slight but undoubted concentration is shown into the lower half of the plate, which would tend to support the idea of actual filtration through it.

To further elucidate the point, a section was cut across the plate and polished and examined under the microscope. This showed plainly a grey surface scale of amalgam lying on the irregular corroded copper surface, many approximately circular or elliptical blue-grey spots, some irregular ones usually of lighter colour, and empty pores or blowholes of very variable shape and distribution. Some of the spots on incomplete polishing exhibited elongated smears which could scarcely be anything but mercury. The section was then heated to a sufficient temperature to expel the mercury and again examined, when the yellow scale of gold was apparent on the edge which originally had been the upper surface and some of the patches were found to have given place to small hollows, some of which seemed to be coated internally to a variable extent, with a yellow film, which as it appears to be quite untarnishable, is in all probability gold. In some places the appearance presented indicated that the amalgam had "sweated" out of the holes and deposited gold around it.

The photographs show the appearances presented, after etching the surface (and actual specimens were exhibited under the microscope at the meeting).

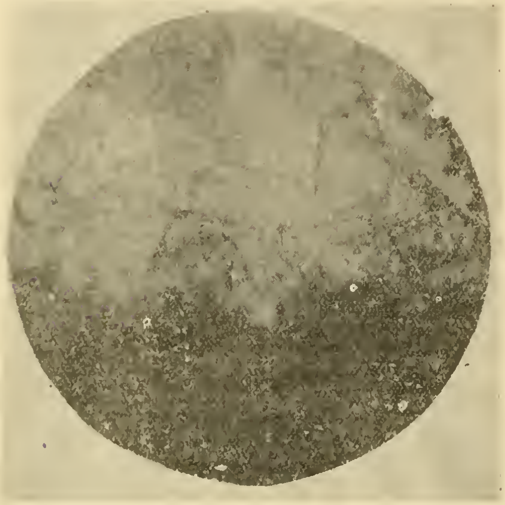
For comparison, another section was prepared (and exhibited to the meeting) from an unused plate: this showed the same hollows and circular or elliptical-shaped blue-grey inclusions, which appear to be an original constituent of the plate, also empty blow-holes or pores, but nothing quite similar to the assumed mercury-containing spots in the used plates, neither was any gold apparent on heating. Following upon this the explanation appears to lie in the fact that the copper plates used are more or less porous and contain minute blow-holes, some of which, at least, communicate with one another. Through these minute passages mercury has percolated carrying gold in solution—or suspension—till the plate became saturated; a small amount may have actually passed out of the plate on the under side, but if so it carried but little gold; however, the amalgamation of the under side may also be accounted for by assuming "creeping" of mercury from the sides, underneath the plate, and there is no evidence one way or the other in the present case. This explanation seems to the authors to be sufficient without having to assume any true *diffusion* of mercury or gold into the copper. Obviously the removal of the surface to a depth of ·02 in. would in this case give all the gold, which, from the mine point of view, would be worth recovering, and this could be effected, if the use of a large planing machine is found to be impossible, by a careful scaling by one or other



Surface of Plate × 175.

Centre of Plate × 175.

of the published methods, and chipping. Though it is perhaps unsafe to generalise from this example the whole examination supports the views expressed elsewhere that gold is not *absorbed* in sufficient amount to influence the percentage recovery: in this case the amount really absorbed is calculated as about ·25 ounces for the whole plate, and the total gold about 46 ounces. It is quite possible, though, for a hard surface scale to form which contains a large amount of gold and is much richer in gold than the usual "scrape" (or amalgam scraped off) even after squeezing, and consequently does not soften sufficiently, during sweating, to be readily removed. It seems very probable, however, that several repetitions of sweating and scraping with hard scrapers, with fresh addition of mercury between each would leave only a very small amount of gold upon the plate, though still enough to affect the return from a short "custom" run; but the authors are of opinion that the figures which have been given in several instances as the amount of absorbed gold have probably been vitiated by the presence of variable amounts of this hard amalgam scale in irregularities or hollows on the surface and do not represent true absorption, which, as already stated, they consider to be practically negligible.

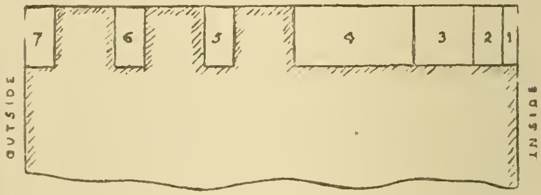


Lower Surface of Plate × 175.

Large Crystal Centre of Plate × 690.

In an appendix, a list of references is given to publications on the subject and the authors would be glad to learn of other experiences in this connection.

Used and discarded retorts are also known to carry considerable, though very variable, amounts of gold, and a specimen broken from one of these was therefore examined in a similar manner. Unfortunately the length of time which the retort had been in use was unknown but this was probably in the neighbourhood of one to two years. The thickness was about $2\frac{1}{4}$ in., and after scraping off as much as possible from the interior surface, successive portions were removed by the shaping machine for assay, as shown in the diagram.

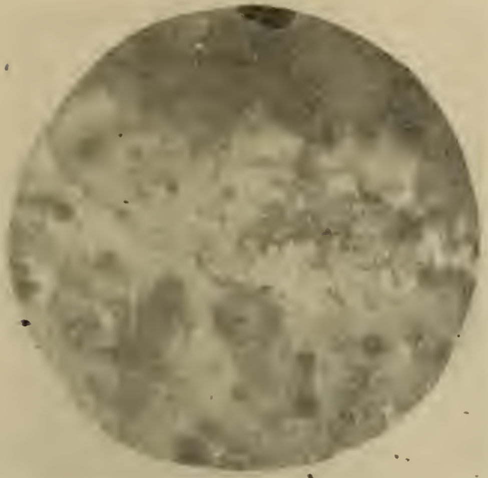


The results of assay were as follows:

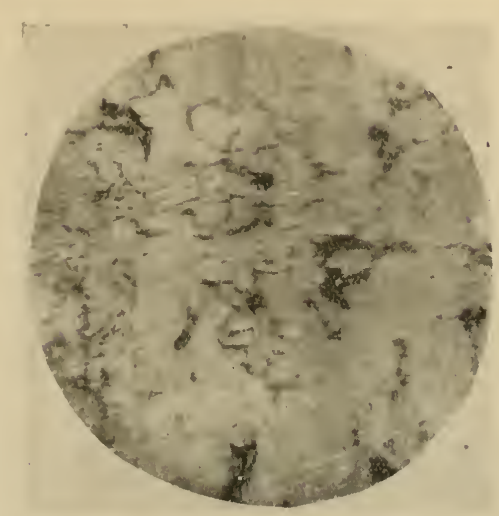
	Gold and Silver Per cent.	Gold Per cent.	Ratio: Gold Gold & Silver
Scrapings from surface ...	79·613	75·300	·9452
Slice No 1 $\frac{1}{16}$" thick	30·696	29·078	·9475
" 2 $\frac{1}{8}$" "	6·388	6·051	·9470
" 3 $\frac{1}{4}$" "	·016	·0152	·9500
" 4 $\frac{1}{2}$" "	·0013	·00117	·9000
Slot No 5 $\frac{1}{8}$" "	nil	—	—
" 6 $\frac{1}{8}$" "	nil	—	—
Outside surface $\frac{1}{8}$" thick ...	·0010	undetermined	undetermined

In this case there is evidently considerable penetration of gold to a depth of over $\frac{3}{16}$ in., and while it is not suggested that the piece examined was representative of the whole retort (in the sense that a true sample is representative) it shows the presence of a very valuable quantity of gold which would not be removed by a mere scaling or chipping of the interior.

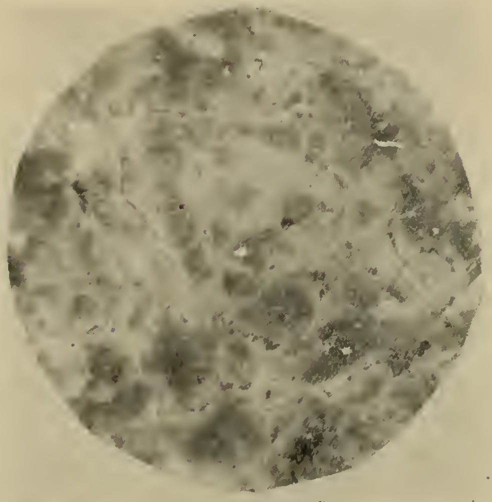
In this case also a microscopic section was prepared and a photograph of the inside edge is given herewith. (The section was exhibited under the microscope at the meeting).



Inner Surface of Retort × 175.



Near Inside of Tray × 175.



Middle of Tray × 175.

The inside surface is seen to be very rough and deeply corroded, doubtless through oxidation of iron and burning away of the graphite, and this has probably left minute cracks down which gold has percolated, most probably in the form of a little liquid amalgam, through the running over of amalgam trays, every time the retort was used. These cracks would tend to open and close on alternate heating and cooling of the retort, thus facilitating the passage of gold. Gold has been detected in association with the graphite under high magnification and therefore may be assumed to pass into the retort along the lines of graphite in the manner suggested. Although no gold was detected in the samples taken from the centre of the mass, it is possible that it exists there in minute quantity, since, although care was taken to prevent salting, gold was found on the exterior. This might point to the passage of gold slowly through the iron till it came to the outside, when of course it travelled no further and accumulated there; but, on the other hand it is also possible that it escaped through the larger cracks which ultimately led to the disuse of the retort. A piece of amalgam tray (cast iron) was also sectioned and examined and two photographs are given herewith. One of them just shows the inside edge and under the microscope particles of gold can be detected among the graphite and minute cracks: the other shows the interior structure and it requires no great stretch of imagination to picture the probability of gold travelling into the mass along the surface of the plates of graphite into the position in which, under the microscope, it can occasionally be seen. The specimen was obviously rich in gold, but no assays were made and the actual amount present cannot therefore be stated.

In conclusion, the authors would express their thanks to Mr. Fraser Alexander for assistance in the matter of obtaining specimens and trust these brief notes will prove to be of interest.

APPENDIX.

Refer to

De Kalb, *Engineering and Mining Journal*, Aug. 11, 1906.

Bayliss, Transactions, *American Institute of Mining Engineers*, Vol. xxvi., pp. 33 and 1039.

Collins, Transactions, *American Institute of Mining Engineers*, Vol. xxvi., p. 1041.

Brown, Transactions, *American Institute of Mining Engineers*, Vol. xxvi., 1043.

Read, Transactions, *American Institute of Mining Engineers*, Vol. xxxvii., p. 76.

Thomae, Transactions, *Institution of Mining and Metallurgy*, Vol. xvii., p. 482.

Halse, Transactions, *Institution of Mining and Metallurgy*, Vol. xvii., p. 486.

Goddard, Transactions, *Institution of Mining and Metallurgy*, Vol. xviii., p. 495.

Breakell, Transactions, *Institution of Mining and Metallurgy*, Vol. xviii., p. 496.

Richards, "Ore-dressing," Vol. ii. (1903), p. 766.

Mr. R. G. Bevington (*Past-President*): I should like on behalf of the Society to extend a hearty vote of thanks to Prof. Stanley for his most interesting paper. This is a subject which has been very much debated amongst metallurgists, it having been most difficult, owing to lack of apparatus, means of section cutting, etc., to arrive at any decision on the point. It is exceedingly interesting to find that Prof. Stanley has been able to demonstrate to us that the gold *does* penetrate both the copper plates and the iron retorts. Of course, the old idea was that the amalgam soaks through the copper plate, but that, I think, is an idea which has been exploded by Prof. Stanley's investigations. The gold simply gets into the copper by virtue of the porosity of the latter and penetrates the smallest cavities.

With regard to the retorts, it appears that through the burning out of the graphite, the gold has an opportunity of finding its way into the minute cavities left.

Mr. John R. Williams (*Past-President*): I have much pleasure in seconding the vote of thanks. I would like to ask the author if he considers it probable that lead amalgam forms *in* the plate. It used to be a trade secret that copper had lead added to it for rolling into sheets.

Prof. G. H. Stanley (*Member of Council*): I do certainly think that it is possible for that to occur. Possibly the mercury has combined with the lead to a certain extent, and formed a soft pasty amalgam which has travelled to the bottom of the plate.

Mr. W. R. Dowling (*Vice-President*): The results obtained by the scaling of amalgamated copper plates go to prove the correctness of Prof. Stanley's conclusions. Six tube mill plates were put out of action in the Simmer East mill at the reorganisation of the amalgamating arrangements on the reduced area, non-shaking basis.* The amalgamated surface of each plate measured 12 ft. × 4 ft. 7 in., the total area of the six plates being 328·5 sq. ft. A preliminary steaming yielded 1221 oz. of amalgam yielding 318·8 oz. fine gold, this after an average run of 42 days since the previous steaming. The plates which had been in action $2\frac{1}{3}$ years were then scaled, the following being the procedure. The plate was heated over a wood fire to drive off all mercury, the heat being just sufficient to char saw-dust thrown on the plate. The plate was then removed and whilst fairly warm was painted with strong hydrochloric acid. After standing overnight the plate was then painted with a mixture in equal quantities of a saturated solution of saltpetre and sal ammoniac and heated again over a bright wood fire. At this stage the scales rise, and after removal of the plate from the fire, may be scraped and brushed off. The scale was melted down like zinc gold slime with a flux of borax, manganese dioxide and sand and yielded bullion assaying 645·8$^{0}/_{00}$ of fine gold and 20$^{0}/_{00}$ fine silver. The first scaling of the six plates yielded 73·62 oz. fine gold. To determine whether any more gold could be removed the scaling operation was repeated a second time but this yielded only 2·25 oz. fine gold. Four holes were then bored through each plate and the borings assayed. The average of all the assays was gold 1·4 dwt. per ton and silver 34·4 dwt. per ton. It appears that the scaling removes practically all the gold as the total gold content of the six plates weighing 3,318 lb. was only 1·16 oz. The silver assay of the copper plate appears high. Unfortunately no information is obtainable as to silver possibly existing in the copper before being installed, but as these plates were originally set with silver amalgam before use, this fact may account for the comparatively high silver value.

The above information obtained on the practical working scale agrees with Prof. Stanley's results that the gold content of amalgamated copper plates exists mainly on the surface and that very little gold penetrates the body of the copper. This position, however, does not alter the fact that considerable gold is carried on the plate as setting as shown by the following table.

Six plates 12 ft. × 4 ft. 7 in. Total area 328·5 sq. ft. Total weight 3318 lb.

I understand that Dr. W. J. Sharwood, of the Homestake Mine, and a Corresponding Member of Council of our Society, did some work a few years ago on the absorption of gold by amalgamated copper plates which showed the gold coating to be merely superficial, and it is to be hoped

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* This *Journal*, "The Amalgamation of Gold in Banket Ore," March, 1911.

	Fine Gold.	Fine Gold per Plate.	Value.	Fine Gold per Sq. Ft. of Surface.
	Oz.	Oz.	£	Oz.
Steaming	318·80	53·13	223	0·97
Sealing	75·87	12·64	53	0·23
In copper	1·16	—	—	—
Total	395·83	65·77	276	1·20

that he will give the Society a contribution to this discussion.

Mr. E. J. Laschinger (*Vice-President*): I consider that the work which Prof. Stanley has carried out is highly appreciated by the members of this Society. In connection with the microscopical work, I want to comment on the extraordinary facility with which mercury gets into the most minute holes. It is well known by engineers who have had to do with mercury that it is most difficult to get a vessel that will contain mercury without leakage, so many kinds of vessels being made of material containing a compound capable of forming an amalgam or having minute pores or channels through the metal. Prof. Stanley's slides show that mercury has penetrated along the smallest line of division which has been left between even minute crystals, or in the pores and openings left by the burning out of the graphite in the cast iron, and I think that it points to the fact that we should not in the matter of retorts and amalgam trays simply take any kind of cast iron. It should be the subject of special investigation to decide what is the best kind of metal that should be used for amalgam trays and retorts.

NOTES ON THE TREATMENT OF MILL CONCENTRATE.

(*Read at July Meeting, 1911.*)

By R. Lindsay (Member).

REPLY TO DISCUSSION.

Mr. R. Lindsay (*Member*): The paper has failed in its primary object, namely, to promote a discussion, and I agree with Mr. W. R. Dowling when he says* that the usefulness of this Society is discounted when members, who can come forward with interesting and useful data, will not do so.

Mr. Graham has pointed out that an extraordinary want of uniformity exists in dealing with this by-product, and I may add, other by-products on our mines, and one of the objects in writing the paper was to assist towards some sort of standardisation in treating this rich material.

Mr. A. F. Crosse does not consider the little tube mill to be the most efficient mechanical device for grinding pyritic material, and from the point of view of fine grinding only, I confess to a liking for his rotary acting machine with oscillating muller, but in this particular case the tube mill has a usefulness apart from fine grinding in that it acts well as a preliminary trap for mercury and amalgam and deserves consideration. I am afraid it is not always realised that the mill concentrate often carries considerable quantities of mercury and amalgam, and the importance of recovering them before the cyanide treatment begins is self-evident.

I have to thank Mr. J. E. Thomas for the description of his mill concentrate plant and method of procedure at the Simmer Deep-Jupiter joint mills. Of course, not having the convenience of compressed air, he has to resort to circulation, then settlement and decantation, thus entailing extra initial expenditure for increased vat capacity. Mr. Thomas passes the tube mill effluent over two stationary copper plates, which I do not think are so efficient as the batea for trapping amalgam and mercury.

The mill concentrate plant at Geldenhuis Deep has now been in operation for twelve months, and by way of supplementing the figures given in the paper, I may say that during this period there have been recovered 382 oz. (= 5·4 oz. per ton) of mercury and 4,442 oz (= 63 oz. per ton) of amalgam, yielding 1148·38 oz. (= 16·4 oz. per ton) fine gold, of which a considerable proportion was obtained from the tube mill. About 70 tons of concentrate were tube-milled during the twelve months.

The cyaniding results for the same period average as follows:—

Before treatment by cyaniding ... 205·47 dwt. F.G. per ton
After treatment by cyaniding ... 3·88 „ „
Theoretical extraction 98·1%
Grading +100 −100+200 −200
= 0·30% 0·93% 98·77%

No mercury or amalgam has been found in the Crosse tank during the last six months.

THE ASSAY OF GOLD-BEARING CYANIDE SOLUTIONS BY ELECTROLYSIS.

(*Read at September Meeting, 1911.*)

By C. Crichton, F.C.S.

DISCUSSION.

Dr. W. J. Sharwood (*Corresponding Member of Council*): In his interesting account of electrolytic assaying Mr. Crichton refers to an earlier article on the subject in an Australian journal. This was probably the description by Mr. A. M. Henderson, of Victoria, included in a paper by F. B. Stephens on the treatment of sulphides at the Cassilis mine, read in 1905 before the Institution of Mining and Metallurgy, but printed in pamphlet form only, not appearing in the yearly Transactions. It was, however, reprinted entire in the *Mining Magazine* (New York) of August, 1905, Vol. xii., p. 142, and the electrolytic assay method was copied by several journals.* Henderson's method differed from the author's in some details, such as the use of an iron anode in place of carbon, in coupling a number of assays in series instead of in multiple, and in the use of a smaller cathode—2·5 in. high by 1 in. diameter, which is about one-third the size of the author's. He also directed the scorification of the cathode with some additional lead, instead of cupelling it directly.

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* See this *Journal*, November, 1911.

Previous to this publication I had made some unsuccessful attempts to make electrolytic assays of a cyanide solution carrying a great deal of copper, and about ten times as much silver as gold, using lead foil for both anode and cathode; the precipitation in this case being imperfect even when the assays were left running overnight. After reading of Henderson's method I again tried to utilise it for some comparatively low-grade solutions, containing lime and some copper with a large amount of thiocyanate, but the results were decidedly inferior to those obtained by precipitation with zinc-dust and lead acetate, followed by treatment with hydrochloric acid and scorification. The original paper only claimed a reduction from 10 dwt. or 15 dwt. (say, $10 or $15) to "below 3 grains" (say, 10 cents) in four hours, which was called "very complete"—in fact with this rich solution it would correspond to 99%.

In order to compete with the zinc and lead acetate method in the matter of time it would be necessary to have the cathodes ready for scorification in about 2 or 2·5 hours. The following are some of the results actually obtained with the above solution, using the method exactly as prescribed by Henderson, excepting as to time, there being ample free cyanide present. The figures refer to gold only.

	Original Solution by Zinc-Lead Method Mgms.	Henderson Electrolytic Method.			
		Hours	Precipitate Obtained Mgms.	Assay of Residual Solution by Zinc-Lead Mgms.	Per Cent Precipitated.
1	1·52	2·5	1·24	—	81·6
2	0·50	2·5	0·30	—	60·0
3	1·45	7·	1·42	0·02	98·5
4	0·40	7·	0·38	0·02	95·0

It seemed impossible, even when a much longer time was allowed, to secure as complete a precipitation as was possible by other methods, and with "barren" solutions the percentage remaining in solution was too large for even approximate accuracy.

The original description reads "the current used is from 0·06 to 1·2 amperes, the fouler the solution the higher the current density that can be safely used. The gold comes down as a bright yellow deposit." In my own tests a 110-volt D.C. lighting current was used, with lamp resistance, and two beakers in series, but the solution resistance must have greatly increased during deposition, as the current fell in the case of tests I. and II. from 0·45 to 0·33 ampere. It was difficult to fold the lead cathode without mechanical loss of the deposited metal (gold, copper, etc.) some of which was very loosely adherent and tended to fall off in the beaker, while a large amount of ferrous hydroxide formed, with apparently some Fe_3O_4. No doubt the character of the solution influenced the results, as the electrolysis of the thiocyanate yielded large quantities of sulphate, reacting with the lime to form a deposit of calcium sulphate and also acidifying the solution, and my criticism must be understood as applying only to the use of the method in this particular case and with the iron anode.

The only carbon at that time available as anode was an inferior lighting carbon, which gave a brown colour to the solutions. I would suggest the use of anodes of Acheson graphite, which is a nearly pure and very dense graphite produced in the electric furnace, and which in other cyanide work has proved almost absolutely insoluble and non-absorbent, and which does not precipitate gold itself when left standing in a cyanide solution, as the author has pointed out that ordinary carbons are apt to do. His use of a larger anode is probably an advantage.

In connection with the electrolytic assay of solutions, some data on their conductivity may be of interest. A series of tests were made, during several days, of working solutions from various stages of the treatment of sand, on the chance that determinations of the relative conductivity might throw some light on the condition of the solutions. All that was apparent, however, was the fact that the conductivity followed the total alkalinity more closely than any other factor, as may be seen from the following tabulated results. Conductivity was measured in the usual way, using a stretched wire Wheatstone bridge with interrupting coil and telephone, a N/50 solution of salt being taken as standard. Alkalinity was determined with phenolphthalein indicator. The other constituents, of which thiocyanate was the most important, were present in quantities nearly proportional to the cyanide, and not to the alkalinity.

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* This *Journal*, December, 1905, Vol. vi., p. 200.

Journal of the Society of Chemical Industry, September 15, 1905, Vol. xxiv., p. 912.

Mining and Scientific Press, July 1, 1905, Vol. xci., p. 5; also discussed April 14, 1909, xcii., p. 259.

Engineering and Mining Journal, August 5, 1905, vol. lxxx., p. 207.

Electrical Conductivity of Working Cyanide Solutions.

	Available KCN Percentage.	Alkali.	Conductivity.	Ratio Conductivity Alkali.
Conductivity standard N/50 NaCl (1·17%)	—	—	100	—
Alkalinity standard N/10 NaOH	—	100	—	—
1 Pregnant solution...	·09	17	160	9·4
2 ,, ,, ...	·09	16	138·5	8·65
3 Barren solution (same as No. 2)...	·09	16	138·5	8·65
4 Pregnant solution...	·075	14	134	9·6
5 First solution turned to weak sump ...	·04	14	123·5	8·85
6 First turned to low sump after beginning treatment ...	·005	12	113·3	9·45
7 ,, ,, ,,	Trace	10	86·8	8·7
8 First solution turned to weak sump ...	·04	9	84·2	9·35
9 Final low solution	·02	8	79·	9·8
10 Pregnant low solution (mixed) ...	·03	7	61·	8·7
11 Barren low solution, same as No. 10 after precipitation, tested after long standing exposed to air, some $CaCO_3$ having been precipitated ...	·02	5	46·6	9·3
Average ratio ...				9·1

It will be seen that the conductivity is nearly proportional to the alkalinity of the solution, only one of the eleven samples showing a variation of more than 5% from a constant ratio.

Dr. J. Moir (*Past-President*): I should like Mr. Crichton to tell us before the discussion closes why he uses such a high voltage in carrying out this electrolysis. By the use of 8 or 10 volts he has a voltage which is in excess of that required to give off hydrogen at the cathode, and this seems to me to be a disadvantage, because the hydrogen bubbles must tend to dislodge some of the gold. Only about half a volt is enough to precipitate gold and leave all the other metals in solution, whereas anything above 1·3 volt, I think, will liberate hydrogen and also precipitate other metals from the solution.

Prof. G. H. Stanley (*Member of Council*): I should like to suggest that experiments be made on the use of rotating anodes or cathodes for this assay of dissolved gold. It is well known that in copper assays the time occupied can be reduced from say ten hours to as many minutes, and it seems to me that in the case under consideration it might be possible by the adoption of such a method to very greatly diminish the time required.

NOTES ON HIGH DUTY GRAVITY STAMP MILLS.

(*Read at October Meeting*, 1911.)

By Peter N. Nissen (Member).

DISCUSSION.

Mr. E. J. Laschinger (*Vice-President*): It is with a feeling of regret that I find it impossible to give this paper the full discussion that the subject warrants, but lack of time compels me to confine the scope of criticism.

Let us first analyse the basis (given in Table I.) of the author's deductions and claims. It is evident that tests such as those whose results are summarised in this Table cannot be made with a high degree of precision in measurement. The difficulty of measuring with a close degree of accuracy large quantities of mill ore can only be appreciated by those who have attempted the task. So it is also with the grading analysis of such ore. The quality of the ore as regards hardness or crushing resistence is variable and of unknown relation to the results obtained. The measurement of power absorbed also presents difficulties, involving in cases of this kind some estimating as regards belt losses and journal friction. It is also hard to say whether the stem guides were in normal or equal condition throughout the tests for each set of stamps so as to be strictly comparable throughout. The grading analysis of the crushed ore products, however, can be arrived at with a fair degree of accuracy if proper care be exercised in taking the samples. The very close agreement of the grading results in all the tests, notwithstanding the difference in screens used, is worthy of note.

If members will kindly follow me through the tedium of a detailed analysis of some of the figures given in Table I, some curious anomalies which become apparent may be explained. This analysis should be of value to anyone interested in crushing efficiency and of special use in the case of further tests on comparative crushing trials with stamps.

Taking the figures on falling weight, height of drop and number of drops per minute as given and calculating the theoretical horse power, we obtain the results set out in columns 2 and 5 of Table "A" herewith. From actual tests conducted on stamps in good working order, I am satisfied that a mechanical efficiency of 87½% is a reliable average figure to assume, *i.e.*, the ratio of the power represented in the lifting of the weight of the stamps to the power actually applied at the periphery of the camshaft pulley should be the ratio 87½ : 100. On this assumption we obtain the probable horse power for the Nissen and City Deep stamps shown in columns 3 and 6 of Table "A." In columns 4 and 7 are shown the figures of horse power as taken from the author's Table I. :

TABLE "A."—Horse Power per Stamp.

Test No. I.	NISSEN STAMPS. Theoretical. II.	Probable III.	Actual Table (1) IV.	CITY DEEP STAMPS. Theoretical. V.	Probable VI.	Actual Table (1) VII.
1	4·27	4·88	4·09	4·00	4·57	4·21
2	4·26	4·86	4·09	3·98	4·55	4·21
3	5·11	5·84	4·95	4·10	4·69	4·10
4	4·96	5·66	4·95	—	—	—
5	4·40	5·04	4·23	3·81	4·35	4·65
6	4·40	5·04	4·23	3·81	4·35	4·65

From an examination of Table "A," it is at once seen that the figures given by the author for the actual horse power consumption are in all cases of Nissen stamp tests less than the theoretical horse powers. This is, of course, absurd, but there is not sufficient reason for assuming that the actual power figures given in the author's paper are therefore also absurd. A reasonable explanation is forthcoming by assuming that the height of drop as given in Table I. is the set drop and not the actual average measured drop of the stamps. It is unfortunate that the author did not mention this.

Assuming the above explanation as fact, we may now calculate from the power figures to find the probable average drop of the stamps during the various tests. The ratio of the actually measured horse power to the probable horse power is then the ratio of the probable average drop to the set drop. Applying this argument, we obtain the figures given in Table "B" herewith. The above basis I consider fair, as it is not reasonable to assume that the Nissen stamps would have mechanical friction losses appreciably different from those of ordinary stamps under equal conditions of maintenance and attention.

TABLE "B."—Drop of Stamps in Inches.

Test No. I.	NISSEN STAMPS. Set Drop II.	Probable Average Drop. III	Depth of Ore on Dies. IV.	CITY DEEP STAMPS. Set Drop V.	Probable Average Drop. VI.	Depth of Ore on Dies. VII.
1	8·50	7·12	1·38	8·50	7·83	+0·67
2	8·50	7·15	1·35	8·50	7·86	+0·64
3	8·75	7·42	1·33	8·75	7·66	+1·09
4	8·50	7·43	1·07	—	—	—
5	8·50	7·14	1·36	8·50	9·09	−0·59
6	8·50	7·14	1·36	8·50	9·09	−0·59

From an examination of Table "B," it is at once seen that there is an anomaly in tests Nos. 5 and 6 for City Deep stamps, and Table "A" also shows that for these tests the horse powers are very much higher than one would have cause to expect. This might be due to undue friction, say, in the stem guides. The anomaly is reflected in Table "B" where, since normal conditions as to frictional losses are assumed, the stamp shoes would have to drop more than half an inch below the level of the dies in order that the City Deep stamps in tests 5 and 6 could absorb the power with which they are debited.

It is therefore clearly unfair to use the power figures on City Deep stamps for tests 5 and 6 for true comparison with those for the other tests on City Deep stamps or with any of the Nissen stamp test figures.

Very interesting studies result by comparing the probable average depth of ore on the dies (Table "B") with the grading of feed ore (Table I.); but it must be remembered that such comparisons are intimately connected with the water to ore ratio and height of discharge, as well as the effect of position of screen with regard to the stamps, which latter condition is characteristically different for Nissen stamp and City Deep stamp constructions. In connection with these studies, it is the more regrettable that the results of tests 5 and 6 on City Deep stamps cannot be used. Further also, it is not certain that the data collected and set down in the author's Table I. were intended or can strictly serve for such close investigations.

The lack of finality of decision on comparisons indicated above should open a field for very useful and valuable experiments to elucidate points in stamp milling efficiency on which we are very much in the dark at present.

I therefore take this opportunity of saying that the author's remark as to the tests carried out at the City Deep being "exhaustive" is by no means justified.

One important deduction seems fairly clear, and that is that the Nissen stamps maintained a thicker layer of ore on the dies during the tests than the City Deep stamps and thereby saved power without any sacrifice of stamp duty but rather with an increase of duty. On the whole therefore, the net gain in power per ton crushed was enhanced.

This point, however, and in fact the whole value of the tests rest upon the assumption that both the Nissen and City Deep stamps were so fed with ore throughout the competitive trials as each to give their maximum possible duty. The author practically opened his paper by saying "to get the highest efficiency from each blow of the stamp, the conditions of feed are most important," but he has said nothing as to how the feed for both sets of stamps was regulated or attended to and checked during the trials, so as to give best conditions for both competitors. The author has probably accepted the condition of maximum duty as tacitly understood, but probably also some of our members are not so trustful as the author may have assumed.

In order to compare the tonnage and power figures, I have worked out the probable power figures and tonnages for 2,000 lb. stamps, in all cases assuming that these would vary in the ratio of 2,000 lb. to the actual weight of stamps in the various tests.

On equivalent bases of 2,000 lb. falling weight, we have the following :—

TABLE "C."

Test No	Duty, Tons per Stamp Day		H.P per Stamp.	
	Nissen.	City Deep.	Nissen.	City Deep.
*1	25·4.	19·6	4·23	4·53
*2	27·8	21·4	4·25	4·53
†3	32·7	24·5	4·41	4·24
*5	30·0	23·6	4·24	5·24
†6	37·9	27·4	4·25	5·24

From an inspection of these figures it will at once be seen that there is quite a variation in the duty figure for the different comparable tests 1, 2 and 5, but a quite permissible variation which should not throw any serious doubt on the accuracy of the results if averaged up.

Coming to the power figures, the Nissen stamp figures show a marked regularity for the five tests, mesh of screen making no difference as to their comparableness with one another. The City Deep stamp figures on power disclose an anomaly (already mentioned) inasmuch as the horse powers given show a variation of 21% between the maximum and minimum, whereas the Nissen stamp powers vary only by 4%.

In tests such as these it is a sound principle to take it for granted that the figures arrived at should be used only to guide the judgment in making intelligent deductions therefrom. On this principle I think it fair to say that the Nissen stamp should average 4·3 h.p. and the City Deep stamp 4·5 each. For tests 1, 2 and 5 the duties crushing through 9-mesh screen would be at about the average, say, 27·7 tons per day for Nissen stamps and 21·5 tons per day for City Deep stamps. The probable average power consumptions would therefore be :—

For Nissen stamps 3·81 h.p. hours per ton.
For City Dp. do. 5·01 h.p. hours per ton.

or a probable saving in power consumption for Nissen stamps as against City Deep stamps of 24%.

On the basis of the above figures, assuming the combined efficiency of motor, belts and other transmission to be about 80%, the saving of power would be about one electrical unit per ton crushed or say 0·6d. per ton as against the author's figure of 0·931d. per ton

On the same basis of comparison as above for the ⅜ in. mesh tests, the Nissen stamps compare even more favourably with the City Deep stamps.

It is, however, well to point out that figures of comparison based on a few crushing trials of only, say, 200 tons of ore each must not be taken as establishing exact figures for close comparison but as giving approximations to the truth.

Following the line of criticism I adopted in the impromptu discussion which followed immediately after the reading of the paper, that it was not fair to compare capital costs for 2,000 lb. Nissen stamps with 1,400 lb. ordinary stamps, I am of the opinion that the scheme of comparison brought forward by Mr. A. M. Kratz is fair and reasonable for the purpose and even favours the Nissen stamps. Taking 48 Nissen stamps at 27·7 tons duty each, working 30 days per month at 95% running time, we get a monthly tonnage of 38,000. Taking 70 ordinary stamps of the same weight at 21·5 tons duty, 30 days per month and only 90% running time, we obtain 40,600 tons per month. With this favourable allowance, which may not be strictly justified in practice, and assuming the ore bins in each case to hold say 2,600 tons of ore, we have the following comparisons: The building and bins

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* Tests 1, 2 and 5 were on 9 mesh screen.
† Tests 3 and 6 were on ⅜ in. mesh screen.

would cost at least 20% more for Nissen than ordinary stamp batteries; twelve mortar foundation blocks instead of seven would cost £1,000 more; battery framing would cost more, as the Nissen battery would have thirty-six king posts and twenty-four guide girts as against twenty-one and fourteen respectively; the mill machinery proper would cost as much and probably more because of the extra number of camshafts, pulleys, bearings, mortars, feeders, bolts, as against a lesser number of cams, stamps, etc.; the motors (assuming individual electric drive), countershafting and belts would cost more, as twelve 25 h.p. motors with switchgear and accessories would cost more than seven 50 h.p. motors and gear; for a line shaft drive the Nissen mill would also cost more; launders, floors and various incidentals would also cost more.

On the whole, estimating roughly, the Nissen mill would cost *at least* £3,000 more than the well known construction, so that the extra cost, counting interest and amortization of extra capital would completely wipe out the author's claimed saving on wear of shoes and dies.

As the author does not make any claim regarding saving in maintenance, which attitude I consider quite justifiable on his part, the net result, taking a fair and reasonable view of the whole question, is that with a new Nissen stamp mill as against a mill of the ordinary type there is a possible saving of about one electrical unit of power per ton milled. Having said this, I may also add that such an achievement is very creditable and deserving of more than proverbial faint praise.

The author and members of the Society are quite aware that the chief object of our Society is to obtain original data and new light on old subjects relating to Metallurgy and Mining and next after that to scrutinise and criticise the information put before us. Therefore, I have criticised, well knowing that the good points brought out in the paper require no special words of mine to drive them home to those interested in stamp mills. The author has certainly improved the gravity stamp as a crushing *unit*, and speaking from experience, I can say that he deserves encouragement and recognition of his efforts when working in a field peculiarly beset with difficulties and hedged about with discouragements.

It may be not inopportune in connection with this paper to mention a few facts in regard to the running of the City Deep stamp mill. Although the tonnage crushed has not averaged more than about half the rated capacity of the mill the costs of milling have been as nearly as possible 1s. per ton. Those costs include all wages, stores, power and maintenance over a period of one year and only two cam shafts have been broken.

Mr. R. A. Barry (*Member*): Having listened carefully to all the criticisms which have been made hereon, I feel more than ever confirmed in my original opinion, formed after a careful examination of this stamp and of the facts and figures supplied by the Engineers of the Central Mining and Investment Corporation, Ltd. The stamp appears to me to be eminently practical, built up on sound common sense ideas, entirely free from frills and fancies, with advantages which far outweigh the minor drawbacks of extra length and increased pumping. One fact which is not touched upon in the paper and may have some practical bearing upon the working costs of the stamp is that, with such a stamp, the spares of which will largely be supplied by one firm, the factor of competition (which we on the Rand have found to be so healthy!) will not hold so large a sway, and the price of spare heads, shoes, dies, etc., may come out heavier per lb. than those for the established type of gravity stamp.

There is one thing on which I would like to remark, a thing which I believe the general consensus of opinion will confirm, and that is the, to us, somewhat unwonted and extraordinary procedure adopted by those responsible for the Nissen stamp. These people brought their stamp into our midst quite quietly and with no beating of the big drum. They persuaded a powerful firm like the Central Mining and Investment Corporation, Ltd. that there was enough in their claims to justify a good trial and they sat quietly down for an extended period, quite satisfied to abide by the results of that trial. No promises, no shouting, no "jaw-tons" of energy expended, little or nothing was heard of the thing until the independent engineers who conducted the trial came forward with what appear to be unassailable facts and figures.

Now, whatever our individual opinions may be as to the merits of the Nissen stamp, and I fail to see in the light of the facts before us how such opinion can be anything but favourable, we must, I think, all agree that the Nissen method of putting things before us is one that could be adopted far more widely than ever it has been in the past, to the great advantage of the industry and all the parties concerned.

Mr. John R. Williams (*Past-President*): I have read Mr. Nissen's paper with a great deal of profit and pleasure, and as I have also had the opportunity of spending some little time in examining these stamps at work, I will make a few remarks as to how they strike me. At the outset I will reiterate a statement that I have often made before, *i.e.*, gravitation stamps take a lot of beating.

I have had some experience with stamps and other would be competitors in crushing machinery. I well remember a mining magnate years ago metaphorically tearing his hair out because he had ordered a 200 stamp mill, and had just heard of a new machine which, at a tithe of the first cost was going to do the work, and do it ever so much cheaper and better. I told this gentleman that he would still order more stamps. Well, since then we have had a very great number of these inventions, but I regret to say they have turned out to be only theoretical machines, and the gravitation stamp still holds its own.

Now, what is the Nissen stamp? A gravitation stamp having all the good points of the so-called California stamp, with many of its weak points obliterated, therefore, an improved gravitation stamp. Personally, I am charmed with the working of this stamp; a look at the dies as compared to these of the ordinary mill must tell one that it is a great stride in the right direction. I have no intention of going into the general construction but am myself satisfied that it is a sound, lasting device. I have talked it over with a number of interested people, and have only heard one or two objections, *i.e.*, that it takes so much room, and such a big building. My reply is that veld is cheap in this country, and that the ore bin space required will depend on the amount of ore crushed.

Now, a point that struck me was the evenness of the product discharged from these stamps, and the great importance of this will be seen at once on a concentration plant; it produces less slime and gives a larger proportion of even grade than any wet crushing machine that I am acquainted with. This must commend itself to every one who has had experience in ore concentration, and I take it that members of this Society are not looking at the gold industry (great as it is) as the only pebble on the beach. We are also in South Africa working copper, zinc, lead, etc, and our members being spread all over the world, will be found working all the metals. All of these metals, other than gold, have great possibilities in South Africa, and will need in many cases careful concentration, and there are also the so-called refractory gold ores. In some cases concentration will be found absolutely necessary, and I have no hesitation in saying that the Nissen stamp will be found greatly superior to the ordinary gravitation stamp. The value of this can be seen from the American mineral production which is over £180,000,000 per annum, of which about £18,000,000 is gold, whereas the copper output amounts to no less than £35,000,000, or about the total gold output of the Transvaal.

I have before preached from the text, "equal your possibilities." We have had some great improvements in parts of the gravitation stamps, such as feeders, cams, etc., all of which we are grateful for, but I am of opinion that by far the greatest improvement is shown in the Nissen stamp, and the only fault I have to find is that Mr. Nissen did not come here years ago. However, I feel certain that these stamps have a great future, and will yet play an important part in winning the metals in which South Africa is so rich.

SINKING OPERATIONS AT THE NORMA SHAFT, KATTOWITZ, UPPER SILESIA.

(*Read at August Meeting, 1911.*)

By B. C. Gullachsen, M.I.M.E., (*Member*).

REPLY TO DISCUSSION.

Mr. B. C. Gullachsen (*Member*): In putting this paper before the Society the author had hoped that it would have led to some discussion of various methods of sinking under difficult circumstances in this country. The author's thanks are due to Mr. Johnson for his contribution and for the questions he raises.

With regard to the timbering, this was made octagonal at the second attempt. Mr. Johnson suggests that it might with advantage have been made 12 or 15 sided; this might have proved successful but would have entailed a large amount of timber work. In a shaft only 10 ft. 9 in. diameter each wall plate would have to be cut very short, about 20 in., and in practice the getting of all these short lengths of timber into position in running ground, would have proved a matter by no means easy.

The reason why the engineers in charge of the work had to start the second attempt of sinking inside of the old broken timbering and rails, was that these had fallen in and become submerged in the sand, and therefore could not be extracted. If the cutting shoe had been placed on planks laid simply on the sand, instead of upon the ash and cement foundation, there would have been a great probability of the tubbing sinking too rapidly, or of the timber platform being broken due to the weight of the tubbing. The pressure ring consisted of two portions, called respectively the pressure ring and pressure shoe, the reason for the two parts being that the pressure ring was built entirely into the brickwork and the pressure shoe was bolted on to the face of it, and so protruded into the shaft above the tubbing, the hydraulic jacks being between the bottom of the pressure shoe and the top of the tubbing. The cost of the sinking was heavy, but it should be pointed out that the cost of £39 10s. included all the expenses from the time the first attempt at sinking was commenced, and not only the cost of sinking by the tubbing method.

No doubt many of the other methods suggested by Mr. Johnson would have proved successful, but in the author's opinion many of them would have been just as expensive, in fact, in the case of the freezing system, more so. The freezing system would also have occupied a long time, as so much preliminary work is necessary. The author was for nearly two years employed at the sinking at Dawdon Colliery Co., Durham, which was successfully sunk by the freezing system. The thickness of sand sunk through was 92 ft. 4 in., but in this case it was situated 371 ft. from the surface, the bottom of the sand being thus 463 ft. 4 in. from the surface. 28 bore holes were put down round each shaft, or 56 in all, each being 484 ft. deep, the bottom of the freezing pipes being thus 24 ft. below the sand. Brine was used in the freezing pipes, and over three months were occupied in freezing the sand.

ACCIDENTS IN TRANSVAAL MINES.

(With Special Reference to Falls of Ground.)

(*Read at November Meeting, 1911.*)

By James Chilton (Member).

DISCUSSION

Mr. W. Cullen (*Past President*): This paper comes at a most appropriate time. The author has given us a few figures—all of them official, by the way—and though in a general way they are quite well known to us, so well known indeed that many forget their significance, he has put them in such a setting that they compel us to think. We have been told the same thing before in other words, but the clarified style of the author will make us remember them. At the same time we must be quite fair, for there are undoubtedly factors which tend to swell our accident rate. The author and the Government Mining Engineer have both referred to the main one; indeed the Government Mining Engineer does it annually in his Report. I mean the presence of such a large proportion of semi-savages absolutely ignorant of the underground conditions. It would have been interesting and instructive if the author had been able to dissect the white from the black, bringing them to a common basis of comparison, but it is probably just as well that he did not do so, for undoubtedly the average man does not look upon the loss of a native life as a very serious matter. If on top of the figures which the author has given us are put those from phthisis and other diseases incidental to mining—which are most difficult to obtain, and never can be obtained with even a fair degree of accuracy—the wonder is that the mines ever get their labour complement at all. I am one of those who think that the mines would not attract so many labourers, either white or black, if the facts with regard to health and mortality were known. The high rate of wages paid to both whites and blacks have so far proved irresistible, and while I do not advocate any reduction in wages all recognise that if the accident rate were reduced and health conditions improved all round there would not be the same necessity for the high wages which now obtain. In England and elsewhere the tendency of miners' wages is to rise, but even discounting this rise the rate paid here is out of all proportion to that paid in England, and this state of affairs is brought about mainly by the dangers which seem to be attendant on mining here.

Just let me here offer a word of warning to our legislators. They may overdo legislation both by acts and by regulation. The English eight hours miners' bill, for which our present Governor-General is responsible, has caused more unrest and more all-round dissatisfaction among miners than any similar piece of legislation ever sanctioned by the British Parliament. It has resulted in the miners being not one whit better off than they were before; many say they are worse; the accident rate has gone up; the price of coal has increased, the consumer of course paying the increase, which he always has to do; the manufacturing costs have risen all round, and Britain has been handicapped in her export trade. Already there are indications that the shortening of working hours here does not meet with universal commendation.

Into the purely mining aspect of the paper I do not propose to enter for very obvious reasons, but with regard to the suggested appointment of additional inspectors of various kinds, I suppose that the same difference of opinion would manifest itself here regarding their utility as was shown in the course of the evidence given before the English Royal Commission. Obviously there cannot be a Government inspector in every mine, and even if there were, he cannot be in more than one place at the same time. To be any use his education must be very different from that of the ordinary miner, and his outlook on things mining must also be different. To my mind the present number of inspectors is ample for the work, provided there were stricter discipline in the mines. It is quite easy to convert inspection into inquisition. It would be presumptuous on my part to criticise the new mining regulations except in so far as they concern that aspect of them with which I am most familiar, but I may venture to hope that their tendency will be all in the right direction, provided the miner reads them and can master their import, points about which I have my doubts. Might I at this stage suggest to the Council that our mining men, of whom we see so little, be given one night all to themselves, when they can have a full opportunity of voicing their criticisms. That these are "frequent and free" I happen to know. Mining can never be a safe avocation, but it can be made comparatively safe and comparatively healthy. This, however, is an aspect of the question which has been discussed here more than once. I have an impression, which may be wrong, that blasting certificates are too easily obtained. In Europe a miner gradually develops, and when he reaches manhood he cannot help being a miner. Here most miners—I quote my mine manager friends—are merely rock breakers, and, this being so, they are liable to more risks than a fully fledged experienced miner would be. At least this is how it strikes me. I know this is a very thorny subject, but so long as our accident rate remains at its present figure, and so long as the matters to which I am now going to refer to remain as they are, just so long will the mines fail to attract suitable men.

However, leaving generalities on one side, I wish to suggest three causes which to my mind help to swell the accident rate. They are, firstly, the constant changing about of both managers and men; secondly, the casual manner in which natives are instructed in their duties; and thirdly, the almost entire absence of organisation in First Aid. The first is one of those subjects which is in everybody's mind but about which few for obvious reasons care to speak. True, the matter has been ventilated recently by our much respected member, Dr. Macaulay, but he did not touch on the aspect of the question as it affects this paper. I think it an axiom to state that a contented workman means good work. I venture to express the opinion that the majority of miners to-day are not contented. I know that as a class they are nomadic and clannish; that they like shifting about; that they not infrequently follow a manager whom they like, that they fling up a contract when they strike a bit of hard stuff; that they get "fed up" very easily, and that they do heaps of things which appear to us to be silly: but even after taking all these things into consideration there is far too much shifting about by compulsion. I could cite wholesale cases of this sort. You know them as well as I do and all I say is that this makes neither for contentment, nor for efficiency nor yet for safety. It pays neither the employer nor the employee. Whenever I meet my mining friends and not infrequently they are managers, it is always the same old story—do not know how long I am going to be on this job. Now my point is, provided a miner knows his job and does it, the longer he is on one mine the safer it is for all concerned. He knows and indeed must know when the roof is shaky better than a new man and he is in a position to communicate this to his learners or his boys. And so on all through the mine. On my particular job a new man means a reduction in the safety factor, and unless I am very much mistaken the same holds for mining. Precisely the same thing applies to the manager. If he is infected with that almost universal feeling of unrest he cannot be expected to do his best and goodness only knows his responsibilities are heavy enough nowadays.

Before I leave this point I wish to say that I hope my remarks will catch the eye of those who are responsible for the great industry of which we are all so proud. If ever there was a time when this matter should be looked squarely in the face it is now. This Society is not supposed to touch on political or financial matters, but I think that what I am going to say next may be included in that important branch of mining called "mining economics." Capital is necessary for labour and labour is just as necessary for capital. When these two do not hit it off—and no one will pretend that they hit it off here—there is trouble. I do think that without very much difficulty the unhappy relationship which at present exists could be so adjusted that the men would have a deeper interest in their work, they would give better work for their wages and indirectly this better understanding would minimise present day risks.

My second point is the casual manner in which natives are instructed in their duties. Now I am free to confess that my general views on the native question may not commend themselves to the majority of the members of this society, but whether they are right or wrong they have worked all right in practice at the Dynamite Factory, and I may illustrate the point I wish to make by drawing a little on my own experience. Visitors to the Dynamite Factory not infrequently remark on the large number of natives who are employed in the handling of explosives, and for that matter doing other work also, and just to make my position clear I ought to say that I would under ordinary circumstances be classed as a follower of one of our Johannesburg "papers" which in season and out of season harps on the question of white labour without however displaying very much practical acquaintance with it. Well I have only the one answer for my interrogation. Show a native only once what you expect of him, but do it thoroughly by people who can speak his "lingo," and he will do that day after day and month after month. The white man who in this country may be classed as unskilled, is too much inclined to exercise his ingenuity by devising short cuts to attain the same object, thereby introducing an element of danger. In other words for certain classes of work which demand no particular intelligence but, a rigid adherence to definite instructions, I prefer the native to the white. The economic aspect I am of course not speaking of, but I can make the general statement that the native is the basis of the whole of our economic system. I believe that my experience is just the same as all those who handle native labour; therefore, I do appeal to all those who are in the same position as myself to do whatever they can, and by whatever means, to instruct the native carefully and patiently in his duties and I know that the results will repay the trouble. A native life is not worth very much just now on the £ s. d. basis, but at the beginning of the last century a slave even in South Africa was a valuable asset in the household inventory. If the natives were as valuable to-day on a compensation basis as they were then, our death rate would show a remarkable diminution. It comes to the same thing if I say that if they were to demand compensation for accident or for loss of life, commensurate with the service which they render, there would be fewer accidents. Shareholders would see to that. Efficiency is a term that we are very fond of up here and so is another, "sound training." We think of them I am afraid in connection with the white man only, but surely ordinary terms like these are applicable to the black as well as to the white.

I am afraid that I may be accused of harping on one string when I refer to the third point. It has been put before our Society not once but many times, so frequently indeed that one despairs of ever getting anything done. I refer to the proper provision of first aid equipment including safety helmets, together with a trained body of men to work them. One reads now and again of heroic deeds in the way of rescue which make one proud of one's fellow men, but in the majority of cases if there were proper appliances with an intelligent staff to use them there would be no necessity to take the risks. No doubt all this would take a little trouble and would cost a little money, but I venture to assert that taking the Rand as a whole the saving effected in compensation would repay the entire cost of the outfit in a couple of years.

In the discussion on Mr. Penlerick's paper* I quoted what was being done on the continent and what it was proposed to do in England and let me add what is now being done. Surely it is high time we were waking up. At the exhibition which the Society is holding in February of next year there will be a very complete exhibit by all the principal makers of such articles which few out here have ever seen, and I do trust that one of the results of this exhibition will be a more general use of such appliances. I know well that much has been done, but there is still much to do in this direction. Before leaving this aspect of the subject I would like to say that all students of the School of Mines will, from next year, be compelled to go through a course of First Aid—which will include the use of safety helmets. The Board of Trade now insists on all ships' officers having a certificate for first aid.

The author in the course of his paper was good enough to refer to certain propaganda work of Dr. Macaulay and myself. I was pleased to learn that it has been productive of good, but at the same time it is work of a very exhausting and discouraging not to mention expensive nature. I would be only too glad to resume it, and so I am sure would Dr. Macaulay, but to men whose lives are already pretty full, it is not lightly to be undertaken.

There are many things one would like to say in connection with this paper and one could have wished that the author had not left so much unsaid, but there are few in this country who can afford to say all that they think, so I appreciate his reticence. His facts, as I have already said, have made us think. They certainly have made me do so. If I have spoken strongly, it is because I feel strongly that we, as an enlightened community, have this huge black mark against us. The English figures which the author has given us naturally include those appalling colliery disasters which go to swell the totals, disasters from which we out here are fortunately free. Just imagine how public opinion here would be aroused if the Johannesburg Municipality by its trams were to kill one person per month, or if owners of motor cars were to emulate their example. Why, there would be such an outcry that neither trams nor motor cars would be allowed to run a single day, and yet here every day of the year three human lives are sacrificed. Probably two-thirds of these are unavoidable—who can say—but who will defend our present figures?

I trust that I have not given offence to anyone in speaking as I have done. Most of those who control this great mining industry, whether on the financial or the technical side, I claim as personal friends. I am sure they cannot realise what is going on silently day by day. After the author's paper none of us can live in a fool's paradise, nor can we plead ignorance. I know full well all that has been done in the way of erecting comfortable dwellings, change houses, recreation halls and even baths, but man cannot live by these things alone. What I should like to see is the industry keeping ahead of legislation instead of lagging behind it.

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* See this *Journal*, Vol. XI., Nov., 1910, pp. 211-214.

Since these notes were written I have had an opportunity of studying the new mining regulations, but my remarks have been of such a general nature that they are not affected by them. The members of the S.A. Institution of Engineers are also engaged at the moment on the discussion of a cognate subject and I note with pleasure that great interest is being taken in the matter. After 40 one does not expect to get reforms carried out by wild talk or revolutionary methods, but I do think that the public conscience is being gradually awakened none too soon—and that those in authority are realising that that branch of mining economics which I shall christen "humanitarianism" has been left far too long unstudied.

I should like to add before concluding that it is perhaps unfortunate that discussions very similar in nature have been going on simultaneously in the two principal Technical Societies. They may seem to indicate in the minds of some people that there is hostility to the mining industry on the part of those who have taken part in these discussions. I wish to say on my own behalf that my feelings are quite the reverse. It is because I feel so friendly disposed towards the mining industry and those connected with it, that I have taken it upon me to point out what those in authority apparently don't see or at anyrate don't realise. I am strongly of the opinion that future economies in mining will be as much along the lines I have indicated as upon mining and metallurgical improvements.

Dr. D. Macaulay, M.L.A. (*Member*): I think it would be better if I reserved what I have to say on this very important subject until your next meeting. There are certain facts and figures which I wish to place before you, which will not be available until the beginning of the year, facts and figures which, I am obtaining through the courtesy of the Government Mining Engineer, and which will, I hope, throw considerable light on one aspect of the question. I have very much pleasure in being before you to-night, and I noticed with great interest Mr. Chilton's paper in the *Journal*. I think it is one of the most important subjects which this Society has had to discuss. You discuss with a great profundity of learning, which I am not able to follow, the absorption of gold by amalgamated plates. Well, the amount of gold absorbed by these plates is infinitesimal compared with the output. I also notice that you discuss the question of the duty of gravity stamps. I think it is equally within the function of the Society to discuss the duty and the gravity of the human factor in the work of production.

The last time I came before you was in connection with a cognate subject, that of safety measures in mining from the medical aspect, when I appeared in conjunction with Dr. Irvine There were then, and there are now, engineering aspects of the question with which we could not adequately deal. Everyone must regret the appalling loss of life which is constantly going on on these goldfields, probably due to carelessness and ignorance. It is one of the functions of this Society, which has been a lamp in the darkness in the past, to do its utmost to reduce this terrible loss of life, which is undoubtedly a blot on our industry.

Mr. T. G. Trevor, A.R.S.M. (Acting Inspector of Mines, Pretoria) (*Member*): As exemplifying Mr. Chilton's statement that a great cause of the high accident rate on Rand mines is due to "speeding up" and to the rush and hurry with which work is carried on in those mines, the following comparison between the accident rates, for the first nine months of the present year, of the Witwatersrand mines and the mines of the Pretoria Inspectorate is instructive:—

WITWATERSRAND.		PRETORIA INSPECTORATE.	
Average death rate per 1,000 per annum.			
Period 1911. January to September.	Gold Mines. 4·01	Gold Mines. 2·71	Base Mineral Mines. 1·66
		2·185	
Period 1911. January to September.	Springs-Brakpan. Coal Mines. 1·70	Coal Mines. 2·32	Diamond Mines. 1·10

To those who know both classes of mine it is obvious that, with the exception of the Sheba and the Worcester, the outside mines are in very much more dangerous ground than those on the Rand. It is also obvious that on small mines in outlying districts, which are worked with little capital and a minimum of white supervision, constant risks are run which are not, or should not, be taken by the highly organized and officered Rand mines. Notwithstanding this the Witwatersrand average is approximately twice as high as that of the outside metal mines.

The writer's position has given him unique opportunity of judging the reason for this and he attributes it largely to "hustle." Where one or even a few men are at work each one attends to his own safety, even if they are all natives; where there is a crowd, however, everyone takes the safety of his surroundings for granted. In travelling ways it is the same, travelling alone each man looks to his foothold, whereas in a crowd he would just push along and frequently come to grief. It must also be remarked that in outside mines far more reliance is placed on natives than is the case on the Rand and that these natives rise to the occasion and justify this trust. On most of the outside mines the majority of the miners remain for years and their principal natives are also very permanent; these natives thus get to know the mine and its methods of timbering and can use their judgment nearly as well as their masters, and are quite on a different footing to the sheep-like herds of raw negroes who are hustled into a Rand stope. Another cause for the low accident rate on the outside mines is the fact that an accident on a small mine is brought home far more closely to the management than on a large Rand mine. On a small mine the manager is sure to see the result of every accident in the shape of the maimed or mangled victim, whom he probably knows personally. On a Rand mine the manager seldom, and the consulting engineer and directors never, see the direct result of an accident. On a small mine also where the labour supply is kept up by voluntary labour an accident will at once cause a number of the natives to give notice and will prevent fresh natives joining; while on the large mine with contract—W.N.L.A.—labour the accident rate has no effect whatever on the labour supply. In every direction therefore, sentimentally, personally and financially, an accident on a small mine is brought home to the management, and in all directions it is equally bad business. On a large mine, however, where the capitation fees are paid from the head office and where every employee is insured, an accident is by no means the impressive event which it is in a small mine, and even with the most humane officers—pressed as they are by the constant effort to keep the mill running—the general accident rate is apt to be forgotten and unnoticed.

In the writer's experience a very frequent cause of accidents is the difficulty met with in enforcing care and forethought amongst both the white and coloured workmen. It is now effectually prohibited to strike or thrash natives and yet no other means of compelling obedience has been found. The mining regulations are all aimed against specified actions or inactions on the part of the management and white miners, but provide no penalty for that general carelessness which endangers the lives of others and often of the man himself. For such acts as shovelling down rock on persons below, tramming carelessly, dropping things off ladders, etc., there is no effective punishment.

At the Premier Diamond Mine there is a special rule framed under Section 4 of the "Mines, Works and Machinery Ordinance" to meet these cases. It reads as follows:—

> "Any person disobeying an order given in the interests of safety or doing anything either through wilfulness or carelessness in such a manner as to endanger the safety of others, his own safety, or the property of the Company shall be liable to instant dismissal and to prosecution under the mines, works and machinery regulations."

This rule has had an excellent effect amongst the natives, for if discovered doing any one of the dozens of acts of carelessness that lead to accidents the offender is promptly caught and reported, and on the next visit of the Inspector of Mines is brought up before his Court and suitably fined if found guilty. It is found that by this the white miners are enabled to have far more control over their boys than formerly; and as notices of all the cases and the fines inflicted are posted in two native languages in the Compound all natives have learnt what punishment they may expect for carelessness, and a general improvement in the petty accident rate has been the result.

Mr. J. Moyle-Phillips (*Member*): Compared with the English death rate ours is certainly appalling, but then the great majority of our mine labourers are not in the same category as the English mine labourer. Here all the unskilled and a good deal of the semi-skilled labour is done by Kafirs.

In England when a lad goes in for mining he is taken in hand by a skilled man or men, one or at the outside two, lads to one gang of men, here a gang of from 15 to 30 raw Kafirs is sent underground in charge of one skilled man. Consequently owing to the Kafir's ignorance of the first laws of self preservation in underground work he ofttimes walks deliberately into danger before the man in charge has a chance to warn him of, or describe the nature of the danger to him, and in the majority of cases the white man is incapable of explaining to the Kafir in his—the native's language. How many of our miners can ask a simple question in the Kafir's own dialect so that he can be understood? I mean if the Kafir is a *raw* one and straight from his kraal. I think this reason has a great deal to do with our minor accidents, also such as drilling into misfires, falling down stopes, and accidents appertaining to tramming, etc. Again ignorance as the cause of accidents might be applied—in a lesser degree—to the majority of our white miners. This has been most noticeable since the war; although I am pleased to see that, under the 3rd column of Table II., accidents due to carelessless or ignorance of the injured have materially decreased. I certainly think this is encouraging, showing as it does that our miners are gradually acquiring that knowledge of mining which is essential to the safety of themselves and their fellow workmen. Our present day *miner* is a man, who in the majority of cases, bases his qualifications on the fact that he is the possessor of a blasting certificate. He has picked up, I cannot say learned, mining in the space of two or at the outside three months as learner and this entitles him, under the system at present in vogue, to his blasting certificate. Once in possession of this he leaves the mine where he acquired the certificate and launches out in search of a contract on his own as a full fledged miner. It is quite sufficient to look at the names of men who are being continually fined for breaking the mine regulations to at once spot the kind of *miner* they are. I had one such applying for work just lately and on looking at his discharge ticket I found he had worked on about 9 different mines in the same number of months, his record of continuous employment being 10 days.

Regarding Column 2, Table II. The increase in the percentage here I attribute to our increased depth in mining, for with our outcrops and upper levels having been worked out and inadequately secured, it stands to reason that the deeper we go the more liable we are to falls of roof, and those of greater dimensions. Although the author proves by figures that the accident death rate in England has increased by 30% since the introduction of the 8 hours day I do not think the same can be said to apply here. For instance, take hammer boys. Even before the introduction of the 8 hours here these boys were seldom in the stope after 2 p.m. so the 8 hours system cannot be said to have affected them adversely. Then take machines. Under the 8 hours system the white man has to keep constant supervision over his boys whereas under the old 10 hours system he was away for lunch, from the time he left his working face to the time he returned, for close on 2 hours so as he started work at 7 a.m. and left off at 5 or 5.30 p.m. I don't see that his ½ hour less supervision under the 8 hours system has been the cause of the extra percentage in accidents under this heading. I quite agree with our miners who assert that the increase in accidents is due to the methods of "speeding up" or as I have heard it named "the deep level *tyelya**," but I do not think it is entirely due to the "rush of earning a living wage and consequently leaving little time for thought or preparation for an emergency." I think there is another and more obvious reason, viz., that of having to keep our big mills going. I maintain that at the present time the majority of our mines are overburdened by our mills and that in consequence of having to keep them supplied with ore we underground men have to take every available, and I am sorry to say in a great many cases also, a great many tons of what should be unavailable ore out. By unavailable ore I mean ore that should have been left in the stopes in the form of safety and boundary pillars. We will say that a certain amount of development has been done which, on paper, warrants the erection of a big mill; that is, a certain extent of ground has been blocked out, ready for stoping, this ground contains so many tons of ore and a corresponding mill is erected to crush it. Now what I would like to ask is, in calculating these ore reserves has any provision been made for the loss of ore left in the stopes in the form of safety pillars? I do not think so! for if so then why are so many of our stopes devoid of such pillars, or where such pillars have been left in, why are they of such an inadequate size as to constitute—in a great many instances—a menace to, instead of a safety for life and limb? Every ounce of ore has had to be extracted to keep the mill going, and woe to the mine captain who fails to supply it. Why is all our waste rock hoisted to the surface instead of being packed underground where it would be of use in supporting the roof? I will answer that question by quoting an instance which occurred under my observation. I was once put into an old stope with two machines where—before I could commence stoping—I had to take down some 150 to 200 tons of bad hanging. This waste rock was securely—as I thought—built into a strong pack which, in the ordinary way, would form a sound support for the roof, but before I was in that stope for 2 months those 200 tons of waste rock along with a goodly number of other tons of waste rock were trammed away and sent to the surface. Why! *To keep the mill going.* Thus was the underground work robbed of a safety pillar or pack to keep going a mill which was at least 25% larger than the mine could carry.

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* Zulu for hurry.

The inadequacy of safety pillars—both in size and numbers—is no doubt one of the primary causes of a stope subsidence to which may be added the occurrence of faults and dykes. In the majority of cases safety pillars are left far too small, and in such case contribute a menace instead of a safety to the stope. It must not be forgotton that 2 ft. around a pillar that has been cut around by machines—is useless as a support, as such a pillar is shattered to that extent. That is, take a pillar 12 ft. × 12 ft. that has been cut around by machines, it will be found that really only 10 ft. × 10 ft. can be regarded as sound, the perimeter outside this having been shattered by the blasting and thus rendered useless as a support. Again immediately a pillar has been cut around it begins to lose its efficiency what with the blasting in the stope together with the "weathering" which takes place and the greater strain which it is being called upon to bear, as the working face get farther away it will be found that what is left of 12 × 12 pillar after 6 months is not worth reckoning on from a *safe* point of view—in fact I have seen such a pillar entirely disappear in 4 months. But there are other causes—although small in themselves and which can nearly always be avoided—to which can be attributed the beginning of a subsidence and also all the principal causes of our "stope" accidents. First among these I would place the sinking and raising of winzes in the hanging wall. To the frequent practice of carrying a portion of the hanging wall in driving can also be attributed some of the "stope" accidents. I have found that when once the hanging wall has been broken into—and stoping has commenced—it not only entails the leaving of what would otherwise be unnecessary pillars, but is often the cause of a whole stope collapsing. I know that in sinking a shaft on the reef that—owing to the irregularities of the dip—breaking into the hanging wall cannot always be avoided, but in winzes, raises and drives it can be avoided and with proper supervision should never be allowed. Another cause is that in stoping, owing to the ignorance or negligence of the stoper a hole is allowed to penetrate the hanging wall and then blasted. When this occurs—more especially if it is a machine hole—the effect is generally the shattering of what was and would have been a sound hanging. A machine hole allowed to penetrate the hanging to the extent of 18 in. and blasted will do more damage that a 12 × 12 safety pillar can rectify. As I have said these things are preventible with proper supervision and under no circumstances should a hole which has penetrated the hanging be "blasted."

The meeting then terminated.

Contributions and Correspondence.

STOPE MEASUREMENTS.

In reply to Mr. Tonnesen's criticism of my contribution on the above subject, published in the *Journal* of Dec., 1910, I may state that the fact of having been so extremely fortunate in "always" being able to find an accessible position above or below a pillar and in intermediate drives from which the face of the stope could be observed with a theodolite, seems very extraordinary and enviable as well. Still, however marvellous, the fact exists. The intermediate drives found essential to the method adopted at the Simmer and Jack mine, in order to develop the steep stopes and at the same time preserve the traffic in the drives overhead, were of the greatest advantage to me in the subsequent survey of the stopes. My experience being limited to the Consolidated Goldfields Group of Mines I am unable for the purpose of theodolite survey to form a definite opinion as to the accessibility of the stopes of other mines of the Rand district.

Answering Mr. Tonnesen's question, "why does the author recommend an extensible bar for carrying an arm to which the theodolite is clamped?" I state that I do so because it has been found very useful and would offer something to "glue" the theodolite to in case no convenient pillar were at hand. An arrangement for this purpose has been in use for some time at the Witwatersrand Deep mine and has proved very effective in placing the theodolite for observation of the stope face. I agree with him that to some extent its use would be precluded in case an extreme diversity in the height of stopes existed; nevertheless it is a valuable adjunct. A modification in which the width of stope face is of no moment, used on the Simmer Deep mine, consists of a footwall plug on to which the bar carrying a modification of the usual tripod head is screwed. Possibly he could survey the stope shown in my Fig. 1.[*] more cheaply than by setting up a transit. I venture to state, however, that in the subsequent plotting, not only in order to ascertain the area stoped but to bring the horizontal working plan up to date, the transit method would give the required accuracy and greater speed, and would, in the general run of work, owing to fewer assistants, be found to be cheaper.

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[*] See this *Journal*, vol. xi., Dec., 1910, p. 261.

Regarding the method of least squares and calculus, I may state that I merely pointed out in the general discussion of methods that the equation of the dip line, and hence the dip at various points, could be determined in this way. I did not make any statement that it was a practicable method, at least not in the hands of a novice. If Mr. Tonnesen will read carefully he will see that in the next sentence in my paper the following occurs: "The most practical way, however, is as I have indicated in the following paper." The idea involved in the discussion pertaining to Fig. 2* is that the errors indicated were liable to occur owing to faulty judgment or carelessness on the part of the observer. I do not agree with Mr. Tonnesen that it is applicable to the theodolite method only. Sufficient illumination is usually obtainable, so that one standing above 100 ft. away can direct the proper placing of the point of sight. Face errors of this nature are of no great moment except when the contractors are being changed. Then an accurate determination of the face contour is necessary. For ascertaining the dip a few errors in selecting points on line of trend of the dip do not matter. The writer states that the surveyor is usually within 20 ft. of the face so that he can easily direct the placing of the tape. What an unnecessary waste of time and labour to be establishing stations so as to be within 20 ft. of the face. A station every 20 ft. or less!

Mr. Tonnesen states that the stope represented by my Fig. 3† would present equally great difficulties to the theodolite method. What these equally great difficulties are I cannot imagine. I have surveyed numerous stopes of this class without great undue difficulties. The stope as drawn is not an exaggeration in the least. The writer has made what he terms a section of my Fig. 3, and then made a statement that his drawing shows my figure to be an absurdity. This hybrid logic does not in the least substantiate the validity of the statement. The method reminds me of the pseudo-mathematician who established beyond cavil, at least in his own fourth dimensional world, the value of π to be exactly 3·00. He constructed a cylindrical vessel diam. 2a and height "a" hence of volume πa^3. Also a cubical vessel of edge "a" hence volume a^3 and, finding by actually pouring water from the cubical vessel into the cylindrical vessel that it would contain 3 times the amount contained by the cubical vessel, argued that $\pi a^3 = 3 a^3$ and hence $\pi = 3$. I beg to point out that my Fig. 3 is constructed on the isometric system of perspective. Without indulging in a dissertation on isometric perspective, in constructing a cross section of my figure, all distances must be multiplied by the secant of 35° 16′ This would give an entirely different figure from the phantasmagoria evolved by the writer. I beg to state that it is no uncommon occurence on several of the mines of the Rand to encounter stopes having a considerable portion of the footwall with radius of curvature far less than "47′."

I must admit a contradiction in applying the method of multiplying the average breadth of area stoped by length of dip line. Mr. Tonnesen agrees with a statement occurring in my paper that each portion of stope area should be weighted according to the dip corresponding to that portion. If the stope is of fairly uniform width from top to bottom then my method would apply without appreciable error. The method as a graphical one would apply to a stope of uniform dip or one of variable dips applying to each portion of uniform dip the length of the dip line corresponding to that portion. The method is not always the best one to use, but is of value.

In regard to having an acute angle of 55° I think Mr. Tonnesen extremely fortunate in "always" being able to carry a traverse forward with the direction of the stope so as to be able to give angles between 130° and 230°; as fortunate as I have been in being able to "glue" my transit to pillars and intermediate drives. Backsights of 20 ft. are not uncommon and when one knows that many precise connections are made in which sights of less than 20 ft. occur, one cannot conceive of any danger in using short sights in the theodolite method. I regret very much that I purposely chose a 20 ft. sight in order to exaggerate the effect of the one foot drop. I may state though I did use the one foot drop in order to minimize the effect of drop; it could have been made 2 ft. without undue exaggeration. I am at loss to comprehend how he can place his instrument vertically under a station and sight to a point vertically beneath the backsight station and to a point at the face, and plot the face of the stope in its relation to the drives and drive pillars. To be in a consistent plane of the reef the instrument should be held at a point on a line at right angles to the plane of the reef and passing through the peg and sighted to a point similarly placed at the backsight peg. I presume the position of the drive is shifted so as to fit in with the position of the instrument, or else a sight taken to where the observer judges the side of the drive to have been before stoping.

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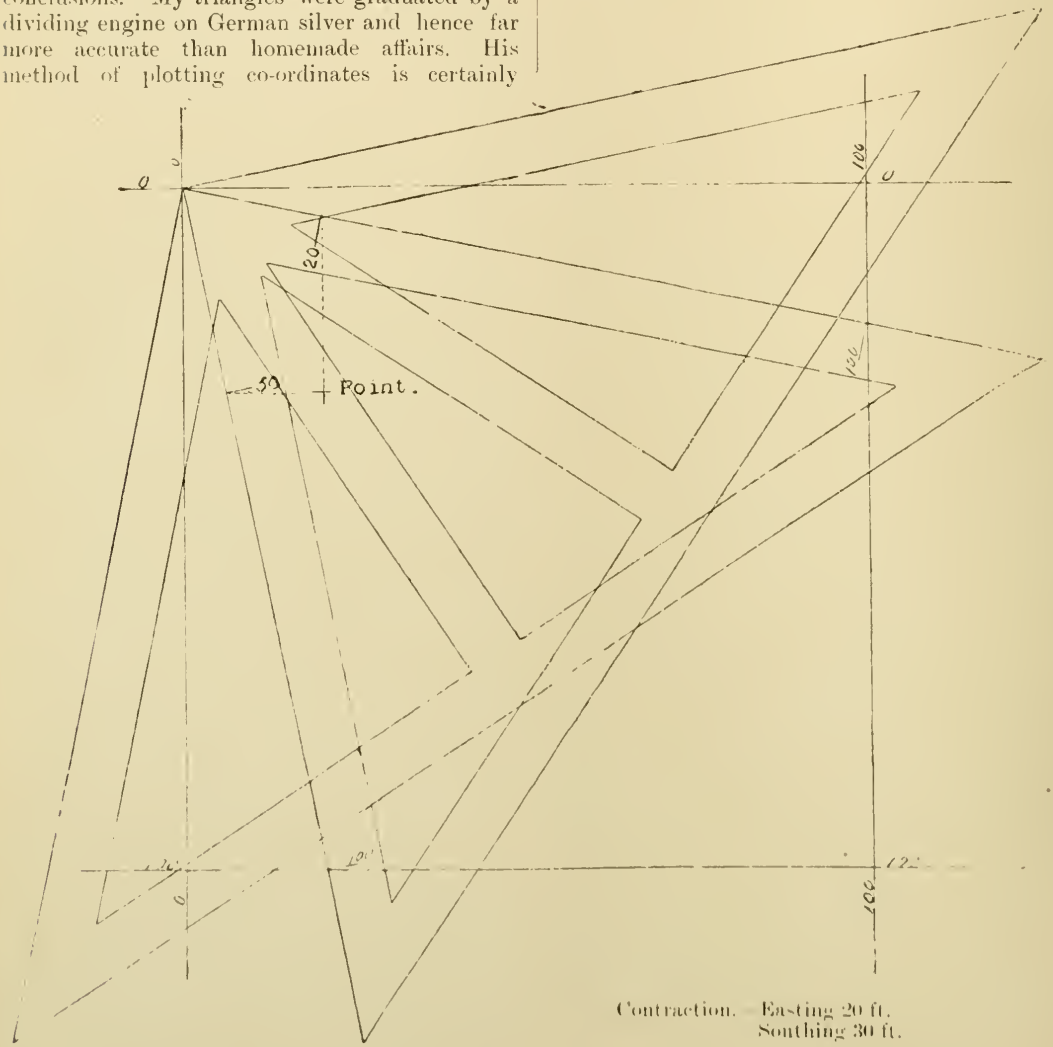
* See this *Journal*, vol. xi., Dec., 1910, p. 265.

† See this *Journal*, vol. xi., Dec., 1910, p. 266.

Mr. Tonnesen points out succinctly that I made two determinations of dip one 35° 17′ and the other 36° 24′ differing by 1° 7′. He seems to have neglected, in order apparently to establish a doubt in the minds of his readers, to mention that I fully explained why this discrepancy occurred. I showed that the mean of the two results was the same as the graphical determination. The triangles as selected were small and, as perhaps the points as selected may have been slightly off the plane of the reef, a small difference in elevation would cause some difference in the determination of the angles. The method in many cases is of practical value and far more precise than judging that one is holding his instrument in the plane of the reef.

My experience in regard to the plotting triangles is entirely at variance with Mr. Tonnesen's conclusions. My triangles were graduated by a dividing engine on German silver and hence far more accurate than homemade affairs. His method of plotting co-ordinates is certainly unique. For example I would infer if the contraction were 2 ft. in a co-ordinate square whose sides represented 100 ft. that the author would place the scale so that the 0 division would be one foot to the left of the first co-ordinate line and the 100 foot division one foot to the right of the second co-ordinate line, thus "meaning" the error due to contraction. A most extraordinary method. Surely he must recognize in this instance the co-ordinates must be reduced proportionally and then plotted. For illustration, suppose the easting were 20 ft. then the distance to be laid off would be 20 × 98 ÷ 100 = 19·6. The writer would plot 20 − 1 = 19 ft.; if he had a co-ordinate 99·5 he would plot it in the next square or judge where it ought to be in its own square. With my triangles this reduction can be made mechanically and is equally applicable



Contraction. Easting 20 ft.
Southing 30 ft.

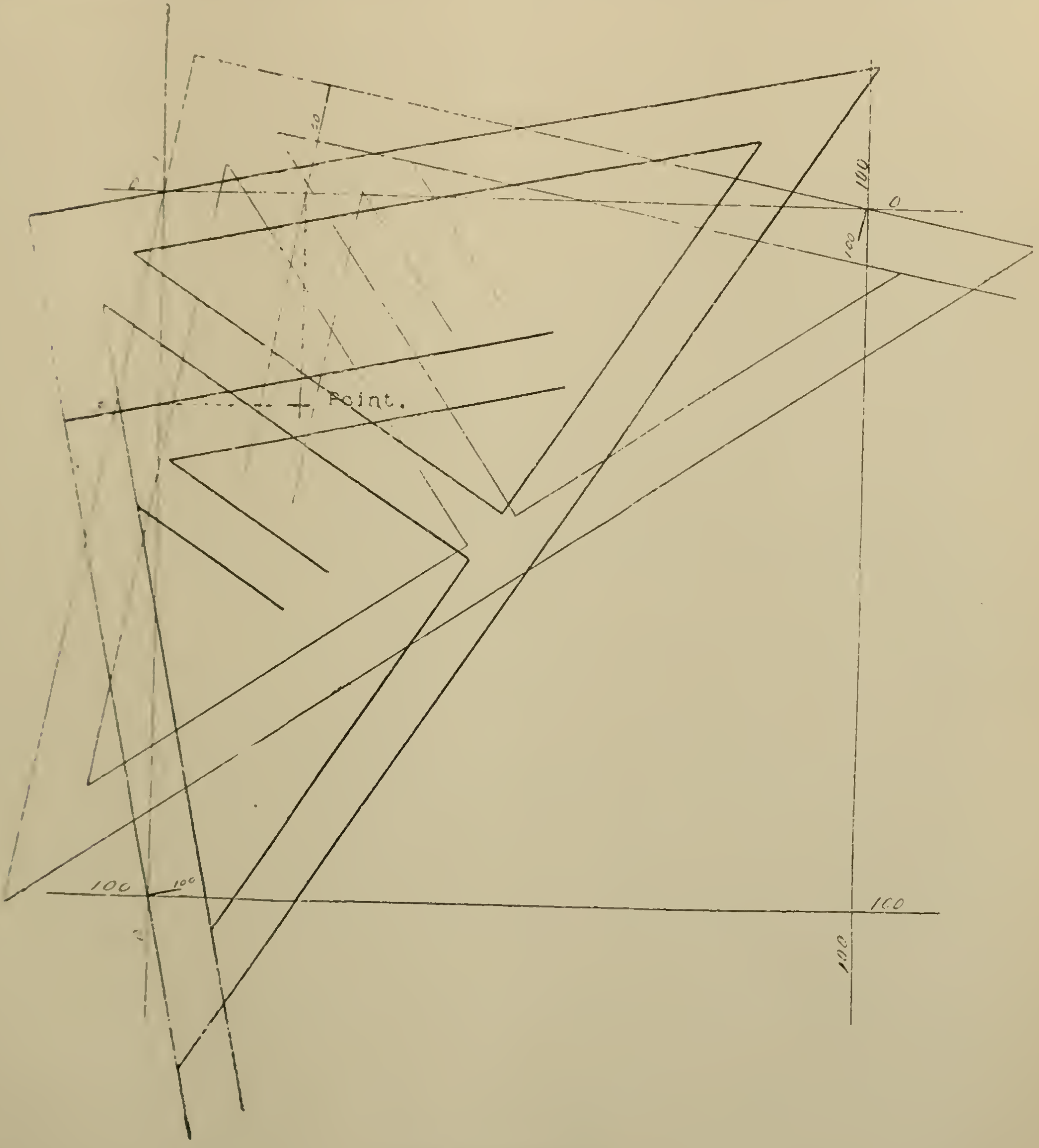
to expansion or contortion. In a contorted figure the co-ordinates have to be reduced according to the lengths of the opposite sides. See accompanying figures.

The graduated triangles are long enough to cover the co-ordinate squares as one can observe from figures following.

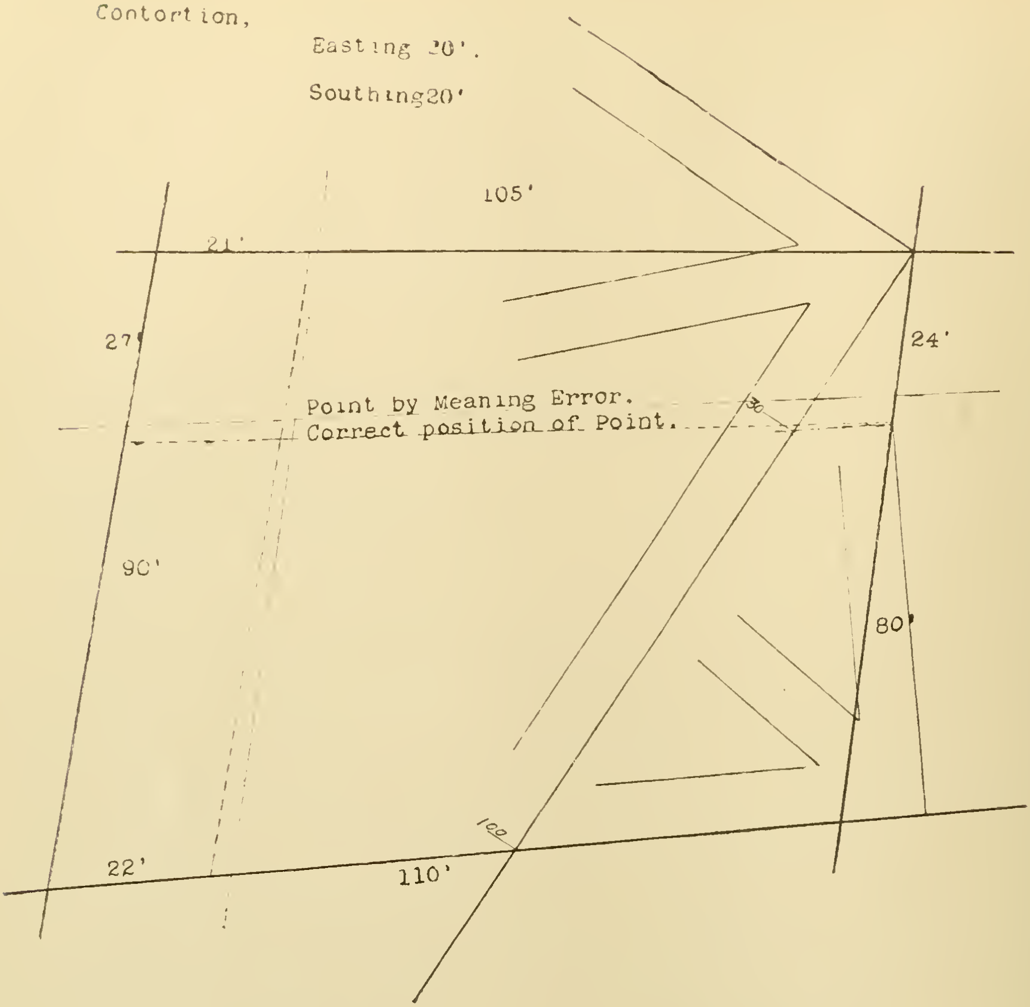
On the drive protractor how can one confuse a 1/500 scale with a 1/240. The difference is too apparent. Necessarily the protractor is numbered in the reverse way since it is revolved while the horizontal limb of the transit remains stationary in observing the angles Naturally one would use the backsight line in preference to some other line. One uses a theodolite showing objects upside down: the results are not invalidated thereby.

In criticising my method the author has applied his own system of logic. Applying it to his methods the following would be a *fac-simile.*

"One has to look at the backsight, and when looking at many points, using the same backsight-



Expansion. -Easting 20 ft.
Southing 30 ft.



Easting.	Southing.
105 × 20 ÷ 100 = 21 ft.	90 × 30 ÷ 100 = 27 ft.
110 × 20 ÷ 100 = 22 ft.	80 × 30 ÷ 100 + 26 ft.

The above figure is an exaggeration in order show to more clearly the principle involved in plotting co-ordinates.

for every observation (in the theodolite method only two observations of the backsight are essential) and at the same time maintaining the instrument precisely beneath the plumbbob while seeking the point at the face, the use of the instrument must be rather slow as one has constantly to shift the alidade, clamp, draw the line on the card, and record the tape measurements as well."

I certainly have to revolve the protractor for every angle read and at the same time shift the slide. In the offset method one has to shift the offset scale, and lay off the reference distances as well. One does not plot from the same station sufficiently often to make the glaring enlargement that the author assumes takes place. I use a fine needle and have yet to see the plan that I have mutilated by its use. All erasure is eliminated leaving the paper in excellent condition for inking in.

I presume that the stope protractor might get out of adjustment in the hands of inexperienced persons or if carelessly handled. A theodolite gets out of adjustment occasionally, so does a watch, etc. They are not condemned as being unfit for use because of this reason. After being adjusted by the maker there is little about the protractor to get out of adjustment as the wear is negligible. The protractor I have at present allows of reading the vertical and horizontal distances simultaneously and requires only two movements easily and speedily made. In using the slide rule, Mr. Tonnesen neglects to state that after obtaining the vertical distance, to obtain the horizontal distance the angle has to be subtracted from 90° and the slide again shifted. In a set of determinations either for horizontal or vertical distance, or of both together, I can state results in one-third the time required by the slide rule. The slide rule for angles of less than 15° is uncertain and necessitates too much estimating.

I am quite aware that Mr. Tonnesen's paper is based entirely on ideal portions of the Rand District. My paper is based on the Rand District also and includes conditions not so ideal as he has observed. He must have been aware of the inadequacy of the plane of the reef tape method in attempting to evolve a plane of the reef angular method. He certainly deserves all praise for his endeavours to supplant the tape method.

I take exception to his statement that I even go so far as to *theorize* on conical stopes. If he will take the trouble he may verify the existence of such stopes by visiting the Western Section of the Glen Deep Mine.

The horizontal plane theodolite method has been in use on the Simmer and Jack Mine for upwards of 15 years and there proved superior to tape methods, both as regards accuracy of determination of areas stoped, and in the preparation of the working plan. At the Robinson Deep it supplanted tape methods. Perhaps my methods will not come into general use: if they do not it will not be because of any inherent defects. On one mine the use of the plotting protractor was greatly appreciated by the surveyors with whom I was associated. When the time for plotting stopes arrived they showed a decided keenness in "gluing" themselves to the plotting protractor in order to plot their stopes with despatch, and to eliminate all the tediousness of the old methods of reduction and plotting.

In conclusion, I hope Mr. Tonnesen will not consider that my remarks show any personal animus. My sole object is both to give and to elicit information, and thus to put the matter on a sound footing.

London, October 11, 1911. J. J. BRISTOL.

Obituary.

The death is recorded with deep regret of Mr. G. O. Smart, a member of Council of this Society, which occurred on Wednesday, 29th November.

George Osler Smart was born in Forfarshire (Scotland) on 1st January, 1863. He came as a young man to Natal about 1881, and remained there for some years. About 1888 he came to the Rand, where after holding various positions he was eventually appointed battery manager at the George Goch Company. He occupied this position till 1898 when he became manager of the Simmer & Jack battery, at that time the largest on the Rand. At the time of his death he had for some years held with conspicuous success the position of reduction works manager of the Simmer & Jack Proprietary Mines, with control of the whole of the ore treatment operations of that Company.

Mr. Smart's reputation for skill and judgment as one of the very best millmen on the Rand has long been acknowledged by his fellows on these fields. Not a few men now occupying responsible positions owe much to their training under him. This was not merely in technical details, but likewise from the example of his methods in the organisation of a large department, and the discipline maintained therein, coupled with kindly consideration for all those subordinate to him, down to the youngest learner. The esteem and respect in which he was universally held was displayed at the funeral, by the very large attendance of his fellow workers of every rank, of representatives of all departments of the Consolidated Gold Fields, the parent corporation of the Simmer and Jack Proprietary Mines, Ltd., and of this Society.

While he was one of the pioneers of the Rand, his long experience of good practice had in no way prejudiced his mind against the possibility of even better methods, and the members of our Society owe him a debt of gratitude for his valuable contributions* to our proceedings especially upon the subjects of advances in amalgamation methods and classification of pulp and various minor matters; and it is fortunate that he has recorded the results of his matured experience of stamp milling in a form which will be made public at an early date.

He had long been a member of the Council of this Society, where his judgment, never expressed without due consideration, had corresponding weight, and he was also a member of the Institution of Mining and Metallurgy and the American Institute of Mining Engineers.

His death, at the relatively early age of forty-eight, from a long standing complaint endured with the utmost fortitude and courage, is a severe blow not merely to his company and many friends on the Rand, but also to the progress of gold ore treatment all over the world, and deep sympathy is felt for Mrs. Smart, and his son and daughter, who mourn his loss.

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* See this *Journal*, Vol. iv., p. 450; Vol. vii., pp. 1 , 293; Vol. ix., June, 1909, p. 25. Vol. viii., pp. 146, 355; Vol. x., Feb., 1910, p. 283

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

RECENT PROGRESS IN CALORIMETRY.—III.—“*Temperature Measurement and Corrections.*—Two previous papers of this series have dealt with the progress that has recently been made in eliminating various calorimetric errors due to uncertain or lagging temperatures in parts of the calorimeter, and, more especially, in its surroundings. To avoid these errors is not difficult; when this has been done the principal errors are in the temperature measurement, and are mainly accidental.

Of *mercury thermometers* for calorimetry the most sensitive are read to 0·001°, but many of the best authorities regard 0·003 to 0·002° as the maximum accuracy with which a temperature interval can in practice be reproduced by one of these thermometers. If somewhat different intervals are employed at different times unavoidable calibration errors will make the probable error appreciably greater. These sensitive thermometers seldom have a range of over 6°. Hence the maximum accuracy which could be expected with them is about 0·4 per 1,000. This, however, supposes that the full range of 6° is employed. Considerably shorter intervals are often recommended on the ground that Newton's law of cooling cannot be assumed to hold for intervals much over 2°. This recommendation is a mistake. The deviation from Newton's law for a 4° interval will not equal the thermometric error (unless the period is considerably over 10 minutes), while, on the other hand, a decrease in the interval will often increase the thermometric error almost proportionally. In fact, the effect of the deviation from Newton's law is usually quite negligible for an 8° interval, or can easily be made so by a rough correction.

The advantage of an increase above 6° is more doubtful with mercury thermometers, since several of the purely theomometric errors would tend to increase with the interval. This remark applies to the substitution of longer range thermometers for the most sensitive ones now in use. *With a given thermometer* it is probably always best, as far as the temperature measurement is concerned, to use the full range. And this has been done, up to 20°, without introducing any error chargeable to the calorimeter or the cooling.

For the best calorimetric work yet reported with mercury thermometers the mean accidental error (that is, the average variation of each determination from the mean result of the series to which it belongs) is about 0·5 per mille.

For electric calorimetric thermometers the sensitiveness and the accuracy can readily be made each 10 times the corresponding value for the mercury thermometer. That is, these electric thermometers can be read to 0·0001°, will reproduce ordinary calorimetric intervals with fair certainty to 0·0002°, and will in general yield measurements agreeing within from one ten-thousandth to one fifty-thousandth (0·1 to 0·02 per mille), their performance in this regard depending mainly on the accuracy of the auxiliary resistances. Two observers, working independently with different types of electric thermometer, have each succeeded in getting calorimetric results whose mean accidental error is one thirty-thousandth (0·03 per mille) or less. The *absolute* accuracy of these determinations may not be greater than 0·2 per mille, but this is sufficient for most present requirements and is as great as the accuracy yet attained in determining the variation in the specific heat of water itself. The electric installation is not only more accurate than a mercury thermometer, but is in some ways more convenient. The observer sitting, notebook in hand, before his galvanometer scale, within easy reach of switches which give complete control of the electrical system, can make readings with great rapidity and in great variety. His facilities in these directions will generally be of more use in the complex and varied problems of the research laboratory than in the simpler and more nearly standardized processes of commercial work. The disadvantages of the electric thermometer are three: The cost, the special electrical knowledge required, and the increased chance of occasional loss of time through slight accidents or blunders in the operation of the more complicated system. None of these disadvantages is really serious, unless the cost is. The strongest reason for not using electrical methods will probably be, in most cases, that greater accuracy than that given by the mercury thermometer is either not needed or is prevented by some cause quite outside the calorimeter, such as impurity or variability of materials.

Two forms of electric thermometer, the resistance thermometer and the thermo-element, have been used in calorimetry and have given results of (so far) about equal and very high accuracy. For some types of recording and regulating apparatus the resistance thermometer is much more sensitive than the thermo-element, but in existing calorimetric thermometers there is little difference in this direction.

The resistance thermometer has at present the advantage that a form of it, developed and now used at the Bureau of Standards, can be purchased ready made, with suitable accessory apparatus. A form accurate at least to 0·0003 in calorimetric work costs, complete with accessories, about $225. For the thermo-element also highly satisfactory accessory apparatus (potentiometers, etc.) can now be purchased, and thermo-elements suitable for calorimetry require for their construction neither skill, experience, nor great labour, but they are not regularly manufactured at present. Most standardizing laboratories are better equipped for the calibration of resistance thermometers than of thermoelements, though the calibration of thermoelements is very easy wherever suitable standards are available. The chief advantage of the thermoelement appears to be the greater ease with which the thermoelectric installation lends itself to the simultaneous measurement of several temperatures, and of currents and voltages as well, if desired, but this advantage, again, is greatest in the case of the research laboratory.

The Calculation of Corrections for Cooling seems to deserve attention here because (1) it is mainly through the abbreviation of these calculations that shortening and simplification of the labour of calorimetric determinations can be secured, and (2) the customary formulæ for these calculations have been misunderstood to a surprising extent, considering their age and simplicity. It will probably be sufficient, however, to give merely results and conclusions. A derivation of the two formulæ here given will be found in the articles on calorimetry already referred to, those in the November and December *Physical Review* of last year. One of the formulæ, also, the Pfaundler, has been often presented, though with a bewildering variety of notations, which I have made some effort to simplify.

Three kinds of data may in different cases be needed to get cooling corrections: (1) Data as to the temperature changes produced in the calorimeter by stirring and evaporation: (2) the temperatures of the calorimeter during three periods, namely the *transfer* period, during which there is communicated to the calorimeter the heat which is to be measured, and two *cooling* periods, before and after the transfer period; (3) the rate of [temperature change of] the calorimeter during the cooling periods.

It is generally known, but sometimes overlooked, that these data may properly be much less accurate than the measurements which determine the main temperature rise, since the cooling correction is only about one-fiftieth of the whole interval. The observer who corrects his transfer period temperatures to thousandths of a degree *is simply wasting time* But the rates, though needing no greater accuracy in themselves, are each obtained from the difference of two temperatures, so that these temperatures need to be known with considerable accuracy. There may be, therefore, an appreciable error connected with the cooling correction, though this error is not in the cooling itself, but is an additional *accidental thermometric* error, coming in the determination of the cooling rate. Methods of diminishing or eliminating it can more conveniently be presented a little later.

Cooling correction calculations are easier to make if the rate is proportional to the temperature difference of calorimeter and jacket—that is, if Newton's law of cooling holds. The same advantages can be obtained where that law does not hold, however, by introducing the variation from it as a correction. Hence, in what follows, Newton's law will at first be supposed to apply, and the correction for deviations from it will then be discussed.

The simplest case of a complete cooling correction formula occurs where the heat of stirring and evaporation and the jacket temperature are known, for the amounts of direct cooling are then simply proportional to known temperature differences and only one cooling period is needed. For instance: Let ϕ_{tr} be the integrated temperature difference (of calorimeter and jacket) for the *transfer* period (i.e., the period when the heat to be measured is transferred to the calorimeter water), and ϕ_c the same for the subsequent *cooling* period. (The notation here is the same as in the *Physical Review* articles already referred to, except that the subscripts indicating the periods are numbers in those articles.) Let T_{tr} and T_c be the number of minutes in each period, w the heat lost by evaporation and stirring per minute (w, of course, will often be negative), $\partial\theta_c$ the observed temperature fall in the cooling period. Then $\partial\theta_c - wT_c$ is the direct heat loss in period (c), and that in period (tr) equals:

$$(\partial\theta_c - wT_c)\frac{\phi_{tr}}{\phi_c}$$

the cooling correction is therefore, simply

$$wT_{tr} + (\partial\theta_c - wT_c)\frac{\phi_{tr}}{\phi_c} \qquad (1)$$

This can easily be manipulated into the form:

$$\partial\theta_c + (\partial\theta_c - wT_c)\frac{\phi_{tr} - \phi_c}{\phi_c} + w(T_{tr} - T_c) \qquad (2)$$

which is very convenient practically whenever the calorimeter is initially near the temperature of the jacket, for $\phi_{tr} - \phi_c$ will then be relatively small, often very small while $T_{tr} - T_c$ will often be zero. Hence w, which is always multiplied by one or the other of these, need not be known at all accurately, and the two multiplications will give small products and so can be done by a slide rule, and usually by inspection.

Equation (1) shows how to avoid the thermometric error, mentioned above, which occurs in the determination of the cooling rate. That error comes in $\partial\theta_c$, which is multiplied by $\frac{\phi_{tr}}{\phi_c}$ hence the smaller ϕ_{tr} is, or the larger ϕ_c, the less will be the final error produced. ϕ_c can be increased by lengthening the cooling period, but this method is time-consuming and may tend to introduce other errors. ϕ_{tr}, however, can be decreased by taking the initial temperature below that of the jacket, so that the calorimeter receives heat from the jacket during part of the transfer period and gives it back again during the other part; and the total net cooling (proportional to ϕ_{tr}) is nearly zero. This is not Rumford's old rule of taking the jacket temperature *midway* between the initial and final calorimeter temperatures, but usually calls for a jacket temperature somewhat nearer the final. If this method is not used, and ϕ_{tr} nearly equals ϕ_c [as in formula (2)] the determination of the rate about doubles the maximum possible accidental thermometric error, but in accordance with the laws of probability, only increases the probable mean error of a series by 42%; hence this method of reducing the cooling correction cuts in two the maximum error, and diminishes the mean error about a third. One price which must be paid for this advantage is an exact knowledge of the value of W, the heat of stirring and evaporation, for errors in this value are no longer diminished as by formula (2).

The method just described, namely, that of diminishing ϕ_{tr}, can be used to save time as well as to gain accuracy. For since it enables us to diminish almost indefinitely the effect of errors in the value of the cooling rate, and, therefore, to get good results with rates inaccurately known, we may determine the relation of rate to temperature (the cooling factor) once for all, and then, assuming that this relation is constant, we may use it in each subsequent determination. We thus get rid of the cooling period altogether, yet with no loss of accuracy. The formula for the correction with this procedure is:

$$wT_{tr} + K\phi_{tr}$$

where K is the constant,

$$\frac{\partial\theta_c - wT_c}{\phi_c} \qquad (3)$$

Some observers have found that the cooling factors of their calorimeters were constant to ½% or better, which suffices for an accuracy of 0·1 per mille even where the calorimeter is started at jacket temperature. The causes which may prevent this constancy do not seem to be fully known as yet, but it seems probable that the measurement of a cooling rate for each separate calorimetric determination will soon be generally regarded as not only a waste of time, but a detriment to accuracy.

Of the labour of getting the cooling correction there now remains only the determination of ϕ_{tr}—that is, taking of temperature observations during the transfer period and the addition of the results. Even this labour is sometimes avoided, and the value of ϕ_{tr}, the total of the transfer period temperatures, is obtained from the initial and final values of ϕ by means of a table of some sort. The saving in this case is relatively not great, but is often easily obtained. As soon as anyone has performed a few determinations he has the material for expressing ϕ_{tr} in terms of the other temperatures. Of course, this method succeeds better the more uniform the various conditions—such as time, room temperature, material.

In many cases the jacket temperature and the heats of stirring and evaporation are not known. But *if they are constant* they can be eliminated by the use of another cooling period. The most convenient way possible of accomplishing this seems to be by the *Pfaundler formula.* The essential principle of that method is this: The proportionality between heat flow and the temperature difference (ϕ) of calorimeter and jacket cannot be used because (ϕ) is not known. But one consequence of that proportionality can be made useful, and that is: That the *difference between any two* cooling rates is proportional to the difference of the corresponding temperatures. Hence, not knowing the jacket temperature, we may refer our formula to any other temperature for which we know the rate and proceed much as before.

The additional cooling period furnishes the reference temperature in question. To illustrate, let this period, which comes before the transfer period, and is therefore a preliminary period, be denoted by the subscript, p. Since ϕ is not known, the *mean* temperature of the periods will be used instead: call θ_p, θ_{tr}, θ_e. It will also be necessary to use, instead of the temperature intervals, $\delta\theta$, the rates, V_p, V_e, which equal $\frac{\delta\theta_p}{T_p}$, etc. Then the mean rate for the transfer period will be the mean rate for the preliminary period, plus the difference in rate due to the difference in temperature. The cooling correction, which equals this rate, multiplied by T_{tr}, the time of the transfer period is, therefore:

$$T_{tr}\left\{V_p + \frac{V_p - V_e}{\theta_e - \theta_p}(\theta_{tr} - \theta_p)\right\} \qquad (4)$$

which is essentially the Pfaundler formula.

Here two things are noticeable. (1) After the first term the formula is exactly like formula (1) in type, but instead of the temperatures of the two last periods there appear the differences between these temperatures and those of the preliminary (reference) period. (2) The heats of stirring and evaporation have vanished, since, being the same for all the periods, they are eliminated in the subtractions. This has always been counted one great advantage of the Pfaundler formula, and is, indeed, evident enough, but still seems to deserve emphasis, for some recent publications indicate that it has sometimes been overlooked.

As ordinarily printed the Pfaundler formula shows a summation, which is the same as the term $\theta_{tr}T_{tr}$ in the form here given. If the periods are all taken equal the computation is easier, since $\delta\theta$ can then be used instead of V. The computation is then as in formula (1), except that the three indicated subtractions must first be performed.

This formula, being like (1), can, like it, be treated as (1) was in getting (2), and thus becomes:

$$T_{tr}\left\{V_e + \frac{V_e - V_p}{\theta_e - \theta_p}(\theta_{tr} - \theta_e)\right\} \qquad (5)$$

which is very convenient when θ_{tr} is nearly equal to θ_e, since then the computed term is so small that it can be obtained by approximate methods. θ_{tr} nearly equals θ_e, when the calorimeter is started at about the jacket temperature, which is usually the most convenient way to do. I believe this variation of the formula is new.

The method described above of diminishing the accidental thermometric error by diminishing the final magnitude of the correction is practically useless with the Pfaundler formula. The proof is simple, and seems unnecessary here. There is, therefore, contrary to a frequently expressed opinion, little or nothing to be gained in accuracy by any particular choice of initial temperature if this formula is used, and the most convenient procedure may be employed, whatever that is. It will often be with the calorimeter temperature near the jacket temperature at the beginning, for which formula (5) is convenient. It also follows that the Pfaundler formula cannot give the highest accuracy, but its compensating advantages are often great. Where the environing temperature cannot be measured, because the environment consists of several bodies at different temperatures, the Pfaundler formula, or something equivalent, is indispensable, and this is probably the case with most existing calorimeters.

On the whole, then, the advantageous treatment of the cooling correction depends upon the observer's knowledge of conditions; the more he knows, the shorter and more accurate his work will be. If jacket temperature and heat of stirring and evaporation are all unknown the Pfaundler formula is essential, with three periods to observe. If these conditions are known the periods may be reduced to two, and accuracy may also be increased by suitable choice of initial temperature. If the cooling factor of the calorimeter is also known the periods may be reduced to one, with no loss of accuracy. If, finally, the form of the heating curve (*i.e.*, during the transfer period) is known, the observations in that one period may be omitted, but, probably, with some loss of accuracy in many cases.

For *adiabatic methods*, also, essentially the same relation holds between the observer's knowledge of conditions and the ease and accuracy attainable. In these methods the jacket temperature, of course, must always be measured, but if the heats of stirring and evaporation are not exactly known two periods are necessary, with the same thermometric error as when formula (2) is used. The periods can be reduced to one, with thermometric error as small as with formula (3), and this requires (as for formula (3)) an exact knowledge of the heats of evaporation and stirring. The cooling factor need not be known for the adiabatic method, but this is not a matter of much importance, since a very approximate knowledge is enough in any case. The computation of the correction is saved in the adiabatic method, but an equivalent labour is substituted for it. In the 'diabatic' method the observer reads certain temperatures, which he afterwards adds; in the adiabatic he duplicates these temperatures in the jacket, and must then refill the jacket and adjust it for the next determination.

Graphic methods are sometimes used for computing the cooling correction. Instead of *adding* the individual temperatures, the observer *plots* them, and then measures the plot with a planimeter. There is no question, however, that, with observations equally spaced in time, the easiest and most accurate way, by far, of getting their sum is to add them. The graphic methods are only valuable and, as a rule, have only been used in cases where the observations are taken at irregular time intervals. With such observations all the periods are of irregular lengths, and the labour of computation is considerably increased. The graphic method is merely a means of diminishing the inconvenience inherent in observations of this sort. These irregularly timed observations result from a method of reading which is often adopted with electric thermometers, but which is not in the least necessary; and is even less convenient with properly arranged apparatus. In general, therefore, they can be avoided, and with them the graphic methods which they make necessary.

The treatment of the *corrections for large intervals* remains to be considered. All the above formulæ treat the cooling rate as proportional to the temperature difference of calorimeter and jacket, and owe their convenience to that proportionality. It is desirable to use the same formulæ even where that proportionality no longer exists. The method of doing this can, perhaps, be best illustrated by the actual description of a simple calculation.

For this method it is necessary to know the difference of calorimeter and jacket, and this involves some measurement of the jacket temperature. But variations in the jacket temperature do not need to be known as accurately as for small intervals. And in the computation only the temperature difference is used; the absolute temperature of the system is of no importance practically.

Suppose, now, in a systematic study of the cooling of the calorimeter, the following results have been reached:

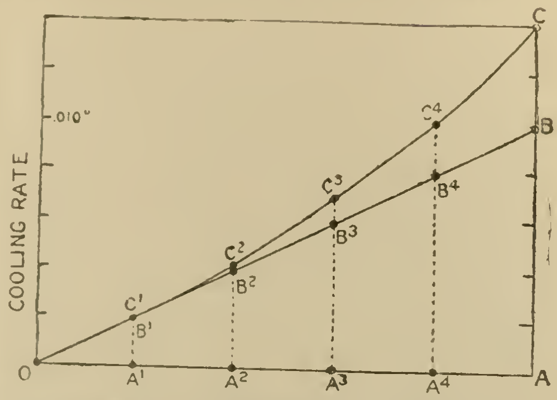
When the temperature difference was:	The cooling per minute was:
1	0·002
2	0·004
3	0·007
4	0·010
5	0·014

These results are plotted as in Fig. 1, where OA_1, OA_2, etc., represent the observed temperature differences, in degrees, and A_1C_1, A_2C_2, etc., represent the corresponding cooling rates of the calorimeter in thousandths of a degree.

The straight line OB is now drawn through C_1. The points B_3, B_4, B_5 on it correspond to 3°, 4°, 5°. Since B is straight, and runs through the origin, the vertical distances A_3B_3, A_4B_4, etc., are proportional to the horizontal distance OA_3, OA_4, etc.

Suppose, now, that a calorimetric determination is made, and that during it observations of temperature difference are made as follows:

1°, 3°, 4°, 5°, 5°, 5°, etc. These numbers have been obtained as a means of afterward finding the cooling that has taken place. That is their only function. It can be performed because to each number there corresponds a particular cooling rate. But a series of numbers *proportional* to the rates, if we had them, would enable us to obtain the rates more easily. Hence, if for the observed differences 1°, 3°, etc., we can substitute another set of numbers which are proportional to the desired rates, the purpose of the original observations will be accomplished and at the same time computation will be facilitated. Such numbers can be obtained from Fig. 1. Thus: the vertical distance A_1C_1 represents the rate for 1°. The ratio between rate and number is 0·002. The rate represented by A_3C_3 does not bear the same ratio to 3°. But if the difference, 3°, is increased in the ratio to $\frac{A_3C_3}{A_3B_3}$, becoming $\frac{7}{6} \times 3°$, or 3·5°, it will bear the desired ratio to the rate, for the rate is 0·007, which is 0·002 of 3·5. Similarly 4°, multiplied by the ratio $\frac{A_4C_4}{A_4B_4}$, or $\frac{5}{4}$, becomes 5° which is proportional to the rate, 0·0100, which prevailed when the difference was 1°. Similarly, 5° becomes 7°, of which the 5° rate, 0·014, is 0·002. And so on.



Temperature Difference in Degrees.
Cooling Rate as Function of Temperature Difference.

Now the condition for the cooling correction formulæ given above was that rates should be proportional to temperature differences. The new temperature differences which we have just formed artificially fulfil this condition, and so they enable us to use *any* of those formulæ with no further trouble.

In practice, the matter is much simpler than in the case just described, which was intended to illustrate the principle. For (1), the curve of Fig 1, is not used directly, but the values B_3C_3, B_4C_4, etc., are plotted in another curve on such a scale that they give at once the amounts which must be added to the temperature differences in order to obtain the proportional numbers. Thus, in the present illustration, against the abscissas 1°, 2°, 3°, 4°, 5° would be plotted the numbers 0, 0, 0·5°, 1°, 2°, and a smooth curve drawn through these points. Again, (2), the correction will almost never be needed for calorimetric intervals under 4°, and only occasionally even where the interval reaches 12°. For instance, in one case, where the cooling factor of the calorimeter increased 18% as the temperature difference increased from 1° to 12°, it was nevertheless found that for determinations involving a 12° interval the corrections for variations of the cooling factor was only $\frac{1}{20000}$ of the calorimetric interval, which, of course, is usually quite negligible. In many cases the total correction would not be negligible as in this case, but the variations between different values of the correction would be so, hence, after a few determinations had been carried through, a single constant value for the correction could be adopted, and no more computation would be needed.

The above method does not require that the cooling factor of the calorimeter remain constant from day to day. It merely supposes that the rates for different temperatures remain in nearly the same relation to each other, which will always be the case. The correction curve may be in error by 16%, and usually much more, without causing perceptible error, as long as it is a smooth curve. But the different numbers obtained from the curve must not show *accidental irregularities* greater than the allowable error of the temperature difference observations, whatever this may be. This condition is easily secured by drawing the curve to a sufficiently large scale. (A discussion of the possible error in this method of correction will be found in the *Physical Review* articles already several times referred to.)

The present paper discusses (1) calorimetric thermometers. Electric thermometers, both resistance and thermoelectric, are at least ten times as accurate as mercurial, and are in some ways more convenient, though more expensive and complicated. (2) Cooling correction formulæ are discussed, new forms being given which both cost less labour than the older forms and allow higher accuracy. Adiabatic methods are subject to the same principal limitations as those where corrections are used. In all, the accuracy and the ease increase with the observer's quantitative knowledge of such conditions as jacket temperature, heat of stirring, etc. (3) A method of applying ordinary formulæ to large intervals is described in detail."—W. P. WHITE, *Metallurgical and Chemical Engineering*, Sept. 1911, p. 449. (H. A. W.)

THE ESTIMATION OF BISMUTH.—"Treat 0·5 gm. ore with 10 cc. concentrated HNO_3 and evaporate almost to dryness. Add 10 cc. concentrated HCl and heat gently until decomposition is complete. Dilute with 10 cc. hot water and filter. Wash residue with warm dilute HCl (1 to 3). Nearly neutralise filtrate with 10% ammonia, dilute to about 300 cc. and pass H_2S. Bismuth, lead and other heavy metals will be precipitated. Filter and wash with hot water. Decompose precipitate with 10 to 15 cc. dilute HNO_3 (1 to 1), dilute and filter; wash with dilute HNO_3 thoroughly keeping bulk of filtrate as small as possible. Dilute with hot water to about 250 cc. Heat nearly to boiling, neutralise cautiously with 10% ammonia until faint opalescence is observed. Dilute to 350 cc. with hot water, add 1 cc. dilute HCl (1 to 3) and 2 to 3 cc. saturated solution NH_4Cl. Keep it nearly at boiling point until the basic chloride of bismuth has settled clear or nearly so, filter and wash with hot water. As the precipitate may be contaminated with a little lead, it is dissolved in hot dilute HNO_3 and the precipitation effected as before. Finally collect bismuth basic chloride on tared filter, dry at 110° C. and weigh. If arsenic or antimony be present the sulphuretted hydrogen precipitate should after washing with hot water be washed two or three times with a warm solution of ammonium sulphide and finally with hot water. Silver can be removed by adding a drop of strong HCl to the original diluted HNO_3 solution before filtering."—FRERICHS and LOW.—*The Journal of Industrial and Engineering Chemistry*, Vol. iii., No. 10, Oct. 11, 1911, p. 775. (J. G.)

RAPID AND ACCURATE METHOD FOR THE ANALYSIS OF WHITE METAL.—"0·5 gm. of fine drillings is dissolved in a mixture of 25 cc. of HCl and 5 cc. of HNO_3 (or 25 cc. of HCl saturated with Br), the solution is heated on the water bath and 1 gm. of tartaric acid added. After diluting with hot water to 400 cc., excess of NaOH is added followed by excess (about 20 cc.) of a 10% solution of Na_2S, made by adding a 10% solution of NaOH to an equal volume of the same solution which has been saturated with H_2S. The precipitate is washed twice by decantation and several times on the filter with a hot dilute solution of Na_2S. The residue consists of the sulphides of copper, lead, iron and zinc: the lead is determined as sulphate, and the other metals as usual. The filtrate, containing Sn and Sb, is treated with a slight excess of HCl, and warmed on the water bath for one hour. The precipitate is filtered off, washed, and the paper and precipitate treated in an Erlenmeyer flask with a measured excess of N/10 I_2 solution, 30-50 cc. of concentrated HCl and 2 gm. of tartaric acid. The flask is fitted with a rubber stopper carrying a small swan neck thistle-tube to condense any iodine vapours, and the contents are heated nearly to boiling for a few minutes, whereby the tin is converted into stannic iodide and the antimony into the tri-iodide. After cooling, the excess of iodine is titrated with thiosulphate: the iodine consumed corresponds to the tin plus antimony. The solution is now neutralised with Na_2CO_3, excess of powdered $NaHCO_3$ added, and the antimony determined by Mohr's method (titration with N/10 iodine solution). If the volume of iodine solution used be multiplied by $\frac{3}{2}$ and the product deducted from the volume of iodine solution consumed in the previous operation, the volume of iodine solution corresponding with the tin is obtained."—J. C. BENEKER, *J. Ind. Eng. Chem.*, 1911, 3, 637-8.—*Journal of the Society of Chemical Industry*, Vol. xxx., No. 19, p. 1165. (J. G.)

COMPARISON OF METHODS FOR THE DETERMINATION OF NICKEL IN STEEL.—"*Dimethylglyoxime Method.**—Weigh out 1 gm. of the nickel steel, place in a 250 cc. beaker, cover, add 20 cc. HCl (sp. gr. 1·12) heat gently till steel is dissolved. Oxidise with 6 to 7 cc. HNO_3 (sp. gr. 1·20). Continue heating a few minutes longer to complete the oxidisation, add 20 gm. of tartaric acid dissolved in 20 cc. of water, remove from fire and cautiously add a strong solution of KOH till solution is nearly neutral, a change of colour indicating this condition. Volume of solution should now be about 200 cc. If solution is made alkaline, slightly acidify with acetic acid, heat to boiling and add 15 cc. to 20 cc. of a 1% alcoholic solution of dimethylglyoxime with stirring. Ammonia is then cautiously added until the solution smells slightly of it; while still hot filter on a weighed Gooch crucible, wash thoroughly with warm water, dry in oven at 110-120° C. for 45 minutes, cool in desiccator, and weigh.

Nickel dimethyl glyoxime is a red crystalline salt, contains no water of crystallisation, sublimes at 250° C., without decomposition, and contains 20·32% nickel.

Determinations on substances containing known quantities of nickel showed this to be a very accurate method. It is easy to operate, and can be readily completed in 1½ hours.

Cyanide Method. Weigh out 1 gr. of the nickel steel in a 150 cc. beaker, dissolve in 20 cc. of (1 : 1) HCl. When action ceases, add 10 cc. of HNO_3 (sp. gr. 1·20); reduce to volume of 15 cc. by boiling, keeping beaker covered, remove from fire, add 32 cc. of 25% H_2SO_4 in water. Transfer to a 400 cc. beaker, add 4 gm. of citric acid, stir well, and then slowly add ammonia till the precipitate just dissolves and the solution acquires a *faint* odour of ammonia. If too much ammonia is added nearly neutralise by the careful addition of HNO_3; dilute to about 150 cc., cool to below 20° C., add 2 cc of a 10% solution of KI, and enough N/10 $AgNO_3$ solution, the volume of which must be noted, to produce a distinct turbidity. Slowly run in the standardised KCN, stirring thoroughly till the turbidity just disappears, and the solution assumes a light golden yellow colour. The solution should remain bright for five minutes, otherwise the titration is incomplete.

Standard Solution. Prepare an exact N/10 $AgNO_3$ solution, 1 cc. of which equals 0·002935 gm. of nickel. Standardise the KCN against this solution by taking

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* See also this *Journal*, vol. x., Feb., 1910, p. 293.

20 cc. of cyanide in a 250 cc. beaker, diluting to 150 cc. with cold water, adding 2 cc. of a 10% KI solution, running in $AgNO_3$ until a distinct turbidity is produced, and then slowly adding the cyanide till the turbidity disappears. From the amount of $AgNO_3$ used the strength of the cyanide in terms of nickel is readily found. The KCN should be made up approximately N/10 and 5 gm. KOH added per litre in order to improve its stability, frequent standardisations being necessary. Check analyses by this method can be completed in 50 minutes.

Resumé.—The dimethylglyoxime is more accurate than the cyanide method, and should be employed in all umpire and check work where exact determinations are required.

The cyanide method is a good, quick, accurate method, particularly adopted to routine laboratory work. It is easy to conduct, though the operator should have some experience with the end point, the greatest source of error in the method."—JAMES J. BOYLE, *Chemical Engineer*, xiv., No. 1.—*Chemical News*, Vol. 104, No. 2706, pp. 168-9. (J. G.)

METHOD OF NICKEL-ZINC SEPARATION IN GERMAN SILVER AND OTHER ALLOYS.—"Weigh out 0·5 gm. of the drillings, or more, according to the percentage of the constituents: take out the tin with HNO_3, the lead as sulphate, copper by electrolysis, and iron with ammonia as usual. Add 5 gm. of ammonium chloride. Make the solution just neutral with HCl, and add $\frac{4}{10}$ gm. of dimethylglyoxime dissolved in a little alcohol for every tenth of a gram of nickel supposed to be present. Add NH_4HO, drop by drop, until the solution smells slightly ammoniacal. Let stand just below the boiling point for one half-hour (more or less according as the nickel separates completely). Filter hot through counter-poised filters or on a weighed Gooch, wash with hot water, and dry in the air bath at 105° to constant weight. Weigh and calculate the nickel.

$$NiC_8H_{14}N_4O_4 \times 0{\cdot}2031 = Ni.$$

Until a little experience has been acquired, the filtrate should be tested to make sure that no nickel has escaped precipitation. Add 0·5 gm. more of the dimethylglyoxime in a little alcohol and let stand for a few minutes. There should be no reddening of the solution, which would mean more nickel. Make the filtrate just acid with concentrated HCl and add an excess of 10 cc.'s. Boil for 10 minutes to break up the dimethylglyoxime in the solution, add 10 gm. of microcosmic salt dissolved in a little water, neutralise with NH_4HO and HCl or acetic acid until the solution neither turns blue litmus paper red nor red paper blue, and let stand just below the boiling point (do not let bump) until the precipitate has become granular. Filter under suction, wash with hot water, ignite in a weighed porcelain crucible, cool, and weigh as zinc pyrophosphate

$$Zn_2P_2O_7 \times 0{\cdot}42913 = Zn."$$

—LA VERNE W. SPRING, *Journal of Industrial and Engineering Chemistry*, iii., No. 4.—*Chemical News*, Vol. 104, No. 2697, p. 58. (J. G.)

NOTE ON THE DETERMINATION OF NICKEL.—"Brunck's method for the determination of nickel by precipitation with dimethylglyoxime from an ammoniacal solution has deservedly come into general use in metallurgical analysis. As this nickel compound sublimes at 250° C., most operators, after washing with hot water, dry at or about 100° C., and weigh. This procedure possesses the advantage that the weight of the precipitate is nearly five times that of its contained nickel, but it is open to the objection that towards the end of the washing the precipitate exhibits a decided tendency to pass through the filter, and it is doubtful, moreover, whether it is absolutely insoluble in hot water.

Bogoluboff has proved that the precipitate may be safely ignited to nickel oxide without loss, if the filter is first carefully charred without causing it to take fire, the temperature being then raised to full redness. The author has repeatedly confirmed this observation, and uses this method of dealing with the precipitate, which makes it permissible to add ammonium nitrate to the wash water and obviate the difficulty mentioned above.

Spring's method for the analysis of German silver has been in use in these laboratories (Sheffield University) for a considerable time with excellent results. It suffices sometimes, however, to determine the nickel contents only of this alloy, and this can be done without previously separating the copper by the following method.

Dissolve half a gram of the drillings in 10 cc. HNO_3 (1·20) and dilute to about 400 cc. with water. Add 2 or 3 dgm. of tartaric acid to prevent the precipitation of the small quantity of iron almost invariably present, and then add ammonia in excess. Heat to about 50° C., add an excess of dimethylglyoxime reagent, and after allowing to stand in a warm place for a few minutes, with occasional stirring, collect the voluminous precipitate on a filter paper, or preferably on paper pulp. Wash with hot water containing ammonium nitrate till the washings are perfectly colourless. If the precipitate is then ignited as described below, the nickel oxide will contain small amounts of copper oxide, making the calculated nickel percentage from 1% to 1·5% too great. Dissolve from the pulp, therefore, in a warm mixture of equal volumes of 1·20 HNO_3 and water, wash well, and repeat the precipitation, filtration, etc. Enclose the precipitate completely whilst still wet in two filter papers, transfer to a crucible, and heat cautiously at the mouth of the muffle or over a small flame until the outer papers are thoroughly charred. Increase the temperature gradually and finish the ignition at a bright red heat."—F. IBBOTSON.—*Chemical News*, Vol. 104, No. 2711, p. 224. (J. G.)

METALLURGY.

BALL MILL PRACTICE AT KALGOORLIE.—"So much has been written about the stamp mill that other efficient pulverizers are apt to be overlooked; so I make no apologies in presenting these notes on the modern ball mill, in this case the Krupp, in the hope that they may be of use to others who have a plant in operation, or contemplate erecting one. It is doubtful whether the ball mill is operated with greater efficiency and less cost anywhere in the world than at Kalgoorlie, that is, crushing dry. We have had great experience with them, and with a constant interchange of ideas, each plant is well abreast of its neighbour, until there is a practical uniformity of practice.

The ores of the Golden Mile, the richest portion of the goldfield, do not vary much in composition, and taken on average, it may be said that they are hard to crush. The ore averages 60% silica. In the Great Boulder the proportion is as high as 75%. Below are short descriptions of the ores treated in the dry crushing plants:

Associated ..	Partly schist, rather above average in silica, hard, and highly mineralized.

Associated Northern...	Main lode of schistose character, west lode hard and silicious.
Chaffers ...	... Varies from schistose to hard flinty quartz, with 3 to 5% pyrite.
Great Boulder...	... Very hard, cherty quartz with a little schist.
Kalgurli ...	... Quartz diabase varying in hardness according to depth, some schistose, and other highly silicious. Average tough.
Perseverance ...	... Schistose in structure, and contains pyrite and magnetite throughout.
South Kalgurli	... Average 60% silica, with little schist, and is considered hard.

The following are analyses from the Associated Northern and Perseverance mines:

Insoluble	60·60	63·46
Aluminium oxide ...	5·75	2·06
Iron bisulphide ...	5·32	6·80
Lime carbonate ...	11·19	13·42
Magnesium carbonate ...	5·29	6·93
Iron carbonate... ...	10·23	6·80
Alkali undetermined ..	1·62	0·53
	100·00	100·00

The average ore is dry enough for milling, although the driest holds from 0·5 to 1% moisture. Anything above this amount retards crushing. As in a stamp battery, or other type of crushing machine, a great deal of the efficiency depends on having massive foundations. So it is with the ball mill. A mill on a solid block of concrete will crush better than one on a timber or steel girder, no matter how well braced. A springy support is bad for the gear, and does not allow the mill to be stiff, as it were. This has been noticed especially in the Associated plant built 11 years ago, in which 10 mills were resting on timber girders, and two on concrete piers, the latter doing far better work than any other pair in the plant. The former foundation consisted of a masonry bed, on which stood 12 × 6 in. steel girders with brackets riveted to support two longitudinal pieces of jarrah (native hardwood) 12 × 14 in., across which were bolted three pieces of 12 × 14 in. for each mill. This would appear to be stiff enough, but the vibration was bad all the same. Now the big mills are on fine concrete foundations, reinforced with old wire rope and rails. The Associated Northern, Chaffers, Great Boulder, and Perseverance ball mills are erected on massive concrete foundations, especially the last mentioned; those at the Kalgurli sit on steel girders 8 × 6 in. well braced and apparently firm; while in the South Kalgurli plant they are bolted to a frame of 12 × 12 in. timbers, well braced and standing on a concrete foundation. This looks a small structure, but it is free from vibration.

As soon as the foundations are sufficiently set, and the four bearings for the mill shaft and pulley shaft are lined up, the construction can proceed rapidly. With 8 men who understand the work, a mill may be erected in 24 hours. Everything is simple. The main walls are bolted or riveted to the cast iron naves, which, when in their proper places, are keyed to the mill shaft. The side liners are bolted loosely until all the mantle plates with the grinding plates are in position, then everything may be tightened; all bolts having check nuts, except those on the mantles holding on the grinding plates, which are riveted. The fitting of the scoops, baffles, inside and outside screens, is quite a simple matter. The feed hopper is in two pieces for bolting around the shaft, and is lined for the wear of ore dropping from the feeder. Unless motor driven, each mill has a fast and loose pulley on the pinion shaft. There are two sizes of mills at work in Kalgoorlie, with the following dimensions:

	No. 5 Inches.	No. 8 Inches.
Diameter	89	106
Width..	46	54
Mantles for plates	10	12
Inside screens	5	6
Outside screens..	10	12
Area of outside screens, each ...	46 × 26	54 × 24

The last item is the full area, without deducting that part of the screen covered by the wooden frame. Approximately, the net screening surface on each mill is 70 and 100 sq. ft. respectively.

The mantles carry four or five grinding, and one perforated, plates, which are tightly bolted together. The bolts have a feather on the tapered portion to fit into the plates, thus preventing turning during tightening. Bolted to the bottom end of the mantles are the baffle and scoop plates, which return the oversize to the mill. The whole mill is enclosed in a tight fitting casing, made in convenient sections for removal during repairs. The spur wheel, keyed to the mill shaft, is usually of cast iron, and should last, if well greased, fully four years; while the pinion lasts about half that time. The loose pulley runs on a sleeve which must be well greased to prevent seizing. On one of the new mills at the Associated, a new idea for loose pulleys has been devised, in the shape of a cast iron bracket with an extended hollow sleeve, bolted to the plummer block furthest from the mill. The pinion shaft revolves in this sleeve, and the loose pulley rests on the latter. When the mill is working the pulley is at rest, thus preventing much wear at this point.

It is found that a good stiff light grease is best for all bearings on a ball mill. A few turns of a grease cup twice in 8 hours is generally sufficient. For the gear, old, but clean, grease from other bearings is good enough. Properly fitted gear, when well greased, makes little noise.

When driven off a main driving shaft, tight and loose pulleys are required; but when motor driven, the former only is necessary. The Great Boulder people are fitting clutch pulleys on the main shaft with satisfactory results. An 8 in. 6-ply balata or rubber belt it strong enough for a No. 5 mill, and a 10 in. 8-ply for a No. 8.

From 24 to 26 rpm. is the correct speed for the No. 5, and from 21 to 24 rpm. for the No. 8. It is generally admitted that the latter speed for the larger mill is quite safe and shows much greater efficiency, although the makers stipulate the lower speed. Mills are loaded with 1 to 2 tons of 5 in. steel balls according to size and hardness of the ore to be crushed; there is no advantage in feeding several sizes of balls. Indicator diagrams at the Associated Northern and Associated have shown that the No. 5 mill takes from 18 to 23 h.-p. when in an average working condition, and the No. 8 consumes 50 to 60 h.-p. Where motor driven, as at the Perseverance and South Kalgurli, the ammeters register 60 to 65 amperes at 550 volts.

It is found advisable to have the storage bin for coarse ore a few feet behind the ball mills, as it enables a proper chute with a gate and a long feeder to be attached, thus giving a steady flow of ore, which cannot be assured when the bin is right over the mills, as originally erected at the Associated.

For the protection of employees, and keeping the plant free of dust, and the better working of a ball mill, a strong fan attached to the mills by large piping is necessary; and a plant properly arranged in this detail is a pleasure to operate. A couple of Sturtevant fans, about 4 ft. diam., running at about 800 to 1,000 rpm. should keep eight No. 8 mills clear; and a 3 ft. fan, as we have it at the Associated Northern, running at 800 rpm. is effective for three No. 5 mills. A 4 ft. motor driven fan of the Associated takes 5 amperes. Suction and delivery pipes should be fitted with sliding doors to facilitate cleaning. The draught in the suction pipe at the Associated Northern averages 0·33 in. The fan generally blows the dust into a canvas lined house, in which it readily collects, or, as at the last named mill, it is blown into the second hearth of the Merton furnaces. At the Perseverance the dust is collected in cyclone arresters, and sent into the main furnace flue. A strong draught aids in keeping screens clear, especially when the ore is at all damp, the steamy air being withdrawn. When dry crushing with stamps at Waihi in New Zealand, in 1894 to 1899, the beneficial effect of a fan was very noticeable. The amount of dust lost in dry crushing is difficult to estimate; but from observations I should say it would represent 1% at least.

The ball mill feeders are usually of the grasshopper or shaking tray type, that is, a sloping tray with sides 6 in. high at an angle of about 25°, suspended at the discharge end by a chain or rod from a beam above, and the top end supported by two iron legs, fitted loosely to the underside of the tray, and to a timber support near the floor. At the back of the feeder tray is a tappet, and on a shaft is a single, double, or three-armed cam, adjustable to give as much as a 1 in. lift to the feeder. This pushes the feeder back, which, when released, drops forward by its own weight, discharging an even flow of ore into the mill. It is the custom now for each mill to have its own feed shaft driven by a 3 in. belt from a pulley fitted to an extension of the mill axle. The mills at the Perseverance are so fitted, the whole being neat and effective. At the South Kalgurli the feeders are arranged with a perforated tray, through which the fine ore passes to a shaking tray fitted with 27 × 27 in. screen, this in turn screening all that portion already fine enough, it passing to the conveyor beneath. We tried one for some time at the Associated Northern but did not think it increased the capacity of the mill. The arrangement requires close watching. Naves are arranged for left and right hand feeding; by this means mills may be run in pairs, the hoppers being reasonably close together; and it may be remarked here that mills should not be too close together, as plenty of space is required for repairs. The balls are fed either through the manhole door or hopper, and ore is added until the balls are well mixed with it. A mill should not be started without ore, as the balls knock the plates about badly. Unless a mill is motor driven it would be difficult to arrange for an automatic feeder. I have not heard of any, but think that one could be devised, depending on the increase of power consumed above a set point. If a mill takes 60 amperes with a full load, and is overfed to 62 amperes, then some automatic gear could shut off the feed. Like many other machines a ball mill is worked by sound. This is an important point to impress on mill men, as a poor attendant may easily cut the efficiency of a mill by 25% in not keeping his ears open. An operator becomes keen to the exact rumble of a ball mill, and can detect a light feed from a distance. A mill running empty simply roars; when light there is a clear metallic sound of balls striking one another; when just right, there is simply a rumbling noise; while in a mill that is full, the balls and ore are carried high enough to fall directly on to the mill axle or shaft, this being bad work, as the shaft is liable to be damaged and worn, and, of course, crushing is reduced to a minimum. The remedy for too much feed, of course, is shutting it off until the correct sound is noted. A millman not too sure about his feeding can easily see this by stopping the mill, and looking into it through the feed nave with a light. The mass of balls and ore should be evenly distributed, and no bare balls should be showing.

The operation of the Krupp ball mill may be described as follows: The mill is a gear driven steel lined drum, charged with 1 to 2 tons of steel balls, revolving at 26 to 24 rpm. according to size. The ore crushed by the balls passes through the perforated grinding plates, and is screened by the inside or coarse screen, the oversize being returned automatically to the interior of the mill; while what passes the inside screen is sized by the outside or fine wire screen, the oversize being also returned to the mill for further grinding. The action of crushing and returning the oversize is ingenious, and it will be seen that as soon as the ore is crushed fine enough it must pass out of the mill. Herein lies the efficiency of the ball mill. In a 5 stamp battery the ore is pulverized excessively by not being discharged directly it is fine enough. In the ball mill there is little of this, and any machine that discharges its product immediately it is of the required size will show high efficiency. As regards the action of the balls, it has been stated that crushing is done by impact; but from careful observation I am perfectly satisfied that this is not so, the only impact that might take place is when the balls drop from one grinding plate to another, a distance of five inches in a newly lined mill; and then, only some fine particles would be crushed by impact at this point. The action of the balls is one of pure rolling, grinding, and abrasion, they being lifted about level with the mill shaft or axle, and then roll back, tumbling over one another, rubbing and grinding the ore. Of course, there is a great deal of attrition between the pieces of ore apart from the work of the balls. This is not mere supposition, as I have actually seen the action. At the Associated mill, we had four No 5 mills crushing our oxidised ore that carries some 5% moisture, so we used coarse screens, in fact the wire screens were removed, and the inside ones used only, these being punched steel with 30 holes per sq. in. In crushing this ore no dust is made, so by the aid of a portable electric lamp, placed well down in the feed hopper, the balls and ore could be seen plainly. It was observed that the balls and ore were carried up nearly level with the shaft, and having reached this point, started to roll back. The 1,200 lb. weight of steel was rolling and tumbling about, grinding and rubbing the ore between the balls. It was a most interesting sight, fully proving that there is no impact grinding. To confirm this, one has only to take note of the ore in a mill when it is stopped for repairs. Down to the small particles, say 6 mesh, it is smooth and round just as if water worn from a river. Pieces of steel, which come from the mine at times, are rounded in the mill, although they hinder good work. If impact were the action in a ball mill, dynamite, which also finds its way with the ore, would explode as it does in a Griffin mill; but it does not, and I take it that it is just rubbed away. F. B. Allen, Director of Technical Education in this State, has kindly examined some ball mill product, which was wet screened to clean the particles from dust, under the microscope, to see whether it had a sharp fracture or not, with the result that they show the +30 screen product to be somewhat rounded, and the +120 to have sharp, ragged edges. I saw the particles under the microscope, and was quite satisfied with the evidence.

This shows that the final particles are split by the rolling of heavy balls, not necessarily by the drop of a ball as in a tube mill, described by H. Fischer in several of the leading technical journals in 1904.

In a table following Table I. will be found the consumption of steel and life of liners. Steel consumption is gauged by the weight of balls consumed per ton crushed, and not by the total balls and liners used, as in a battery where shoes and dies are calculated. The average steel consumption in the ball mill plants here is 0·55 lb. per ton, while the hunch or grinding plates last about 190 days, the perforated as long, and side liners about twice as long. Manganese steel of the Hadfield or Krupp make is most satisfactory. As regards capacity we may say that the efficiency depends on the quantity crushed per horse-power consumed. A No. 5 Krupp mill crushes ordinary sulphide ore up to 43 tons daily, and the No. 8 up to 100 tons, and dividing this by the horse-power consumed (18 to 23, and 60 to 65) we get 2·1 and 1·6 tons per h.-p. respectively. In a small plant of 100 tons daily capacity, the No. 5 is a useful unit, but in a large plant the No. 8 is the best size. This has a large capacity, runs sweetly, and occupies little more space than the smaller unit. The quantity of ore crushed as given above is from actual weighings in some cases, and from the tonnage crushed divided by the hours run in others. On quartz the capacity of a No. 5 mill is lower, being about 36 tons per diem. We had excellent opportunities of finding out the capacity at the Associated Northern, as all the custom ore is weighed.

TABLE I.—Details of Ball-Mill Practice at Kalgoorlie (dry-crushing).

Mine.	Number of Mills.	Weight of Balls. lb.	Screens.	Speed r. p.m	Power h.p.	Drive.	Capacity Tons daily	Steel Consumption per ton.	Life of Grinding Plates. Days.
Associated ...	4 No. 8	4,480	30 × 30	21	60	Belt from shaft ...	92-95	0·50	170
	6 ,, 5	2,240	25 × 25	25	23		43		
Associated Northern	3 ,, 5	2,350	27 × 27	26	18	,, ,,	40	0·32	270
Chaffers	3 ,, 5	2,300	27 × 27	25	25	,, ,,	40	0·74	180
Great Boulder ...	4 ,, 8	4,480	30 × 30	24	60 25	Belt from shaft and friction clutch ...	80-90	0·64	105
Kalgurli	9 ,, 5	2,200	26 × 26	25	60	Belt from shaft ...	40	0·45	300
Perseverance ...	8 ,, 8	4,400	27 × 27	24	65	Motor and belt ...	100	0·65	118
South Kalgurli ...	3 ,, 8	4,480	27 × 27	24	30	,, ,,	95-100	0·47	210
	1 ,, 5	2,800	—	—	—				

On an average, 63,000 tons monthly are dry-crushed in ball-mills on the Kalgoorlie goldfields.

It is impossible to prevent damp ore, from 1 to 3% moisture, coming into a plant. Then the efficiency decreases greatly and the wear and tear becomes heavier, as the feed must be light to prevent screens clogging, a steaming being set up, that makes everything sticky. When crushing dry ore, the inside of the mill becomes hot from friction, a recent test showing up to 130°F. when the outside temperature was 97°. The Kalgurli used to dry its damp ore before milling, but rotary dryers are a nuisance. The soft oxidised ore can best be crushed, as at the Associated and Associated Northern, by only using about 1,200 lb. of balls, taking off the outside screens, and passing it through the inside or coarse screens. Pans reduce this product quite easily. Actual tests show that a No. 5 mill will crush up to 82 tons daily, but for the 5% moisture the capacity would be greater. The Chaffers has one mill on this class of ore, wet crushing 93 to 100 tons through 26 × 26 screen per diem, using 2,400 lb. in balls.

It is advisable to open mills for inspection once a week, unless something goes wrong in the meantime, and every two months the balls should be taken out and weighed, all balls under 1½ in. being discarded, as they are of little use in crushing, and the original weight made with new balls of 18 lb. each. Foreign matter, such as drill-ends, bolts, nuts, spanners, accumulate in mills, and as they hinder the smooth work of the balls they must be thrown out.

One man at 11s 8d. per shift of 8 hours can look after eight No. 8 mills; but in smaller plants the millman attends to conveyors, elevators, dust-pipes, etc. Ball-mills should be fed with no larger ore than that which will pass a 3 in. ring. With fine ore the balls are apt to bed, as it were. Actual weighings have shown that a 3 in. feed is crushed faster, and the wear of steel less, than when the feed is about 1 in. with a lot of fine.

Grading analyses from ball-mills at different plants are given in Table II.

With the exception of the Associated and Great Boulder, whose screens are 30 × 30, the others are about 27 × 27. They are fairly uniform, although the −150 is low at the Associated, Associated Northern, and Kalgurlie for some reason not understood. The ball-mill does not make so much −150 as the Griffin and other pulverizers.

The coarse screens used to be fixed to the mill by studs, but a local improvement was introduced in fitting them on with 1½ in. angle-iron, bent to circle and bolted. A large number of bolts are used in a mill, and cost a good deal. For some reason, perhaps expansion of plates due to heat, or loose plates, these break and fall into the mill, and then a lot of coarse ore gets through the bolt-hole to the space between the mantle and screens, causing the mill to run out of balance. Experienced millmen soon detect this. On account of this trouble, there is a

TABLE II.—Grading Analyses.

Name.	+ 30	+ 40	+ 60	+ 80	+ 100	+ 120	+ 150	− 150
Associated	—	16·16	15·66	10·46	6·43	3·76	3·46	44·00
Associated Northern	—	11·20	20·10	8·30	11·80	3·20	2·00	43·60
Chaffers	—	10·25	13·19	8·60	7·25	2·70	2·70	56 30
Great Boulder	1·00	5·80	14 20	8·70	9·00	2·30	2·30	52·00
Kalgurli	—	8·20	18·00	13·40	9·10	2·40	2·40	45·30
Perseverance	0·20	3·80	20·00	7·80	5·40	4·00	4·00	54·20
South Kalgurli...	—	10·20	21·20	6·00	2·60	3·10	3·10	55·90

difference of opinion as to the use of the baffle plates fixed to the return scoop-plates in the mill.

The fine screens should be brushed once daily, if the ore is dry, and oftener if damp, by a wire brush. They last from 30 to 40 days, providing no stones work out and tear them. When the grinding-plates get thin and nearly worn to the mantle-plates, the mill requires re-lining. This is done differently in several plants; some millmen prefer to have a spare set of mantles and fix on the plates at leisure, and when the time comes for re-lining, just to take out those worn out, and fit in the prepared plates, this taking about 10 hours with 6 men; while others consider it best to leave the mantles in place, and cut off the old grinding-plates, and fit new ones as they proceed. This can be done in about the same time, but is awkward, heavy, and hot work inside the mill. There are several little points in practice on which we shall never be agreed, but their effect is trifling.

A No. 5 ball-mill, including foundations, may be erected for £600, and a No. 8 for £1,000, bin and conveyors not included. The former will cost about £300 yearly for maintenance.

As regards wet crushing, I have seen No. 4 mills at work in Broken Hill plants crushing jig-middling that passed a 8 mm. screen; a No. 5 on the Lake View Consols crushing sulphide ore: and the No. 5 at the Chaffers working on the soft, schistose, oxidized ore.

In the first instance the mills could not show great efficiency on such feed; in the second, the mill never had real trial, as the grinding-pans and tube-mills were already overtaxed; while at the last-named the ore is so soft that, in any comparison with stamps on hard ore, it could not be used. Some years ago the Eclipse mine here had ten 900 lb. stamps crushing similar ore to the Chaffers at the rate of 10·5 tons per stamp per diem through a 20 mesh screen; so the capacity of the Chaffers ball-mill of 93 tons to 100 tons through a 26 mesh is somewhat better than the 10 stamps. The wet mill product is:

	Per cent.		Per cent.
+ 40 ...	10·60	+ 120 ...	3·56
+ 80 ...	16·12	+ 150 ...	7·59
+ 100 ...	4·44	− 150 ...	57·69

The ore is crushed with weak cyanide solution in the proportion of 1·6 to 1. The wear and tear in the wet mill is almost twice as heavy as in the dry, the grinding-plates lasting 105 days.

Max Drott made some comparative tests on hard quartzite ores between a 5-stamp mill and a ball-mill for the Krupp firm, and the latter crusher showed an output of 40 % above the battery, even the dry mill being better on the same ore. I am convinced that on a thorough trial, all things being equal, the ball-mill will show a greater output and efficiency than the stamp-battery. Will somebody take up this question, and show that the modern heavy gravitation stamp is not the most economical crusher for its present work and that there are other machines that can do the work better?

Costs of ball-milling are given in Table III. The methods of segregation of costs are not uniform, as will be noted at once. On some mines spares, stores, and repairs are lumped together. The cost of power is high at the Perseverance and South Kalgurli on account of motor-driving. The Associated is in a state of transition from the old No. 5 mills to the No. 8 size. The Great Boulder has lately installed a high-class steam-engine, and it is expected to reduce the cost of power to the above figure."—M. W. VON BERNEWITZ.—*The Mining Magazine*, August, 1911, p. 139. (A. R.)

LOSSES IN CYANIDING.—"*Introductory.*—The interest manifested in the filtration of slime, especially with reference to the prevention of mysterious losses of gold and silver during cyaniding, has of late been much intensified by sundry unfortunate events. The fact that such losses do occur is beyond doubt, but the cause as well as the remedy is far from apparent. It may easily be proved that the presence of slime accounts for much of the loss, which in a measure is preventible by the aid of filters. The following few notes touching on the more important uses and features of the vacuum type of filter, may be of service in opening a discussion with a view to elucidating matters.

Preparation of Pulp for the Filter.—There are various sources of loss in the treatment of the pulp before beginning to separate the solution from the

TABLE III.—Cost in Pence per Ton.

Mine.	Wages.	Power.	Spares.	Stores.	Repairs.	Sundry.	Total.
Associated	6·45	13·78	6·73	2·67	0·49	—	30·11
Associated Northern ...	5·87	12·08	4·73	1·42	0·10	—	24·22
Chaffers	2·09	10·64	—	0·25	6·88	1·71	21·57
Great Boulder	1·81	9·75	—	6·26	—	—	17·82
Kalgurli...	—	15·23	—	7·49	—	—	22·72
Perseverance	3·21	19·18	3·32	0·46	0·11	0·54	26·84
South Kalgurli	2·05	20·82	5·54	—	3·68	0·21	32·30

solids, but it is not necessary to deal with these other than incidentally, because the filter is capable of augmenting the dissolving treatment and practically eliminating these sources of loss. Suffice it to say that the slime is mixed with solution in the proporton of 1 ton of solid to 2 tons of liquid (the range in practice is from 1½ to 3) and the mixture is agitated, the object being to facilitate the dissolving of the gold and silver by constantly changing or circulating the solution about the metal-bearing particles, and at the same time to aerate the pulp. In cases where the slime settles readily when agitation is stopped the pulp is thickened, before it is passed on to the filter, by decanting the supernatant solution. A pulp having a consistence of about 1 : 1 is generally desired in order to avoid any necessity for agitation while filtering, but in some cases it is not practicable to thicken to this extent and the pulp goes to the filter containing as much as 3 of solution to 1 of solid. When filters are provided with means for effecting a constant circulation of the pulp during filtration the question of thickness resolves itself into one of economy. When the pulp contains less than 1 of liquid to 1 of solid it is not sufficiently mobile for economical handling; but, of course, there are exceptions to this rule. The coarser the slime particles the less solution need be present.

The thickness of a pulp is generally expressed by the ratio of solid to liquid, which from a given ore is conveniently determined from the specific gravity of the pulp.

Cake Formation and Thickness.—When filtering slime containing much amorphous clay it is good practice to apply a low suction (about one quarter of the maximum) until the cake is about ⅛ in. thick and then the higher vacuum. By this means a more permeable cake is formed than if the higher vacuum is applied at the start. It is consequently more quickly washed, that is, exhausted of its valuable contents. When the slime consists chiefly of crystalline material (tube-mill product) the effect of the low suction at the start is not so apparent, but it usually results in a clearer filtrate. Operators as a rule, however, pay little attention to this variation in the vacuum, but keep it constant during cake-formation.

The thickness to which a cake is allowed to build-up is governed chiefly by economic considerations. At the beginning of each cycle of operations, when the filter-cloths are clean, the solution filters through readily, and with a constant vacuum the rate becomes slower and slower as the deposit increases. A point is reached when it becomes economical to stop and proceed to wash and discharge the cake and start the cycle again. Exactly when this occurs must be found by trial. It naturally varies with the character of the slime together with the altitude of the locality. In some cases the limit is reached when the cake is only ½ in. thick, as when dealing with highly amorphous or gelatinous material, while with crystalline silicious slime a cake of 2 in. and even more may be economically formed before beginning to wash.

The usual way to find when the cake is of the desired thickness is by direct measurement or by time comparisons, but it is obvious that the rate at which the filtrate is withdrawn may be utilised for this purpose with advantage.

When using a constant vacuum the rate at which the cake forms is shown, graphically, by the parabolic curves in the annexed diagram, Fig. 1. It will be noticed how the rate rapidly decreases as the cake thickens. Thus in the first ten minutes the cake forms six times as rapidly as in the last ten minutes. For example, if a 1¾ in. cake took 100 minutes to form, over 9/16 in. would have been deposited within the first ten minutes, while in the last ten minutes it would build up only 1/12 in. Again, if the cake at the end of 100 minutes was only ½ in. thick, about ⅜ in. would be formed in the first 50 minutes, and it would take another 50 minutes to add ⅛ in.

The porosity of a cake is affected by the pressure of formation. Thus a cake deposited by vacuum contains 40 to 65 lb. dry ore per cubic foot, according to the specific gravity and fineness of the material, while the same material filter-pressed in a cake contains 70 lb. to 110 lb dry ore per cubic foot.

The foregoing considerations supply food for thought to the practical investigator. They give an insight into the reason for the well known fact that high-pressure filters, when dealing with amorphous slime, produce thin sloppy cakes, from which the metallic contents cannot be economically and efficiently washed, whereas the vacuum-filter on the same material forms thick, porous, and well dried cakes in excellent condition for washing.

If the filter is to be used simply for dewatering or drying, the curves show graphically how much more economical it is to clean-off at short intervals rather than at long intervals when washing is employed. This is particularly evident with the so-called continuous filters, which, of course, do no effective washing, but simply act as dewaterers.

The permeability of the cake adjusts itself inversely as the pressure of the atmosphere plus the hydrostatic head, and although the cake may be thicker in one part than another it will allow only an equal flow of solution to pass per unit-area of the cake. It is, however, important that the cakes should be fairly uniform in thickness and should not build up so as to allow two opposite faces to touch or 'freeze' together. An uneven thickness is due chiefly to settling and is most apparent when the pulp is of thin consistence. It may be completely avoided by agitating or circulating the pulp while the formation of cake is proceeding, but if the pulp is sufficiently dense, say, 1 of solid to 1 of liquid, little or no trouble is likely to arise from uneven thickness.

Characteristics of Filter-Cakes.—It is generally accepted that cakes of slime formed by pressure-filters differ in character from those formed by vacuum-filters, although, as is obvious, the formation of both is due to pressure, the difference being only one of degree. It may be that this difference in character is also only one of degree. In practice it has often been shown that when filtering so-called natural slime the vacuum-filter produces not only a thicker cake but one that is more porous and more uniform in its porosity, so that it can be more efficiently washed. On the other hand, when dealing with fine sand or crystalline slime the high-pressure filter produces the heavier cake in a given time, but not heavier in proportion to the power expended and not so uniform in its porosity.

The cake formed on the vacuum-filter consists of an infinite number of layers roughly parallel to each other and the washing is done at right angles to the layers, that is, across the grain of the cake. On the other hand, when forming cakes in the chambers of the filter-press, a segregation of the ore particles invariably occurs, producing a distorted grain and irregular porosity. There is no way of remedying such a defect, because the cake is held between two filter-cloths so as to necessitate the use of a clear wash-liquor that continues to pass unevenly through the cake. With the vacuum-filter the cake has a-

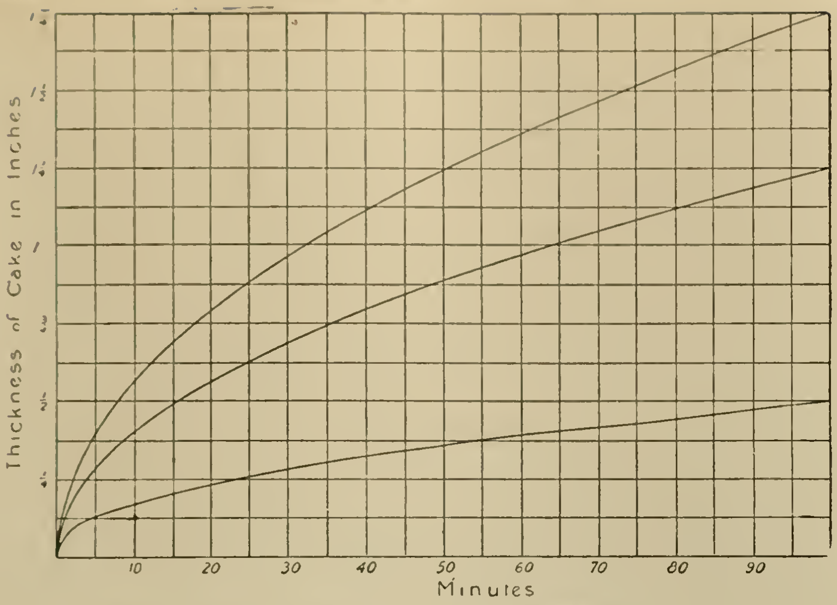

Fig 1. CURVES SHOWING THE RATE OF FORMATION OF CAKES

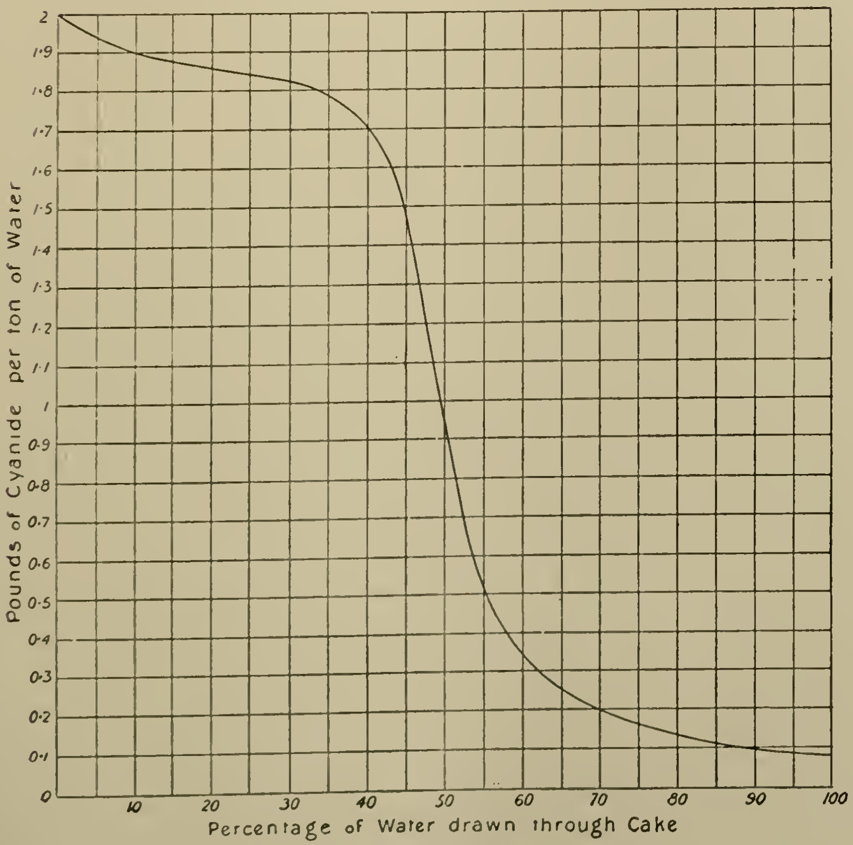

Fig 2. WATER-WASH DILUTION.

face in contact with the wash-liquor, which is purposely kept turbid. If there are any parts of uneven flow in the cake they soon choke causing the porosity to become adjusted until more or less uniform washing ensues. However, when dealing with some classes of ore excellent washing has been done with filter-presses, but my point is that the vacuum-filter has a wider range of application and invariably dilutes the filtrate less for equally satisfactory washing.

Care of the Cake.—During the time that the cake is exposed to the air it should be treated with care. If any appreciable sucking action occurs, air is drawn through the cake, tending to disturb its grain and causing compacting in parts and increased porosity elsewhere. In order to wash such a cake much liquor is required, which results in a filtrate poor in metal. This, however, may be partly obviated by using a highly turbid wash-liquor.

If suction is cut off the grain of the cake will remain unaltered during the time required for transfer. With cakes held vertically, or overhanging, the stoppage of the sucking action is not always practicable, because the small amount of moisture always left behind the filter-cloths descends by gravity, and should this ooze through to the cake adhesion would fail, causing portions of the cake to fall away. One maker of vacuum-filters successfully overcomes this difficulty by rendering the lower margin of the filter-cloth impervious, thus forming a trough for retaining the drained solution.

It is also important that the cake while exposed to the air should be subjected to as little vibration as possible. If this is violent some cake may be knocked off, but even a slight vibration, although producing no perceptible effect to the eye, may seriously disturb the grain of the cake and spoil its uniform porosity. The risk of injury from this cause is reduced to a minimum by using a non-moving type of filter.

After the cake has been formed and exposed to the air 30 to 35% of its weight consists of valuable liquor, that is, metallic solution formed in the treatment-vats, together with undissolved metallic gold and silver particles or compounds. The former can easily be removed fairly well by means of a water-wash, but if the latter, which is usually by far the most valuable of the two, is to be extracted, a solution-wash is applied before the water-wash.

Washing the Cake.—This operation requires more attention than it usually receives. After sliming, cyaniding, and forming the cake, it is rarely economical to hasten the process at the expense of washing. Too often it happens that filters are so over-loaded with work as to make the solution-washing little better than a farce. There is no excuse for this when we consider that the value of the metal removed by the wash far exceeds the cost of the operation.

The main object of the solution-washing is to extract on the filters the metal that has failed to dissolve in the treatment-vats, or was re-precipitated during treatment, and incidentally to wash out all dissolved metal held by the cakes. Various independent investigations have shown that the gold and silver that remain undissolved in the treatment-vat after agitation are partly dissolved while solution-washing the cake on the filter. The undissolved metal is in two forms: that which is intimately associated or combined in its natural state with the gangue, and that which after dissolving has again returned to a solid state. I refer to this as being re-precipitated. Both forms are recoverable to an appreciable extent, at a trifling cost, after the cake has been formed.

Without going deeply into the chemistry of the subject, which is somewhat obscure, it is enough to say that the gold and silver dissolve on the filter after escaping solution in the vat, but how much of this is metal in its natural state and how much is due to re-precipitation is not discernible. Periodical tests show that the rate of dissolving in the treatment-vat always becomes slower and slower as the operation proceeds, and in time may become so slow that the cost of agitating exceeds the value of the metal dissolving. Theoretically the rate of increase should be only in proportion to the diminution of the surface of metal in contact with the solution, but in practice this is far exceeded; in fact, tests show that at times dissolving appears to cease altogether in spite of metallic particles and free cyanide being present. The phenomenon is explained by the solution becoming degraded in presence of finely divided solids. The term 'degraded' applies to a diminution of the solvent power of the solution due to the introduction, from the ore, of soluble impurities having a reducing action. If it were made possible to bring fresh potent solution continuously in contact with the ore particles, instead of simply circulating the degraded solution, the rate of dissolving would be kept more constant and the tendency to re-precipitate would be minimised.

The vacuum-filter provides means for performing this operation almost ideally. The cake is submerged in a potent wash-solution which is slowly drawn through the pores and in its course dissolves a portion of the gold and silver together with degrading matter, as in the treatment-vat, but as the solution becomes degraded it is drawn forward out of the region of the gold and silver particles and is replaced with fresh potent solution, the action being continuous. In this way the rate of dissolving of the gold and silver, per unit area exposed to the solution, remains constant. The rate lessens proportionately only as the area of the exposed metal diminishes.

The foregoing applies with even greater force when a compound, such as silver sulphide, is also present. Silver sulphide, although only sparingly soluble, readily degrades a solution, whereby it stops or hinders the dissolving of any metallic gold and silver, as well as further silver sulphide that may be present in the pulp. Various devices are employed for reviving this lost dissolving power in order that the solution may be enriched before filtering, and even when they are effective the dissolved metal is often in such an unstable condition that some of it re-precipitates either in the vat or in the cake. The solution-wash, if properly applied, practically removes the whole of this re-precipitated metal, together with native metal that had hitherto escaped solution.

Re-precipitation and Re-solution.—It has often been observed that a minimum extraction in the agitator-vat is followed by re-precipitation of the gold and silver. What is not so easily observed and not so well-known is that precipitation and dissolution proceed in the vat at the same time, and for this reason among others the gold and silver in the pulp are never completely brought into solution. It is also not realised that this re-precipitated metal does not re-dissolve in the same solution although the native metal may continue dissolving slowly in its normal manner. Without going into the subject deeper, we may say that re-precipitation is due to local action of both a physical and chemical nature. Graphite, tellurides, sulphides, arsenides, charcoal, grease, oils, and even silica or alumina in an extremely fine state of division, precipitate both gold and silver from working cyanide solutions, and this precipitate shows little or no tendency to re-dissolve in the same solution. If this re-precipitated metal be filtered off, washed, and re-pulped with a fresh solution, a portion re-dissolves and if the operation be repeated a further portion re-dissolves. In time nearly the whole of the metal may in this way be re-dissolved. Such a process could not, of course, be employed economically, but the vacuum-filter makes it possible to apply similar fresh solutions continuously and without the necessity of re-pulping. Thus when a fresh potent solution is drawn steadily and evenly through the cake the metallic particles, being continuously surrounded by the advancing liquor, re-dissolve and pass away with the filtrate. This re-dissolving is not instantaneous but requires time for contact: hence a rapid flow through the cake does not hasten matters, but results only in a large volume of filtrate poor in metal. A cake ¼ in. thick requires about as much time to be solution-washed as one 2 in. thick.

Silver is much more prone to re-precipitation than gold, and is afterward more slowly re-dissolved on the filter. Even the addition of water to the pulp is sufficient, in some instances, to cause an appreciable proportion of the silver to assume a solid state.

It often happens that scum or froth accumulates on the surface of the pulp in the agitator-vats. This sometimes becomes extremely rich in gold and silver, a large proportion of which may be directly traced to re-precipitation. The enrichment of the scum is evidence of re-precipitation occurring in the mass of agitated slime. It is difficult to bring the solid particles of the scum in contact with a solution in the treatment-vats, but if it be transferred periodically to the filter, along with some slime, it may be solidified with a little trouble into a cake, and freed of metal with a suitable wash-solution.

Preparation and Application of the Wash-Solution. —This is usually made from the barren solution that leaves the zinc-box. Before being used it should be thoroughly aerated to ensure complete oxidation of any reducing matter present. If the wash-solution is not already muddy it is advisable to add a little of the finest slime, the object being to provide means for readjustment of any slightly uneven permeability in the cake, such as might accidently have occurred during the time that the cake is exposed to the air. The solution just before use should be made stronger in cyanide than that originally used in the treatment-vats. How much stronger is found by trial, and if silver sulphide be present the strength of the wash requires special attention. It is generally inadvisable to add free alkali to wash-solutions.

The object of applying this wash is to impoverish the cake with a minimum quantity of solution in a minimum time and obtain a filtrate as rich as possible in gold and silver. As already stated there is nothing to be gained by rushing the solution through the cake at a high velocity. It is only necessary to give it such a rate of flow as would prevent backward diffusion, and if this were maintained the cake would be washed with a minimum quantity of solution and produce a filtrate of maximum richness. In practice, however, this slow rate of flow cannot be employed because it is necessary to accomplish the washing in a limited time, even though the filtrate should suffer in richness. The time allowed by operators to solution-wash the cakes is as a rule absurdly short, with the consequence that great loss in the aggregate occurs. In every case when applying a solution-wash the vacuum should be kept as low as possible for the time allowed.

After completing the solution-washing the cake contains cyanide and should carry only a minute quantity of dissolved gold and silver, chiefly obtained from the barren solution. It is generally economical to save the cyanide by applying a water-wash.

Water-wash.—One-third of a ton of wash-solution for every ton of ore is retained by the cake. If therefore this solution contains 3 lb. of cyanide per ton the solution in the cake will contain ½ lb. per ton of solid. The object of the water-wash is to remove this ½ lb. of cyanide, or a large proportion of it for further use, without unduly diluting the filtrate, and incidentally any dissolved metal present. Theoretically this would appear to be a comparatively simple matter, for it may be argued that it is necessary to draw into the cake only one-third of its weight in water and thus push forward and displace the retained solution. In ordinary practice, however, it is found that when only this amount of water is used less than half the cyanide and metallic contents is removed. The difficulty is to recover an appreciable amount of the remainder without undue dilution of the stock solution. The curve in Fig. II. shows graphically the diluting of the filtrate as the water-wash is drawn through the cake, the vacuum being at a maximum and constant. It will be noticed that the first 40% of the filtrate was only slightly diluted (2 lb. to 1·7 lb.) The next 15% caused a dilution from 1·7 lb. to 0·5 lb. per ton. After this the dilution became less rapid and the removal of the cyanide from the cake was comparatively slow. A low vacuum throughout the operation is desirable for a maximum strength of filtrate, but the curve would indicate that a high vacuum at the start would save time if maintained no longer than is just necessary to withdraw the stronger portion of the filtrate, and then if a low vacuum be applied the greater part of the remaining cyanide, being held by adsorption,* and requiring time of contact to diffuse into the added water, would produce a filtrate sufficiently strong not inconveniently to dilute the stock solution. The extent to which the vacuum should be lowered depends on the porosity of the cake, and the length of time that the wash is applied depends on the fineness of the slime. The more porous the cake, the lower should be the vacuum; and the finer and more amorphous the slime, the longer should be the time of contact. There is no way of determining with accuracy the quantity of water giving the most efficient wash consistent with a given strength of filtrate, except by trial.

Washing by Filtration and Decantation. — The simplest of all methods of washing is by decantation. In practice, however, it presents serious objections, not the least of which is the unavoidable loss due to dissolved metal remaining associated with the residue discharged after the final decantation. One example on gold ore will suffice as an illustration. Assuming that the total solution in use is 7 tons to 1 ton of ore, the number of decantations is three and the residue pulp contains 50% moisture. The losses then for various assay-values of original solution are given in the following table, (1) when the added liquor is free from dissolved metal and (2) when, as is usual, the added liquor contains a small quantity of dissolved metal.

These are theoretical losses, on the assumption that the moisture after each decantation is reduced

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* 'Adsorption' refers to a surface condensation whereby the dissolved metal and cyanide are held in a concentrated form at the surfaces of slime-particles. The condition increases with the fineness of the slime.

(1) Added Liquor Free from Metal.		(2) Added Liquor Containing Metal.					
Dissolved gold before first decantation per ton of solution.	Loss of gold in residue solution per ton.	Loss of gold in pence per ton due to added liquor containing per ton.			Total loss of gold in residue solution in pence per ton when added liquor contains per ton.		
Dwt.	Pence.	2 gr.	3 gr.	4 gr.	2 gr.	3 gr.	4 gr.
0·5	0·94	3·76	5·65	7·53	4½	6½	8½
1	1·88				5½	7½	9½
2	3·76				7½	9½	11
3	5·65				9½	11	13

to 50%. In practice, it is often difficult to maintain a residual moisture so low, especially in cold weather, owing to the slowness of settling and the necessity for keeping the capacity of the plant abreast of the mill. Thus if the residual pulp contained 60% moisture, instead of 50%, the total loss would be increased by 8%. When we allow for loss due to adsorption, which in practice always occurs, the above figures may be greatly increased.

In filtering, unfortunately, there is no simple way of calculating the theoretical loss after washing, for comparison with decantation, owing to a number of uncertain factors. The amount of dissolved metal left in a cake has no relation to the richness of the original solution. Thus if a cake be washed with its own weight of water it matters not whether it contained ½ dwt. or 3 dwt. per ton of solution; the loss in the residue solution would be much about the same. In decanting the metal contents are removed by two to four washes or dilutions, while in filtering it is done by an infinite number of washes. It may be proved mathematically that the greater the number of washes with a given quantity of water, the more perfect is the wash. This may be forcibly illustrated by an example: If the total quantity of liquor in use in the decantation process be made the same as in the filter process, say, 3 : 1, the theoretical separation of dissolved metal by three decantations would be only 78·4% In practice 75% would be considered fair work. In the filtration process, with this same proportion of liquor, there is no difficulty in separating 99% of the dissolved metal. Again, in order to get 99% of dissolved metal separated by three decantations it would require a total of 12 tons of liquor to 1 ton of solid. These figures are based on the assumption that the added wash-liquor contains no dissolved metal.

In practice the wash-waters generally contain small quantities of dissolved metal, and the solution in the residue-pulp contains this metal in proportion to the amount of moisture retained, which for filtration is approximately 1 of solution to 2 of solid and for decantation 1 : 1. The loss of metal from this cause is therefore twice as much for decantation as for filtration. Then again the wide difference in the amount of wash liquor required in filtration and decantation processes affects the cost per ton of ore, in the subsequent treatment of the solution to recover the gold and silver. In filtration the amount of wash-liquor used seldom exceeds twice the weight of the solid, and with care may be considerably less without increasing unavoidable loss of metal. On the other hand, with the decantation process five of wash-liquor to one of solid is common practice, and this cannot be economically reduced without increasing the loss of metal.

Another feature for comparison between washing by the two systems is the loss due to re-precipitation and adsorption. The decantation process necessarily involves a successive dilution of the solution, on account of the large volumes in use and to prevent excessive loss of cyanide in the residue.

Dilution of the liquor containing dissolved metal, in presence of finely divided solid particles in suspension, invariably results in loss of dissolved metal, whether it be by re-precipitation or by adsorption. The extent to which these phenomena develop depends on the constituents of the pulp and the fineness of the ore particles. Silica, which is generally present in a gelatinous or colloidal condition, has the property of carrying down an appreciable amount of gold and silver and cyanide by merely diluting the solution with water. Hydrated alumina possesses this property to an extraordinary degree. There is no doubt that much of the loss in residue from the decantation process is due to adsorption and re-precipitation, caused by dilution in washing stages.

These phenomena also take place in the filtration process, but since the solution need not be diluted until the dissolved metal has been removed from the cake by the solution-wash the loss of gold and silver by adsorption is minimised. The loss due to the dilution that occurs in the water-wash is chiefly in cyanide. Again, re-precipitated metal is re-dissolved in passing the strong wash-solution through the cake whereas the weaker solution added in decantation, although potent on natural metallic particles, has but little effect in re-dissolving the re-precipitated metal."—H. FORBES JULIAN.—*Mining Magazine*, October, 1911. (A. R.)

THE GIESECKE MILL.—"It was claimed for the Giesecke mill that it would do at one operation all that Californian stamp and tube mills would do and that at less than a third of the capital outlay.

The mill works on somewhat the same principle as a tube and ball mill combined but is much larger and heavier. That experimented with at the Geldenhuis mine on the Rand was 24 ft. long with two compartments measuring 6 ft. and 18 ft. in length, the short compartment having a diameter of 7 ft. 6 in. and the long compartment a diameter of 6 ft. These two compartments are divided by a plate with circular perforations of three-fourths of an inch. The tube or shell weighs about 15 tons, but inside the tube, steel balls, totaling as much as 20 tons or more, are provided so that the weight of the whole mill, without its charge of ore, is actually in the neighbourhood of 70 tons. The steel balls run from 1 in. to 4 in. in diameter in more than half a dozen different sizes, the large ones working in the larger diameter compartment, where the feed, consisting of 50 tons of ore, is first introduced. The ore is first crushed to a suitable size, then passed into the first chamber by an automatic feeder to be broken by the four largest sized steel balls to a pulp, such as will pass through the perforated dividing plate into the smaller compartment, about 28% of water being used in the operation. For some reason or other the weak cyanide solution is not used until the final grinding operations, a special chamber provided with a mixing arrangement being used for the purpose. Inside the tube, lining plates projecting from the sides are provided.

When charged the mill is revolved at a speed of 24 r.p.m., for which purpose a 400 h.p. motor is provided. It was at one time stated that this mill was crushing 360 tons per day, of which 81% passed a 200 mesh, with only 15% plus 100 mesh. This, however, is far from being a true representation of the mill's capacity because if such a tonnage was handled in that period, a very much smaller percentage of the pulp would pass the necessary mesh of 200 to allow proper extraction to take place. A more correct duty would probably be 250 tons per day and then the grading analysis would be more approximately as follows: +60, 6%: −90, 15%: +200, 15%: −200, 64%, which is a far different result than that claimed to have been produced shortly after the mill was started. With the above results it will be seen that the Giesecke mill leaves much to be desired, as over a third of the pulp would need to be reground, for which it seems no adequate arrangement is provided; or if on the other hand the charge was retained in the mill until the whole of the pulp was sufficiently ground for proper treatment, the duty of the mill would be materially reduced.

During the four months' trial stoppages for want of spares or other shortcomings have taken place rather frequently, which go to show that the mill has failed to come up to anticipations. In fact, the Trials Committee has found it necessary to reject the mill, because it failed to come up to the guarantees. Whether suitable modifications will be made to make it a workable machine remains to be seen, but probably the experiments will not be altogether lost, if full advantage be taken of the lessons learned in comparison with the ordinary tube and ball mill.

The drawbacks to the Giesecke mill are largely due to its abnormal wear and tear when compared with the Californian stamp mill; in fact it is estimated that the wear and tear of the steel balls must amount alone to about 1s. 6d. per ton, while it is so large and unwieldly that even the end plate also is effected by wear and tear. Under these circumstances it will be seen that the favourable opinions formed concerning this machine were altogether too premature, the costs of wear and tear and the running costs being so high as to make it quite incapable of competing with the Californian stamp mill. These costs were regarded from the outset with some suspicion, but it was the low first capital of £6,000 as compared with £20,000 for a stamp mill and necessary tubes, that led to so much interest being taken in the Giesecke mill trials on the Rand.

In considering the Giesecke mill one is inclined to compare it with the ball mills so extensively used in Western Australia, apparently with some success, but in comparison with these it appears an unwieldly machine, and many practices in its use are at direct variance with many essentials found necessary in Australia for success in their manipulation. In ball mills the weight of the steel balls is only about 2 tons, as against 20 tons in a Giesecke mill, the duty in the former being from 30 to 100 tons per day, according to size. Even in a ball mill the consumption of steel runs about half a pound per ton milled, so that with ten times that amount the wear and tear must of course be greater. Then in the ball mills moisture is found to greatly increase the wear and tear, and is consequently kept down, while in the Giesecke mill the process with the Rand experiments was a wet one using 28% of moisture, so that there is little wonder that the wear and tear was found to be excessive. The cost of milling by the ball mills in Australia seems to be a little less than the lowest cost of milling on the Rand with the California stamp and tube mills, but the stamp mill has the advantage of greater reliability, and when aided by tube mills can perform a duty up to 20 tons per stamp per day.

In conclusion it must be pointed out that the excessive weight of steel balls and other moving parts of the Giesecke mill seem to have been fatal to its success on the Rand, as also the fact that it appears to work quite on opposite lines to those found necessary for success in other ball mills. Still it is hoped that although the Giesecke mill has been rejected by the Mines Trials Committee on the Rand, some lessons at least will have been learned from the tests that may with advantage be applied to the Giesecke mill, so as to better fit it for conducting further experimental milling operations on the Rand. Whether it will ever, as originally claimed, succeed in displacing the California mill, however, seems more than doubtful, no matter in what way it may be improved or modified."—*Mining and Engineering World*, Oct. 7, 1911, p. 688. (A. R.)

THE BUTTERS FILTER.—"An official report is just out dealing with the results obtained from the trial of the Butters filter at the Crown Mines. The broad result is that the Butters plant has been taken over as a permanent feature of the Crown Mines equipment and is also to be enlarged to a capacity of 1,200 tons per day. The report gives the following interesting information:

(1) The true average of Adair-Usher residue was shown to be 0·423 dwt., whereas the average of vat samples, taken in the usual way, gave 0·225 dwt. per ton. (2) The Butters filter made a total recovery of 0·333 dwt. per ton from Adair-Usher residue averaging 0·423 dwt. To quote the report: 'The conclusion reached from this six months trial may be stated as follows: (1) Owing to the fact that in the past Adair-Usher residues have been considerably under-estimated, as confirmed by recent investigation, the filter may reasonably be expected to show a net saving over the Adair-Usher process of from 0·20 dwt. to 0·5 dwt. per ton of slime filtered. (2) Under normal conditions the filter may reasonably be expected to show a gross saving over the decantation process of about 0·20 dwt. per ton, or a net saving of about 0·12 dwt. per ton of slime filtered. (3) As regards cost of treatment this has been given at 9d. per ton during the earlier stages of the trial."—*The Mining Magazine*, August, 1911, p. 112. (A. R.)

MINING.

A POINT OF VIEW ON ORE RESERVES.—*Abundant Reserves Cause Feeling of Security.*—"Two important aspects of the question are overlooked: (1) The effect of good ore reserves upon the managing and working force; (2) the greater hope of finding new ore while the mine is in pay, rather than when pay is exhausted. The feeling of security which abundant ore reserves give to an organization is an enormous aid to the morale at the mine, encouraging the manager, foreman, miner, down to the humblest trammer. It gives them a feeling of permanency, which is sadly lacking when the usual hand-to-mouth methods are pursued. If a miner sees that all the available ore in a mine is rapidly being exhausted, and that his job is not likely to last, he becomes restless and careless. This feeling of uncertainty is one reason why our precious-metal mine workers are such nomads, travelling about from mine to mine. Few of our mining camps have any element of permanency; no attempt is made to soften the surroundings. The practice of taking out all the ore reserves, and then looking for more ore, is discouraging to the miner. The depressing effect of barren drifts and abandonded crosscuts upon the minds of men whose duty it is to develop a property is a potent influence in discouraging further exploration. *Per contra* ore in sight encourages both miner and manager in the search for new ore.

The best time to do development work is while a mine is in full operation, with pay in sight. 'Follow your ore' is a good maxim, but a better one is, 'Follow ore indications while you still have some in sight.' Many a rich pocket or streak of ore, perhaps leading to other bonanzas has been passed over or left behind in the hurry with which we deplete our ore reserves. For example, in 1884 I visited one of the most important mines in Utah, possessing at that time an ore body from which about $10,000,000 was later extracted. It was mined rapidly and without proper precautions, with the result that the ground caved about 500 ft. in depth after the ore was taken out. This prevented judicious exploration of the ground lying contiguous to the ore body. As the occurence of ore in limestone replacements is peculiarly uncertain, it may readily be believed that splendid opportunities were lost in not looking for extensions of this ore body while it was in process of extraction. Development from stopes is just as important, in narrow-vein mining as that carried on by levels, raises or winzes. In many cases I feel confident that silver-gold properties would be working to-day had not the hurry and greed of their owners demanded the extraction of every possible dollar with the utmost speed, without due regard to the safety or future of the mine. The influencing cause behind a hasty extraction is, in many cases, the desire to increase the speculative value of the stock by rapid production of ore.

Too Rapid Extraction Hinders Exploration.—Why is it that many of the well known mining districts of Europe are still active producers of ore, while our Western mining camps are deserted? We seldom hear now-a-days the names of the famous mining camps which were on everybody's lips less than 10 years ago. I believe that many of these camps would be alive to day had not the owners of the mines been in such a hurry to extract the ore, to the neglect of exploration work. It is hard to raise capital for the further development of a property that has the reputation of having been worked out, even though a large portion of its territory may still be undeveloped. It is difficult to persuade owners of mines to develop their properties unless you can show them some ore. For this reason the psychological moment at which to propose development, is while the mine is working in pay ore. It is difficult to extract ore with the rapidity to which we are accustomed, and at the same time carry on systematic and judicious development in the direction of following the ore. This question was debated at some length in the spring of 1904. H. C. Hoover argued that the more quickly a mine was exhausted, the greater the profits would be. To this W. R. Ingalls replied that while the theory of intense production is correct, it is practically applicable only to gold mining, because if other kinds of mines were operated uniformly in that manner, the effect upon the price of their products would be so adverse that probably no profit would result. Mr. Hoover accepted this point of view The principle enunciated by Mr. Hoover leads to a discussion of the effect exerted by the nature of a deposit upon the amount of reserve to be maintained.

Reserves Depend upon Nature of Ore.—Mr. Hoover's views are undoubtedly sound when applied to gold mining in South Africa. The conditions there are such, the quartz being of uniform width and regular value, as to enable the operator of a mine to look many years ahead, and after judicious development to figure approximately the life of his mine. Maintenance of ore reserves can then be accurately regulated to yield such profits as the capitalisation of the property demands. The copper lodes of upper Michigan are another case in point. In both of these districts the psychological advantage of maintaining large reserves is not so great as in the cases previously mentioned. It is plain, therefore, that the character of the ore and its occurrence, its eveness or uneveness of value, must be an influencing factor in the settlement of this question In the case of limestone replacement, as at Bingham, where lenses of ore having irregular dimensions occur, and in the case of Cobalt, where the narrow veins of extremely rich ore are exceedingly uneven in value and may at any time be lost, the psychological value of ore reserves is much greater. One might even formulate a law that the advantage of maintaining ore reserves is directly in proportion to the uncertainty of the deposit. Putting it another way, when a mine is operating in small veins of rich ore, interrupted by barren zones, where the ore is of uncertain occurrence and where a maximum amount of development must be done to discover new ore, the wisdom of maintaining a certain amount of ore in sight is much more apparent than in a mine blessed with ore of greater regularity. From the psychological point of view, likewise, it is apparent that the more irregular the occurrence of ore, the greater the care that should be exercised in providing ore reserves. The loss of the ore streak, often a temporary phenomenon, will not affect the mind of a mine manager or foreman, if he has abundant reserves to draw from, nearly as much as it would if he had already exhausted his orebody. From the practical point of view, it is of great importance to prospect the ground while mining is going on. If the little signs which lead to ore are not followed immediately, they are generally overlooked, and the opportunity to find additional ore is wholly lost. Veins diverge, and not infrequently form wedges behind foot or hanging walls. If a crosscut is not driven almost immediately upon the discovery of such a phenomenon, the circumstances which seemed to demand a crosscut at that point, when the drift was freshly broken, are forgotten and the crosscut is never driven.

To sum up, the time to prospect is while the mine is producing, both because finances more readily permit it, and because the knowledge of ore occurrences is freshest and keenest; furthermore, the buoyancy and energy with which the possession of good ore reserves imbues the manager is an important psychologic factor in promoting efficiency and maintaining organisation." B. B. LAWRENCE, *Engineering and Mining Journal*, Oct. 1911, p. 743. (A R.)

MACHINE STOPING ON THE RAND.—"The gradual spread of the small stoping machine is being steadily effected along the Reef. The tendency in this direction is becoming more evident month by month, and it is not unlikely that before another twelve months have passed the relative positions of the hand stoper and the machine will be very distinctly reversed as compared with the condition of things a year or so ago. Machines are doing better work to-day on the Rand than they have ever done before. They are beating the unaided aborigine at what has hitherto been considered to be his special business, by doing it more cheaply and quite as well. The hindrance to their more extended employment lies in the want of skilled supervision. If a machine is to be run with the degree of efficiency which alone warrants its use, it must be run intelligently and the supply of trained intelligence, like that of unskilled brute force, is not adequate to the requirements of the industry. It is a difficulty that will be gradually overcome, however, and in proportion as it is remedied so will the advance of the small machine be continued. In connection with this matter, reference may be made to the remarks of Mr. C. D. Leslie, Superintending Engineer to the Consolidated Goldfields group, which have been taken from his annual report. It is interesting to learn that even in the smaller mines of the Rand the use of the machine drill has been found capable of meeting the labour difficulty to a considerable degree. At the Lancaster West, for instance, there are not two hundred hammers in operation, and within the next two or three months the number will have been reduced, in all probability, to not more than seventy-five. Waugh and Ingersoll-Sergeant machines are being run in the stopes, carrying a width of 36 inches, and ten 48 in. holes per shift per machine is said to be the usual work accomplished. The milling width has not been increased, the breaking is cheaper, and the system appears to be giving a very fair amount of satisfaction."—*The South African Mining Journal*, Nov. 1911, p. 362. (A. R.)

MACHINE *versus* HAND STOPING.—"Interesting figures are to hand showing the results of stoping with small reciprocating machine drills as compared with large machines and hammer-boys at the Robinson Deep mine. The comparative figures are as follows:

	Cost per fathom s. d.	Stope-width Inches	Work done per shift Fathoms	Tons
By hand	80 4·7	51·4	0·076	0·915
By small machine	75 7·2	54·7	0·55	6·432
By large machines	72 9·8	59·9	0·63	8·410

From the above figures it appears that the cost of hand labour is above the average; 0·076 fm. per boy shift is bad. This figure should be about 0·1 fm. in good ground and with good hand-stopes. The small machines do comparatively well and the ground must be easy to obtain these results. From a labour point of view a small machine requires the same number of boys as a large machine, and except in stopes where the hanging is bad and a large machine would shake the ground too much, they will not bear comparison with large machines, even when it is required to keep the stope-width as low as possible. The figure 0·63 fm. per large machine is also much below the average for a stoping width of about 3 ft.; this figure should at least be in the neighbourhood of 0·75 fm. per machine per shift. In March 1911, 44·55% of the ground broken at this mine was done by small machines, 31·84 by hand, and 23·61 by large machines." *The Mining Magazine*, Aug. 1911, p. 112. (A. R.)

MISCELLANEOUS.

THE DEVELOPMENT OF THE WITWATERSRAND.—"It was 25 years ago, on the 21st of July, 1886, to be precise, that Johannesburg was proclaimed a township, but it was two months later that several of the principal farms now constituting the Central Rand were declared public diggings, and so began what has turned out to be the largest gold field of the world. It may be that the geological character of the gold-bearing beds has always laid the field open to the charge that they must depreciate in depth if not entirely disappear. In its early days the gold field suffered somewhat severely from these charges, and there are those to-day, after a period of 25 years, who still maintain that the conglomerate beds of the Rand gradually become impoverished as greater depths are attained; in fact, one engineer has stated recently that below a depth of 4,000 ft. mining operations cannot be profitably carried out on the Rand. This statement is based on an apparent general falling off in grade with depth and an expectation that as the mining operations increase in depth the costs of working must also necessarily increase.

On geological grounds there was every justification for regarding the permanence of the gold-bearing beds in the early days with suspicion, and, owing to the unique character of the deposits, perhaps, some of the early engineers cannot be blamed for condemning the fields. There can, however, be little justification for fixing a vertical depth limit of 4,000 ft. to payable mining to-day when, as a matter of fact, the gold field has not reached its prime as a producer, and in several instances has even passed the depth limit pessimistic engineers would seek to place upon its operations. Judging from actual mining operations on these fields, down to a depth of nearly a mile, neither underground temperature, so great a bugbear elsewhere, or superincumbent pressure with proper mining methods, can prevent operations being carried on to at least three times 4,000 ft. in depth so long as the yield of gold does not fall below two-thirds of its present average. Twenty-five years' mining have shown on the Rand that poor areas alternate with good areas to a far greater extent than was anticipated in the early days; even the beds vary considerably in character as the early engineers anticipated, but, taken over extensive areas, there is a remarkable general consistence both in character and yield.

In the gold field's infancy it had the advantage of working the outcrop reef where natural concentration had been going on for many centuries and oxidation rendered the gold comparatively easy to extract from the ore. Hence we find that for the first 4 or 5 years the yield per ton milled varied from £2 to £4, as compared with less than 30 shillings to-day. After 5 years' working, however, several of the richest mines reached the unweathered pyritic zones and the yield per ton fell on the average to 35s. per ton, but in 1894 the introduction of the cyanide process added a pound sterling to the recovery and proved the salvation of the Rand, for to-day, without the cyanide or equally successful process, it is doubtful whether more than two or three mines would have been able to carry on operations at a satisfactory profit.

It was not until 1902 that reliable details of mining costs were published, and then we find the working costs figuring at 25s. 9d. per ton, a level at which, with to-day's average recovery and scale of working mining operations on the Rand, would be attended with but small success. Since that date, however, working costs have fallen gradually until in 1909 they reached their lowest level of 17s. 1d. per ton, but since that date an opinion has been gaining ground that however desirable it may appear to have low operating costs they may be obtained at a sacrifice of grade, for during the last 18 months the new policy has at several mines added slightly to the cost. It is worthy of note that while during the last 10 years working costs have fallen per ton milled to the extent of 8s. per ton, the average yield per ton has fallen to a much greater extent, to be exact, 14s. 6d. per ton.

A chronic scarcity of unskilled labour has pursued mining operations on the Rand practically during the whole 25 years of its existence. There have been few occasions during that long period when the wants of the mines have been adequately supplied. Perhaps the best labour position existed in 1906, when nearly 50,000 Chinese were imported as a last resort to supplement the local unskilled native labour at considerable expense to the mining industry. The welfare of the mining industry was, however, sacrificed for political expediency, and notwithstanding the immense outlay incurred in the introduction of Chinese labour with its increased efficiency—when compared with local native labour—legislation was introduced and passed compulsorily repatriating the Chinese. Chinese labour, however, left its mark on the Rand mining industry for good, which will be seen by the fact that during its stay on the Rand skilled white labour was able to produce 1,114 tons per head per annum as against 660 tons before its introduction. Despite the introduction of additional machine drills and the milling of larger tonnages of low-grade ore the efficiency of white labour is only 906 tons per head to-day. If the same test be applied to unskilled labour we find that before the introduction of Chinese labour on the Rand a good output per native employee was 105 tons per year per hand employed, but with 42% of Chinese labour the average production per head increased to 126 tons per annum. When the Chinese proportion fell next year to 31% the tonnage per unskilled labourer fell to 109 tons per head per annum.

Another interesting feature is the fact that since 1897 white labour cost per ton of ore milled has steadily decreased, but since the departure of the Chinese the cost of unskilled coloured labour has steadily advanced. For instance, in 1897 white skilled labour cost 9s. 5d. per ton, and even in 1906 the cost was still 9s. per ton, and in 1907 as high as 10s. 6d., but in 1908, after the strike, it was reduced to 6s. 5d. per ton, while in 1910 it was reduced to 6s. 4d. per ton. On the other hand, when 42% of the unskilled labour on the mines was Chinese labour the cost of unskilled coloured labour was but 3s. 11d. per ton of ore milled, but in 1909, only 3 years later, it had risen to 4s. 2d. per ton, and has been increasing ever since.

As far as the figures are concerned they show conclusively the steady decline which has taken place per ton since the war, in cost, profits and yield, but they are eloquent of the Rand's progress as far as milling capacity and stamp duties since the introduction of tube mills in 1905 and gross yields are concerned.

Up to last year the total dividends paid to shareholders showed a steady growth in somewhat the same relation as the total yield, but last year there was a setback approaching half a million sterling, and so far this year there has not been any indication of the Rand returning to its former total-dividend standard, although the total yield has maintained its steady growth, as the following figures will show :

Year			Total yield, £	Total dividends, £	Per cent of yield.
1902	..	...	7,179,674	2,121,126	29·5
1903	...	...	12,146,307	3,345,502	26·7
1904	...	...	15,539,219	3,877,624	24·9
1905	...	...	19,991,658	4,832,436	21·9
1906	...	...	23,615,400	5,565,969	23·5
1907	...	...	26,421,837	6,962,420	26·3
1908	...	...	28,810,393	8,536,773	29·6
1909	..	...	29,900,359	9,310,751	31·1
1910	...	...	30,703,912	8,876,085	28·8

If, however, we compare the percentage of the dividends paid to that of the total yield during the last 10 years, they have varied somewhat, but there is one remarkable fact, and that is that the percentage of the gross yield paid in dividends was highest, namely 31%, in 1909, when the costs of working were lowest, namely, 17s. 1d. per ton milled. Further, if the figures be closely examined it will be found that the percentage of the total value of the output paid in dividends to the shareholders bears a striking relation to the working costs, for the lower the working costs per ton milled the higher the percentage dividends paid to the shareholders. This must be taken to be a convincing proof of the integrity of the mine management on the Rand and of the fact that after all low tonnage costs mean higher dividends, a condition which, of late, in certain quarters on the Rand, has given rise to considerable discussion."—ROWLAND GASCOYNE.—*Mining and Engineering World*, Oct. 14, 1911, 744. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE WITWATERSRAND SYSTEM. By W. E. BLELOCH
Pamphlet and Map. London : E. Marlborough & Co. Price 42s.

It is a fact worthy of note that all those banket beds which have been proved to be important were discovered early in the history of the Rand and that in spite of much search no other beds containing similar gold contents have been found since. The Main Reef series in the Central Rand ; the Van Ryn in the Eastern Rand ; the Nigel Reef in the Heidelberg district ; the Randfontein, the Botha's, and the Battery Reef series in the Western Rand ; the Buffelsdoorn and others in the Klerksdorp district ; and the Rietfontein series lying by itself to the north of the Main Reef series, were all discovered quite early and no other line of banket of comparable value has been discovered since. At the same early stage the several other horizons of poorer banket principally above, but also below, the Main Reef series became known, and it may be said with regard to such beds also that no important later additions to their number have been made. Not only did the early prospectors discover so much, leaving, as events have turned out, their successors so little reward outside of that which might come to them from the development of the initial discoveries, but in the same note-worthy manner the early geologists and engineers put together the framework of the structural geology of the field upon which their successors also have been content to build.

The work done since those days has bridged many of the early gaps upon the surface ; has established the continuity in depth ; and has put together such a record of structural geology and stratigraphy as will afterwards doubtless be referred to as a classic. From the time in the year 1886 when gold was first discovered in the banket beds, systematic and most effective search went out to the east and west of the point of discovery. So keen was this search and moved by so much impetus that breaks in the surface continuity were spanned, each extension becoming the index to the one further ahead. Then when the breaks became too big, giving the Main Reef series space to change beyond recognition, the quest was still continued, using the trend of the enclosing sediment lines as a guide, until in the end the encircling traverse became complete, east returning to the starting point through west, and west through east. It was then that the position became reviewed, and the ideas concerning the correlation of the different beds to one another shaped themselves slowly, unforced, and in conformity with reason. Among other determinations accepted was the one that the Van Ryn series in the east was the continuation of the Main Reef series of the centre, this dignity falling to the former because, being found within the line of search, it contained important resources in gold and was of banket identical in appearance to that of the standard series. All the work done since then and all the observations made have confirmed this correlation, in which confirmation the lessening of the gap between the workings on the two series has been the most important factor. This gap, caused by a thickness of more recent beds upon the surface, has given way before the advance of underground work and the probing of diamond-drills, till, in spite of local disturbances, the terminals have been so joined that there exists in the mind of most responsible persons today no doubt of the correctness of the identity of the two series.

Mr. Bleloch in his geological map of the Southern Transvaal and in the short treatise, 'The Witwatersrand system,' which accompanies it, questions and refuses this identity, affirming at the same time that the Van Ryn series is a continuation of the Rietfontein series. It was of course open and still is for anybody to bring forward and submit data that might bring the acceptance of this position into question, and, in doing so, to leave the community his debtor, always provided that proper respect were paid to the effort and the intelligence of previous workers. There is however very little of submission in the way that Mr. Bleloch has put his ideas forward, nor does he show a proper regard for the opinions of those geologists who have studied the same ground; he alone would be therefore to blame if his views were rejected with scant consideration.

Before weighing his views it is necessary to know the position generally assigned to the Rietfontein series. It was long held that this was a banket bed at a definite horizon in the lower portion of the Witwatersrand system, but during recent years this view has been questioned, more particularly by Hatch and Corstorphine, who state that this series has certain resemblances to the overlying Elsburg series, which again they considered to be in all probability unconformable to the beds of the Witwatersrand system. Mr. Bleloch in affirming the identity of the Van Ryn series with the Rietfontein, assigns to the latter at the same time a definite horizon in the above mentioned system, the horizon in fact which according to general opinion is occupied by the Main Reef, a view that compels him to place that series a good step higher in the system. It is in this manner that he would prove that in the place of one there are two important reef-horizons in the system, a fact which he thinks should provide all the encouragement necessary for greater and renewed activity in prospecting.

Before however such further prospecting is justified, it has to be considered that that which has already been done leaves small hope of any extent of valuable reef along a possible second horizon remaining undiscovered, and that for this reason the encouragement from theoretical consideration must be better based than would be the case if the previous work in that direction had not been so thorough. Unfortunately, though it is claimed that the views in the treatise under review are built upon many new facts, these new facts do not stand out clearly, and the bare statements consequently carry but little conviction.

These statements, for whatever they may be worth, Mr. Bleloch bases primarily upon the pronounced apppearace in one section of the Central Rand of an additional bed of quartzite between the Main Reef series and the Hospital Hill Quartzites, and he states that along any normal line of section the Rietfontein series and its overlying reefs occur in conformable position overlying the Hospital Hill quartzites, this being evidenced, according to him, in the line of section mentioned above, by the banket beds on Langermans Kop, which he says belong to the South Rietfontein series. It is, however, held by all the geologists who have studied this line of section that this additional bed of quartzite is but a duplicated outcrop of the Hospital Hill quartzites, this view having first been presented by Walcot Gibson and afterward by Hatch and Corstorphine, who explained that no duplication of the Main Reef series had occurred at the same time because the fault which was the cause of this disturbance did not reach that reef series but changed its direction somewhat to run parallel with it. This view has again lately been put forward in the most convincing and scholarly manner by E. T. Mellor in a paper read before the Geological Society of South Africa and entitled 'Some structural features of the Witwatersrand System of the Central Rand with a note on the Rietfontein series.' As Dr. Mellor is one of the geologists on the staff of the Geological Survey and his paper was read by permission of the Minister of Mines, there seems little doubt but that it indicates what will likely be the official view of the question. In that paper also Dr. Mellor gives some facts about the occurrence of the Rietfontein series and that on Langermans Kop, from which it appears more than likely that these two occurrences of banket are rightly correlated, as they had previously been, with the Elsburg series, and that where they are now found they lie unconformably to the beds of the Witwatersrand system, in which they can therefore occupy no definite horizon.

Between the reasoning of the geologists mentioned and the affirmations of Mr. Bleloch there can be no ground for hesitation; the reasoning and the general view will remain while the affirmations of Mr. Bleloch, which form the basis of his arguments, must go. It may also be said that even if Mr. Bleloch's reading of the lower horizons of the Witwatersrand system had been correct, the other evidence based upon the identity of the overlying reefs and upon the characteristics of the two series themselves, would remain invincible to arguments that would place the Main Reef and the Van Ryn series at different horizons."—S. J. TRUSCOTT, *The Mining Magazine*, Aug. 1911, p. 153. (A. R.)

Adams, Henry, and Matthews, Ernest R. Reinforced Concrete Construction in Theory and Practice. 8vo,, pp. 330. *Longmans.* Net 10s. 6d.

Alford, L. P., Bearings and Their Lubrication. 8vo., *Hill Publishing Co.*, Net 10s. 6d.

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Abstracts of Patent Applications.

(C.) 433/10. William Henry Rufus Munnery. An improved double action stone crusher. 3.9.10.

This invention relates to an improved machine for crushing and breaking rocks, ores or other hard material.

The machine is a double ended crusher fitted with fixed and moveable jaws at each end, the moveable jaws operated by a double action swing cam so actuated by an eccentric on a central shaft that stones are crushed at both ends by alternate strokes of the cam.

(C.) 474/10. John Webb. A self-supplying water power pumping machine. 26.9.10.

This invention relates to an apparatus for a self-supplying water pumping machine.

The inventor makes, amongst others, the following claim:—"An apparatus whereby a wheel is kept in motion by means of water from a tank which is itself kept supplied with such water by a pump worked by the said wheel."

(C.) 250/11. Wilhelm Gunther. Processes and apparatus for extracting and lixiviating ores metallic residues and the like. 3.5.11.

This application refers to a process for treating ores, etc., characterised by the raw material being subjected to the lixiviation process in great heaps on a water-proof or lye-proof surface.

(C.) 269/11. James Lester Clark. Improvements in pipes or launders particularly suitable for conveying abrasive or corrosive material. 11.5.11.

This application is in respect of a method of manufacturing pipes, launders, etc., suitable for the conveyance of abrasive or corrosive material.

The pipes, etc., are proposed to be made of sheets of solidified wood pulp, or canvas, or other suitable textile material, waterproofed and bound together with a suitable agglutinant to which an abrasion resisting material such as cement, sand or flint is added. The sheets are laid together on a mandrel or other suitable form, and subjected to heat and pressure.

(C.) 281/11. Peter Norman Nissen. Improvements in driving means for the ore feeders of stamp mills. 18.5.11.

Patent application No. 281/11 made by Mr. P. N. Nissen describes an improvement in the mechanism for actuating the driving wheel of ore feeders for stamp mills.

The intermittent motion of the feeder plate is obtained from one of the working stamps through the usual arrangement of bumper lever and rod or cord, actuating the clutch mechanism described in this specification, the clutch being so connected with, and arranged in relation to, the driving wheel on the feeder as to impart motion to it in one direction only.

The clutch consists of two similiar discs, mounted on a common axis, with sufficient space between their inner faces to permit the rim of the driving wheel to be between them. Each disc carries two projections on its inner face, one being placed at a greater distance from the disc centre than the other, while the two are some 60 degrees apart circumferentially. When the discs are held together, one on each side of the driving wheel, there is thus one projection on each disc on the inner side of the driving wheel rim and one on the outer side. The discs being loose on a common axis can rotate relatively to one another and are caused to do so, but in opposite directions, by a lever which is set loosely between other projections on the inner sides of the discs at that part of the circumference remote from the driving projections. The outer end of this lever is connected to the bumper rod or other source of intermittent motion, and when pulled in one direction one of the projections becomes the fulcrum of the lever, and the reaction between the fulcrum and the short arm of the lever makes the discs rotate in opposite directions, which causes the driving projections to nip the rim of the driving wheel, carrying it round in proportion to the extent of the motion of the bumper rod. When the force is removed the clutch releases the rim of the driving wheel, which remains stationary, and the whole clutch mechanism gravitates to its original position, ready for the next application of the power.

The specification describes also a modification of this arrangement, which is very similar.

(C.) 385/11. Alfred Andrew Lockwood. Improvements relating to the extraction of the precious metals from ores by the cyanide process. 8.7.11.

This patent refers to the recovery of precious metals from cyanide pulp or slime by means of mercury in a state of division, and in the presence of unamalgamated zinc, preferably granulated.

The ore is placed with cyanide solution in a tower and circulated by means of a pump or otherwise. Zinc in the form of a powder, chips, turnings or granules is added, either at the commencement of the treatment with the cyanide, or at the time of adding the mercury, which takes place after the pulp has been circulated for some time.

When the operation is completed the mercury is obtained after settlement, and the precious metals recovered in the usual manner.

(C.) 526/11. William Lambert Imlay. Improvements in apparatus for the treatment of gold and silver ores. 25.9.11

This application refers to an apparatus for agitation of crushed ore with a suitable solvent and consists of a horizontal rotatable cylinder with specially arranged ribs or other means for giving a series of falls to the moving pulp.

(C). 533/11. Turbo-Amalgamator and Extraction Co., Ltd. (1), A. A. Lockwood (2). Improvements in the amalgamation of the precious metals and in apparatus therefor.

This is an amalgamation process in which the pulp is caused to flow from a conical bottom vat into a revolving amalgamator treated with mercury in a state of division, and re-elevated to the storage vat by means of compressed air. The treatment may also be combined with a dissolving process with precipitation of the values in solution on copper plates through which is passed a current of electricity.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The numbe given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 582/11. James Hamilton (1), Amstruther Macadam (2). Improvements in apparatus for use in recovering diamonds and other precious stones from matter containing the same. 26.10.11.

(P.) 584/11. Charles Legward. Improvements in means for transferring comminuted solid matter from one liquid to another. 26.10.11.

(P.) 585/11. Croft William Caw (1), Frederick Roberts (2). The handy drill holder. 27.10.11.

(P.) 587/11. Giuseppe Antonio Pietro Provay. Improvements in bottom discharge buckets, boxes, skips and the like for handling, or receiving and discharging coal, grain and other materials. 27.10.11.

(C.) 589/11. Emil Deister. Improvements in ore concentrators. 27.10.11.

(C.) 590/11. Ernest Clayton Johnson. Improvements in grinding mills. 27.10.11.

(P.) 591/11. Edmund Jerrard. Improvements in safety devices for lifts and the like. 28/10/11.

(P.) 592/11. Wilhelm Manss. Improvements in electrical resistances. 28.10.11.

(P.) 593/11. Wilhelm Manss. Improvements in electro-magnetically actuated percussive apparatus. 28.10.11.

(C.) 594/11. William Thomas Bell (1), Arthur John Wood Graham (2). Improved construction of valve for air compressors, blowing engines and other fluid pressure engines or apparatus. 28.10.11.

(P.) 596/11. William Blane Dodds (1), David Charles Bowen (2). Improvements in linings for the ends of tube mills and the like. 30.10.11.

(P.) 597/11. Edward Harrison. Improvements in ventilating drives raises and the like. 31.10.11.

(P.) 599/11. Augustus Meyrick Hewett Thorp (1), John Evelyn Hoff (2). Improvements in apparatus for braking and stopping runaway skips or the like. 1.11.11.

(P.) 600/11. Hugh Denehy (1), Frederick William Bergner (2). An improved automatic overwind trip-gear for electric hoists. 2.11.11.

(C.) 601/11. Ciro Fidel Mendez (1), Roberto Ypina (2). Propulsion device. 3.11.11.

(C.) 603/11. Wilhelm Meyer. Improvements in detonating compounds for use in rim-firing cartridges and percussion caps. 3.11.11.

(C.) 604/11. Robert Thomson (1), William Affleck Thomson (2). Improvement in ferro-concrete floors. 3.11.11.

(C.) 605/11. Robert Thomson (1), William Affleck Thomson (2). Improvements in the reinforcement of columns, ferro-concrete pillars and the like. 3.11.11.

(C.) 606/11. Thomas Dawson. An improved process and apparatus for attaching and fixing bands to articles made of wood and other similar materials. 6.11.11.

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(P.) 608/11. James Esson. A new method for generating power in primary motors. 7.11.11.

(P.) 610/11. Peter Macadam. Improvements in valves. 8.11.11.

(P.) 611/11. Cornelus Marinus Roeters van Lenulp. Improvements relating to the mortar boxes of stamp mills. 9.11.11.

(C.) 614/11. Dr. Robert Gans. Improvements in and relating to the softening of water. 11.11.11.

(C.) 615/11. Dr. Robert Gans. Improvements in and relating to the manufacture of zeolites or hydrated alumino silicates. 11.11.11.

(P.) 616/11. Charles Edwin Norton. Improvements in ratchet spanners. 11.11.11.

(P.) 617/11. Thomas Dorman Kirkwood. Chuck for rock drills and the like. 13.11.11.

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(P.) 619/11. Rollo Bowman Ballantine. A condenser leakage indicator. 15.11.11.

(P.) 620/11. Robert Rodger. Improvements in mills or apparatus for reducing ore, or other material. 16.11.11.

(C.) 622/11. Leon Del Monte y Aldama. An improved process for obtaining various products from solid carbonaceous materials. 17.11.11.

(C.) 623/11. Charles Anthony Vandervell (1), Albert Henry Midgley (2). Improvement in and relating to variable speed self regulating dynamo electric machines. 17.11.11

(C.) 624/11. George Gilbert Bell. Improvements in apparatus for heating liquids, gases or other media or for producing steam by means of electricity for heating or cooking purposes. 17.11.11.

(C.) 625/11. Norbert Ceipek. Improvements in the manufacture of a safety explosive. 17.11.11.

(C.) 626/11. Norbert Ceipek. Improvements in the manufacture of a safety explosive. 17.11.11.

(C.) 627/11. Pierre Ferra. A new compound applicable as a heat insulating material and for other purposes. 17.11.11.

(C.) 628/11. Sidney Williams. Process for separating precious metals from their ores. 18.11.11.

(P.) 629/11. Edward Dollar. Improvements in means for lifting and supporting, and thereby throwing out of action the stamp of stamp mills. 18.11.11.

(P.) 630/11. George Smith. Improvements in or relating to tube mills and the like. 18.11.11.

(P.) 631/11. Peretz Manoim (1), Benjamin Harry Chein (2), Morris Samuel Sachs (3). Improvements in checking runaway skips and the like. 20.11.11.

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THE JOURNAL

OF THE

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Vol. XII. JANUARY, 1912. No. 7.

Proceedings
AT
Ordinary General Meeting, January 20, 1912.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, January 20th, Mr. H. A. White (Vice-President) in the chair. There were also present :—

48 Members : Messrs. K. L. Graham, J. Gray, A. Richardson, Dr. W. A. Caldecott, A. McA. Johnston, J. R. Williams (Members of Council). H. A. Adams, W. Beaver, J. H. Brown, J. Chilton, C. Crichton, J. McK. Dixon, L. L. Ellis, R. C. Evans, R. Gascoyne, B. C. Gullachsen, W. T. Hallimond, H. L. Harland, C. E. Hutton. W. H. Jane, A. J. Johnson, R. N. Kotzé, A. M. Kratz, G. A. Lawson, H. Lea, J. Lea, C. D. Leslie, Dr. D. Macaulay, M.L.A., W. Mackesy, G. Melvill, W. I. K. Minards, P. N. Nissen, F. S. Pearce, F. D. Phillips, H. W. Pidsley, H S. Potter, W. S. V. Price, J. F. Pyles, M. H. Reid, F. W. Rush, G. H. Smith, A. L. Spoor, A. Thomas, J. E. Thomas, C. Toombs, T. G. Trevor and J. T. Triggs.

17 Associates and Students : Messrs. L. T. Bayes, J. P. Beardwood, H. G. Brickhill, C. A. Damant, A. Gilbertson, H. L. Krause, G. F. Mathews, D. McDivett, W. H. Pickburn, F. J. Pooler, R. A. Porter, A. G. Rusden, E. H. Spence, H. Stadler, W. Waters, A. Willcox, and S. A. Woolf.

19 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as published in the December *Journal*, were confirmed.

NEW MEMBERS.

Messrs. T. Johnson and R. Gascoyne were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been duly elected as follows :—

EMETT, HARRY CHARLES, Nourse Mines, Ltd., P. O. Box 88, Denver. Smelter.

GEMMELL, HUGH, Transvaal Gold Mining Estates, Ltd , P. O. Box 40, Pilgrims Rest. Mining Engineer.

HARTOG, VICTOR, Alexandersfontein Hotel, Kimberley, C.P. Mining Engineer and Geologist.

KELLY, ANDREW, Transvaal Gold Mining Estates, Ltd., Pilgrims Rest. Mining Engineer.

KETHEL, ROBERT, Transvaal Gold Mining Estates, Ltd., Pilgrims Rest. Assayer.

KIRBY, ALFRED GEORGE, Box 740, Cobalt, Ont. Canada. Metallurgist.

MACKESY, WALTER, Central Mining and Investment Corporation, Ltd., P. O. Box 4563, Johannesburg. Draughtsman.

REES, DAVID, Transvaal Gold Mining Estates, Ltd., P. O. Box 65, Pilgrims Rest. Mine Overseer.

REYNOLDS, ARTHUR EDWARD, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Tube Mill Foreman.

ROBERTS, EDWARD JOHN, Transvaal Gold Mining Estates, Ltd., Pilgrims Rest. Mill Manager.

The Secretary notified that since the last meeting of the Society the following Associates had been admitted by the Council:

As Associates—

BAYES, LENNOX TODD, Simmer Deep, Ltd., P. O. Box 178, Germiston. Assistant Sampler.

EDGAR, JAMES, Transvaal Gold Mining Estates, Ltd., P. O. Box 23, Pilgrims Rest. Cyanider.

KNIGHT, LLEWELLYN, Transvaal Gold Mining Estates, Ltd., Pilgrims Rest. Cyanider.

RICHARDSON, KENNETH, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Assistant Sampler.

SHKAIDY, JEHUDA LOUIS, Little Octopus Gold Mine, Post Office, Pilgrims Rest. Cyanide Manager.

As Students.—

STIRTON, PERCY WILFRED, Transvaal Gold Mining Estates, Ltd., Pilgrims Rest. Learner in Assay Office.

WHITE, JAMES LOVE, Knights Deep, Ltd., P. O. Box 143, Germiston. Learner in Cyanide Works.

GENERAL BUSINESS.

The Secretary : I wish to draw the attention of members to the Exhibition which we propose to hold early next month. Up to the present we have had very few applications by members and others for space to show models or interesting pieces of machinery which would be an attractive feature of the Exhibition, and we are very keen on making that portion of the exhibition a success. There are many members who, at one time or another, have described items of interest at the meetings and in the *Journal*, and we should like to have them represented in the Exhibition. If any member will advise me as soon as possible as to any model or machine he may wish to exhibit I should be pleased to hear from him.

I have also received (for the benefit of those members who have not yet seen them) a number of the Stobie respirators, which may be seen on the table during the recess.

THE DETERMINATION OF CARBON DIOXIDE.

Mr. Chris. Toombs (*Member*): There is a subject of great present interest to many of our members which I would like to bring up this evening, viz., "The Determination of Carbon Dioxide in Mine Air," under the new Mining Regulations.

Since the new law came into force making it compulsory to keep periodical records of the amount of CO_2 present in the air at all working faces, I have been asked by some of our assaying members about a convenient method for determining this gas. One query was whether Orsat's apparatus could be used. Another: was Dr. Moir's method useful? A somewhat absurd story was also told me about a particular method which must be employed, and which can only be done at one of the public laboratories in town.

What I would like to propose is this. Would Dr. Moir be good enough to republish his original method (in the next *Journal* if possible), giving its limits of accuracy, and would the Council have it put on next month's agenda for discussion?

Personally, where a large number of samples have to be done for CO_2 only, I prefer Pettenkofer's classical method, but I would like to see the subject discussed, and if possible a standard method established for use throughout the Rand.

Mr. A. McA. Johnston (*Past-President*): This subject has come before us very prominently just at the present time owing to the new mining regulation, and I think some method ought to be brought forward which could be uniformly adopted on all mines. No doubt there is some diversity of opinion as regards which method should be used. I think that if the Council or a sub-committee of the Society were to bring forward a method which would give in detail a scheme for obtaining this CO_2 figure, it would be of great advantage not only to the mining Inspectors but to the mining houses and members of the Society. I do not think the method brought forward some years ago by Dr. Moir is quite accurate enough, and I believe the best way to open up the question would be for Mr. Toombs to bring forward a method and let us criticise it.

The Chairman: Some assayers along the Reef are of the opinion that in a large majority of cases such work cannot be suitably done in the ordinary mining office and under the ordinary conditions obtaining there. They think the samples should be sent to certain central laboratories or to analysts in town to be dealt with. We might have some expression of opinion on the subject.

Mr. Johnston: In my opinion, the average assayer on the Reef, when given a standard method, would be fully competent to carry on these tests. There is no extraordinary difficulty about the analysis.

Dr. J. Moir (*Past-President*) (*contributed*): As requested by Mr. Toombs and others, I beg to submit a revised edition of the test which I published in 1906.*

Shaking Bottle Method for Testing whether Mine air contains under 0.20% Carbonic Acid Gas.

1. *Stock solutions.*—These must be made with recently-boiled good quality distilled water, and kept in yellow bottles with stoppers vaselined to exclude the air.

A. *Sodium Carbonate.*—Ignite "chemically-pure" Na_2CO_3 to expel the moisture which it contains: weigh out 2·120 gm. and make up to 200 cc. with the special distilled water, thus giving a solution of *M*/10 or *N*/5 strength.

B. *Alcoholic phenolphthalein.*—Dissolve 2·120 gm. pure phenolphthalein in 200 cc absolute alcohol (rectified spirit will not do). This gives a solution of *M*/30 strength.

From these the "working-solution" is made *just before use*, as the latter keeps only half a day.

C. *Working solution.*—Into a 200 cc. bottle put 2 cc. stock solution A, then 6 cc. stock solution B, then 192 cc of the special distilled water. Use narrow pipettes and do not blow into them. After shaking, the pink mixture obtained is a $\frac{M}{1000}$ solution of Na_2CO_3 and phenolphthalein or more strictly, $\frac{M}{1000}$ solution of $NaHCO_3 + \frac{M}{1000}$ solution of sodium-phenolphthalein $C_{20}H_{13}O_4Na$, the latter compound being the active ingredient.

MAKING THE TEST.

Sample the air at the place required by blowing into a clean 200 cc. bottle with a bellows. Ten puffs of the bellows are sufficient. The nozzle must be narrow enough to enter the sample bottle: use a rubber tube on the bellows if necessary to achieve this end. The operator must keep his face away from the entrance valve of the bellows or hold his breath while sampling: this is vitally necessary.

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* See this *Journal*, vol. vii., p. 66.

Next replace stopper of bottle. Have a measure-glass ready holding 12·9 cc. (delivering volume) when filled to the top. Fill it with the pink working solution and pour off the "dose" (12·9 cc.) into the test bottle containing the air. Replace the stopper and shake uninterruptedly for at least two minutes. If the pink changes to yellow or colourless, the air is "bad" in terms of the Regulation. If the pink only fades slightly, the air is good. If the pink nearly disappears the CO_2 is about 0·20% in the air. Do not expose the main quantity of the working solution to the mine air more than is necessary.

THEORETICAL BASIS OF TEST.

1. The test depends on the fact that carbonic acid is a stronger acid than phenolphthalein is, and therefore expels it from its (pink) sodium salt, forming sodium bicarbonate, which is *not* alkaline to phenolphthalein:—thus

$$C_{20}H_{13}O_4Na + CO_2 + H_2O = \underset{\text{colourless}}{C_{20}H_{14}O_4} + NaHCO_3$$

2. The working solution is $\frac{M}{1000}$ in sodium-phenolphthalein: hence 1,000,000 cc = 106 gm. Na_2CO_3 = 44 gm. CO_2 = 22·36 litres CO_2 at NTP. To correct this to mine conditions, assume 2,500 ft. depth (where $\frac{273+T}{B-f}$ is $\frac{297}{668} = \cdot 445$), hence 1,000 cc. reagent $= 22{\cdot}36 \times \frac{760}{273} \times \cdot 445 =$ 27·60 cc CO_2 gas at 2,500 ft. depth.

3. Now the test bottle, with reagent in, holds about 187 cc. air, which, at 0·20% CO_2 means 0·374 cc CO_2 to be tested for. Hence the volume of reagent required for a test is $\frac{374}{27{\cdot}6} = 13{\cdot}55$ cc. This is the theoretical quantity calculated from the equation, but as the reaction is slow and somewhat (about 5%) incomplete, a deduction of 4% or 5% should be made.

4. Hence 12·9 cc. is the correct practical quantity for a depth of 2,500 ft.: near the surface about 11·8 cc. should be used, and at 5,000 ft. depth about 14·0 cc. and so proportionately, although this correction is scarcely worth making as the method is not reliable to 0·01% of CO_2.

5. The following table indicates how the method can be made into a rough analytical method, trustworthy to about 0·02% of CO_2:—

(1) Colour gone in 1 minute CO_2 over 0·30%
(2) Colour ,, $1\frac{1}{2}$ minutes ,, 0.25%
(3) Colour ,, 2 mins. *or over* ,, 0·20%
(4) If the colour does not go, and x is the estimated percentage of colour left, $CO_2 = 0{\cdot}20 - \frac{x}{600}$: thus half colour left means $CO_2 = 0{\cdot}12\%$ in the air.

NOTES:—(1) This description differs from the earlier one by the introduction of two fresh corrections, *viz.*, moisture in air, and incompleteness of reaction.

(2) I am authorised by the Government Mining Engineer to state that the interpretation of Regulation 59 (periodical air-tests) is:—that for the present, records must be made by any chemical method that will distinguish between, say 0·20 and 0·21% of CO_2, but that no particular method is laid down as a hard and fast rule.

As regards the shaking method, I might claim an accuracy of 0·01% when it is done with special care; but if the work of air testing is to be done by people unaccustomed to it, I should prefer to recommend the methods which depend on absorption in barium hydrate and back-titration with oxalic acid—always assuming that the operator is competent to perform the calculations.

SOME OBSERVATIONS ON ANCIENT MINE WORKINGS IN THE TRANSVAAL.

By T. G. Trevor, A.R.S.M., Acting Inspector of Mines, Pretoria (Member).

Though the ancient mine workings of Rhodesia have attracted world-wide attention, the fact that similar workings extend into the Transvaal is not usually realized. It is the object of the writer to call attention to such of them as he has examined in the hope that this paper may act as a nucleus around which the information collected by other observers may gather, until sufficient evidence is collected to form a fair basis from which to reason as to the origin and date of the workings and the race of the workers.

Ancient as against Native Workings.—The modern Bantu tribes of the Transvaal undoubtedly worked copper and iron mines and possibly gold also, but there is no evidence that they had any idea of the existence of tin.

The amount of copper smelted by the Bantu was probably extremely small. In twenty years the writer doubts if he has come across fifty pieces of native smelted copper either in the raw state or made up into bangles, and a close inquiry from many Native Commissioners shews that their experience has been the same, and it is improbable that a ton of native smelted copper could now be obtained in the whole of South Africa.

Iron, of course, unlike copper, becomes quickly rusted and worn out and the quantity of native smelted iron now in evidence is no guide to the quantity extracted by the Bantu since they first settled in the Country. Still, the actual demand for iron ore for the manufacture of assegais, hoes and pigments among the Bantu could not have been very large.

In differentiating between "Ancient Workings" and comparatively modern native workings, the writer has treated all *small workings* either for copper or iron as presumably modern, and all workings for gold and tin and all copper and iron workings of *great extent* as presumably ancient in origin, even though it is known that in some cases the present natives continued to obtain small quantities of ore from them.

Districts in which Ancient Workings occur.—Ancient workings extend all through the Northern Districts of the Transvaal down to the northern edge of the High Veld—say the latitude of Pretoria. South of that latitude the writer has never encountered any. They are extremely plentiful in the Zoutpansberg-Waterberg-North Rustenburg and North Middelburg Districts and to a lesser extent in Lydenburg and Carolina.

They are also reported from the Marico District, but in that locality dolomite caves and fissures have undoubtedly given rise to many false reports and such workings as the writer has examined were, in his opinion, probably of recent Bantu origin.

Minerals worked for.—The minerals worked for were copper, tin, gold, haematite and micaceous iron, but the noticeable peculiarities about all those workings which the writer classifies as ancient, are the same whatever the metal mined for, so that before describing each group of mines these general peculiarities may be stated.

General Description.—The present appearance of ancient workings is nearly always a line of clumps of bush and timber growing far denser than on the surrounding country. When one penetrates these clumps one finds them growing in a series of artificial depressions up to 40 ft. deep. Each depression is surrounded by a very small dump or embankment of a size quite inadequate to represent the waste that must have been excavated. When sinking to prospect in the depression one invariably finds that the original working has not fallen in or been washed full of debris but has been deliberately filled in with material from the dump, and in some cases—which will be mentioned later—this has been done so thoroughly that no idea of the presence of ancient workings was entertained till they were struck in a shaft from 40 ft. to 80 ft. below the surface. In this case the filling in must have been done to disguise the existence of the mine, but in others, where it was not carried to this point, it may have been done from superstition or to prevent others from obtaining ready access to the mine. As in two instances known to the writer the filling rested on a stull it was probably done by the original workers and not by subsequent inhabitants.

In no case can any roads, ruins or other traces of the workers be found, nor is there ever any sign of anything in the nature of a tailing heap or important slag heap. At Messina, Palabora and most of the ancient copper mines very small slag heaps, containing only a few tons in the aggregate, can be found at the base of the kopjes in the neighbourhood, but these bear no more proportion to the amount of ore that has obviously been mined, than the slag from the assay office would to the output of a modern copper mine, and as the foundations and twyers of native forges and the foundation rings of native huts are found with these slag heaps, the writer attributes these heaps to modern native workers.

At Rooiberg some distance from the mine there is a hill now known as "Smelting Kopje" on which there is a small accumulation of tin slag, but no signs of any ancient habitations remain. An ingot or furnace bottom of tin weighing some pounds was also found on this hill. No other smelting places have yet been found in the Rooiberg Tin Fields and the amount of slag on the hill mentioned is quite disproportionate to the quantity and size of the workings in the neighbourhood. In all the ancient workings many hand hammer stones are found. These are usually hard waterworn stones which have evidently been used for striking metal gads as they are worn out in cup shaped holes at the striking points. No attempt ever seems to have been made to have mounted them with a handle. Large stone mullers up to 100 lb. in weight have also been found. These were probably used for roughly grinding or breaking up the ore broken from the face with the gads.

In the Rooiberg Mine some metal gads were found ; these are of iron about 18 in. long and of rough dumb-bell section made in one continuous taper from the head to a fine point. The following analysis of the metal of these gads, for which the writer is indebted to the Manager of the Rooiberg Mine, shews this metal to be of extremely good quality and exactly suited to the purpose for which it was used.

Silicon	0·17%
Sulphur... ...	0·01%
Phosphorus ...	– Trace
Manganese ...	– Nil
Combined Carbon	0·35 (approximately).

Bushman Tikoes or perforated stones are frequently shown as hammers, but these are picked up on the surface all over Africa and have nothing to do with the ancient workings.

One of the most peculiar points connected with these workings is the extreme cleanness of the dumps and the difficulty of determining till the holes are thoroughly opened what was the metal mined for. In the dumps at Messina, Palabora and other ancient copper mines it is almost impossible to discover a fragment of ore and often quite a time has to be spent in finding even a stone stained by copper. At Rooiberg so little tin has been left in the dumps that the workings were for years supposed to have been made for iron. Where gold has been worked the same thing is noticeable, it is usually impossible to get a colour out of any rock left behind at or near the surface

There are still many groups of ancient workings which have never been satisfactorily opened, and it is impossible at present to tell for certain for what they have been worked.

Just as little or no ore was left in the dumps, so also was every fragment of ore taken out of the reefs down to the depth at which the working was abandoned, and where there are ancient workings it is quite useless now to look for anything but barren reef left in situ on the surface.

In Rhodesia, where great experience has been gained in reopening ancient workings, these are seldom now reopened without satisfactory results being obtained, but Transvaal engineers and prospectors have not yet studied the question and in nine cases out of ten work is abandoned before anything is proved or any information is obtained. In the case of Messina Mine more than two years work was done before the real run of the deposits was discovered, and in the meantime one large Rand Company had thrown up their interest in disgust and it was only the faith of Mr. Grenfell, who had had experience of ancient workings in Rhodesia, that saved the property from abandonment. As it is, on some farms, where there are as big old workings as at Messina, thousands of pounds have been wasted without any results because the prospectors worked without considering what the proposition in front of them was.

Consider for a moment the outcrop of a reef at any angle worked as an opencast down to 70 or 80 ft., then stoped irregularly for another 30 ft. or 40 ft. wherever the ore was payable, leaving all the barren reef and rock in situ. This being done, imagine the whole filled up to the surface with debris and allowed to settle for years. The problem then is how to reopen that working in the cheapest way in order to find the original pay shoot worked. The usual method of the Transvaal prospector, when faced by this problem, has been surface cross trenching across the line of old workings somewhere between two of them where the ground appears undisturbed. If he finds any reef of course it is barren, or nearly so, or the old workers would have removed it. Yet many properties have been condemned on this evidence alone. Another plan has been to sink in the old working till the "made" ground was bottomed, and then to condemn the property if nothing was struck. In this case, the great probabilities are that the shaft goes into the footwall from whence the reef has been removed. Even if the reef is struck it is very likely in a barren or poor spot where the old workers have left it as unprofitable. The only satisfactory way to reopen these workings is to bottom the "made" ground, then to follow the footwall up to the reef or fissure, or failing that, the lowest point of the "made" ground, and then to drive along the ancient face till the whole lowest face is exposed and the shoot found. To sink outside the working and to cross cut under it is not altogether satisfactory as the shoot worked may have trended off to one side and the cross-cut may pass through entirely barren ground.

Description of Particular Workings. — In grouping the ancient workings for descriptive purposes it will probably be better to classify them according to the metal worked so that some idea of the output of that metal may be formed.

Gold Workings.—The only undoubted workings for gold observed by the writer, are at Honingkloof No. 112 in Mapoch's Gronden and at Weltevreden No. 215 near Waterval Onder.

At Honingkloof the outcrop of a quartz reef had been taken out for several hundred yards in one continuous trench ; the trench had been filled in again so that it only appeared as a hollow a few feet deep. When this was opened it was found however that the workings extended for more than 50 feet downwards. The reef proved to be one of white quartz showing visible gold in places, but an attempt to run a five stamp mill on it failed as the values were so erratic. It is probable, however, that the outcrop worked by the ancients was much richer as some small but very rich fragments of float were found in the neighbourhood. From this working one might fairly presume that some two to three thousand tons of gold bearing quartz had been removed.

At Weltevreden No. 215, sections of an outcrop of a huge vertical reef have been trenched out. In the quartz rubble under the grass roots alongside these trenches there are quite good values in gold, both free and entangled in quartz, but in the actual reef the writer failed to find even a colour. The trenches extend for about 1,000 yards along the outcrop and they have been filled to within a couple of feet of the surface. Some 20 years ago an attempt was made to work the property and the old workings were found to extend to some 40 ft. in depth, but the gold worked must have petered out at that depth for the reef where left by the ancients was either entirely barren or extremely poor. Some recent prospecting has also been done but though good values have been found in the rubble nothing of importance was struck in the reef.

Pilgrims Rest.—The writer is informed by Mr. W. H. Struben that there were large old workings in the Theta Reef when this was first worked—from 1870-1880—and that he himself bought the farm on which the Clewer Mine now stands purely on the strength of these workings. Modern work has, however, completely obliterated these ancient workings.

Complex Gold Ores.—In the North Middelburg District in the area enclosed by the Kruis River, Selous River, Oliphants River and Bloed River there are a very large number of ancient reef workings where reefs seem to have been worked out in long underhand stopes. These are, however, filled in to within a few feet of the surface. Fragments of ore which the writer has obtained from the dumps are in all cases complex—containing copper, zinc, lead, gold, etc. Personally, the writer has never had an opportunity of seeing any of these workings opened, but a Mr. Pullen states that in one case he sank 70 ft. in "made ground" without getting to the bottom. So far as the writer knows none of these workings have yet been properly opened.

At Elandsfontein, No. 167, north of Pilgrims Rest, there are some considerable ancient workings on a reef carrying both gold and copper, but the writer's recollection of these is not sufficiently clear to warrant a description.

At Wagon Drift No. 765, on Koedoes River, in the Zoutpansberg District, is a very considerable single working on a reef carrying some 5% copper and a few dwt. of free gold. The reef dips at a low angle, say 20°. For some 200 yards along the outcrop this reef has been completely quarried out to a vertical depth of some 50 ft., the overburden having been thrown back in rough layers of fine and lumps, a prospecting shaft sunk through it having the appearance of passing through stratified material. Below the depth to which this deposit has been opencasted irregular drives and headings have been carried in for many feet, but on the surface there is no indication of this, the surface appearance of the workings being merely an area of slight depression some one or two acres in extent carrying a different vegetation to the surrounding country. No dumps or slag heaps of any sort are known in the neighbourhood though several thousand tons of ore must have been taken out of the workings.

Copper Workings.—Copper workings exist from one end of the Northern Transvaal to the other, in fact wherever copper has been found (with the one exception of the Motali Copper Fields), it seems to have been worked either by the ancients or by the modern Bantu. The chief workings are, however, in the Messina and Palabora areas of the Zoutpansberg.

Messina.—In this area there are at least five or six very large groups of ancient workings. That at the Messina Mine which has been so successfully opened is, the writer thinks, the largest but the others are not very much smaller. The workings here are all filled up to within a few feet of the surface and show as lines of cup-like hollows of all sizes and shapes extending irregularly for about a mile in length and often in three parallel lines. In the aggregate there are some 120 of these holes, and development has gone to indicate that each one was on a lens of copper glance or bornite. One of the main holes has been emptied of the debris that filled it, and it now shows as a chamber some 80 ft. deep by 50 ft. in length and up to 30 ft. in width. The bottom of this chamber was still in solid unworked glance. Judging by the size of the chamber and the experience gained of these lenses in other parts of the mine this excavation alone must have given its workers well over one thousand tons of copper and possibly very much more. It is, of course, quite impossible to estimate the amount of copper the ancient workings at the Messina Mine must have produced, but it certainly could not have been less than several thousand tons and if the other workings in the locality yielded proportionately, then the total output from the ancient workings in the whole vicinity must have mounted well into the tens of thousands of tons. That quantities of this magnitude could have been used by the local natives for their petty needs, even in many generations, is unthinkable, even if we neglect the dozens of small copper mines all over the country which must have produced ample for their requirements. That the natives actually did smelt copper here in small quantities is an historical fact and some of their ingots are still in existence. These ingots are shaped like a minature top hat with the crown decorated with nipples and weigh about 2½ lbs. This shape of ingot is peculiar to the Baquena tribes and was the shape in general use in the Transvaal. The ingots smelted at Palabora by the Bapidi are of the same weight but entirely different in shape, being rods about 2 ft. long with a knob like a flattened pipe bowl on the one end, the rod leaving the bowl at the bottom as a tobacco pipe stem does. The knob or bowl is in this case decorated with two or more nipples These nipples are, however, in the writer's opinion, merely varieties of breast mark placed on so many articles of native workmanship to indicate fertility.

Palabora.—The name Palabora is used for the general locality lying to the north of the junction of the Selati and Oliphants Rivers. There are many ancient workings in this locality, but the chief of them lie on and around a small hill known as "Lulu." These workings have never been re-opened (June, 1911), and with the exception of one or two abortive prospecting shafts they are still as they were when first discovered by white men. The workings cover many acres, some of which have been as completely turned over as the ground of any abandoned alluvial gold field.

For the purposes of description it is useful to divide the workings here into three areas:—

"A." Western Portion.—This consists of the western slope and summit of "Lulu," which is covered with one mass of pot hole excavations with debris heaps between them. So close together are these excavations that the whole appears like an old alluvial gold field where all the ground has been turned over. The debris heaps are covered with grass and have worked down into smooth hummocks, no rock and few stones being visible. If these workings were alone it would be difficult to tell what they were worked for or what rock they were in, so much has everything weathered down.

"B." Central Portion.—This is immediately to the east of the western portion and covers most of the top, south and east sides of the hill. This is also covered by one network of pot hole workings, some of very large size—some of the smaller are still open for a few feet below the surface and the country rock is exposed and the debris heaps, though overgrown with bush, are still undecomposed stone. Some of the larger pot holes must have represented very considerable open-casts as they are still 40 ft. or 50 ft. deep and 100 ft. or more in diameter.

"C."—The third area consists of an entirely dissimilar series of workings on the flat at or along the east foot of "Lulu." Here we have a series of reef workings in which the reef has been stoped out neatly and in a most workmanlike manner, neat pillars having been left to support the sides. Such thick bush covers the whole area that it is almost impossible to form a general idea of the workings, but they appear to extend for quite one mile along three parallel reefs. There are practically no dumps at the side of these reef workings, and though one can still get down some 30 ft. or 40 ft. into the stopes it is obvious that they were very much deeper but have been filled in. No copper ore and very few malachite stains can be found in any of the dumps or workings, and samples taken out of the pillars that remain *in situ* show very little copper. The assay of certain samples which the writer took from the pillars, gave only 2% copper. The reef material is, however, completely oxidised, and if the workings are really of very great age, contemporaneous in fact with the gold workings of Rhodesia, as the author thinks probable, it may well be that the copper has been leached out of the ore. Until the bush has been cleared from the whole of the Lulu Kop workings it will be very difficult to form a true idea of their magnitude, but the writer believes that in the aggregate, as in the individual workings, they are far larger than those at Messina. At many other places in the Low Country there are large groups of ancient workings quite comparable with those at Messina, but so far none of these have been satisfactorily opened. Even if, however, only 1% of copper was extracted from the amount of ground shifted, then many thousands of tons of metal must have been obtained.

Tin Workings.—The modern Rooiberg Tin Mines were discovered by re-opening a group of ancient workings. These workings appeared as insignificant 'pan'-like hollows covering quite a large area. In the aggregate it can easily be that 100,000 tons of rock have been turned over here. As the poorest tin ore found in these workings has not less than 20% tin, quite 1,000 tons of tin must have been derived from them. The workings appear to have been on more or less horizontal flats or shoots and not to have extended to any great depth—40 ft. or 50 ft. probably being the maximum.

Weynek Tin Mines.—At Weynek a reef inclined at an angle of about 30° has been mined out by the ancients down to some 200 ft. from the outcrop, the outcrop itself having been first opencasted to a vertical depth of, say, 20 ft. Such pillars and fragments of ore as were left near the surface are quite rich in tin, and the amount of tin extracted may well have been several hundreds of tons.

Leeuwpoort Tin Mine.—On this farm, as on nearly all of the farms in the neighbourhood, there are many small groups of ancient workings, but none of these have been successfully opened yet. On the hill behind the camp, however, where no trace of an ancient working can be seen on the surface, a shaft of 40 ft. broke into some large excavated chambers; some of these were empty, but some had been filled in. The excavations had evidently followed the run of certain very rich pipes or shoots of tin ore—what remained of the ore on the sides of the working was nearly pure cassiterite. Most of the filling had been put in artificially and wherever the opening from the surface is, it is certainly so well disguised as to be quite undetectable. It is impossible to estimate what amount of tin these Leeuwpoort workings might have accounted for, but if we say that the whole of these tin mines in the Rooiberg area could not have accounted for less than 2,000 or 3,000 tons of tin we are not, it is thought, overstating the case. The only known sign of ancient occupation other than the actual mines, are the slag heaps at Rooiberg on the hill known as "Smelting Kopie" alluded to previously.

Mr. Schoch, the Manager of the Rooiberg mine, has kindly supplied the following analysis of a piece of bronze obtained from a small slag heap, on the farm Blaaubank near the Rooiberg mine :

Ancient Workings at Palabora.

Copper, about 80% ; Iron and alumina, about 5% ; Tin, about 7% ; Nickel, about 3% ; Arsenic, about 2% ; Gangue, about 3%.

The piece is about $\frac{1}{4}$ to $\frac{1}{2}$ lb. in weight and is unformed, appearing to be more like the overflow of a crucible than anything else. The ingot or furnace bottom of tin now in the Geological Survey Museum, Pretoria, was also discovered in this vicinity.

Concerning the bronze, there is no known ore in the Transvaal that by simple smelting would yield such an alloy as this, nor is there any copper mine known in the immediate Rooiberg district. As the composition is that of an excellent bronze it is impossible to doubt that the alloy was intentionally made of a definite composition and was not an accidental production. It is perhaps unnecessary to remark that there is no evidence of any native race south of Benin and the Gold Coast being acquainted with either tin or bronze.

Iron Workings.—Modern native workings for smelting iron ores and for pigments are all well known to living natives and are easily distinguishable. There is however another class of workings which are frequently of enormous size and easily discernable from the modern Bantu excavations. These have apparently been made either for micaceous iron ore of a specially flaky quality or for red haematite, not necessarily very rich in metallic iron but worked apparently for its value as a pigment. Both these materials are used and have been worked to a certain extent by natives, but the present natives have usually no knowledge of these large workings, and the style in which they are executed is not that of the modern Bantu.

The whole Waterberg, Zoutpansberg and north Middelburg districts are full of workings which, in the absence of other evidence, we must assume to have been made for a particular variety of micaceous iron, though in some only an extremely small percentage of this mineral is present. The following is a description of two typical workings of this class:—

Workings on the Farms "Palmietvlei" No. 1,868 and "Keerom" No. 1,869, Zoutpansberg.—These ancient workings consist of one long shallow trench where an outcrop has evidently been worked out. As seen at present this trench is from 6 ft. to 10 ft. deep and up to 20 yards wide; it is absolutely continuous for about four miles and recurs again at intervals on the same geological horizon for many miles: in fact the great excavations on the town lands of Potgietersrust are on identically the same beds—

Ancient Workings at Palabora.

though these beds are not continuous the whole way, having vanished under the granite for several miles. There are occasional hollows in the bed of the trench, but usually it is continuous and resembles an old canal bed or railway cutting. In one portion the trench is duplicated for about a mile by a second one parallel to it. The trench runs east and west and its north side is usually a wall of quartzite up to 20 ft. high dipping at an angle of about 60° south into the trench :—the south side is a debris heap, but the amount of debris is extremely small and even that is entirely overgrown. The hollow of the trench in cross section is even, the bottom being approximately flat and there is no appearance of it having been filled with debris washed down the hill. On the other hand the appearance would lead one to suppose that the material from the dumps had been thrown back by hand. Here and there along the trench are spots filled with large loose stones which give the impression of being winzes or shafts which have been filled up by hand. At one spot there has recently been a subsidence and a hole about 12 ft. square and 15 ft. deep has sunk down in the middle of the trench. It has gone down in one unbroken mass showing that there must have been a very considerable open space below. The sides of the hole caused by the subsidence are in entirely "made" ground. In another place a pillar seems to have been left across the working and on the west side of this pillar a hole has fallen in to what appears to have been a stope. If the hanging wall of a steeply inclined stope caves in, an open space will still be left immediately under the upper face of the stope ; this hole was just such an open space and the worked out ground below it may have been of any depth. The writer got down 30 ft. or 40 ft. into this hole and saw the following section of the pillar :—

"*A*." Massive bed of sericitic shale containing some layers in which there is a considerable quantity, say, 15 of very flaky micaceous iron. This bed is 2 ft. 6 in. thick, and as far as one could get in it was covered with gad marks.

"*B*." A bed exactly similar to "A," but very shaly in structure and showing no gad marks.

"*C*." Two beds, each 2 ft. thick, of a massive ferruginous quartzite, the ferruginous matter being specularite. These beds are much richer in iron than the others, possibly containing 30% or so over all, but they do not appear to have been touched and outcrop at intervals all along the old workings.

Evidently the bed "A" was the one worked and contained the mineral sought for ; the bed was sampled carefully by the writer, but only micaceous iron was found in it and that only in quantities altogether insufficient for the rock ever to have been used as an iron ore. The bed "C" moreover contains far more micaceous iron than the others and is untouched. Two theories to account for this are possible :

1. That micaceous iron was not the mineral worked, but merely an associated mineral, and that the pillar mentioned was left because it was barren of the ore really mined.
2. That micaceous iron was really the mineral mined, but that the quality in bed "A" was either superior in flexibility, lustre, or size of flake to that in the other beds, and was therefore more valuable to the workers.

In the writer's opinion this question can not be settled without a thorough exploitation of the excavations. In the similar ancient workings at Potgietersrust the same bed of rich iron ore has been left standing where an underlying bed has been worked, but at that place no section can be obtained.

The Chief of the local tribe informed the writer that the tribe knew nothing of the origin of these workings, but that their folk found them there when they came to the country in the early years of the last century. He says that his predecessor did have some shafts which were open filled in and that to hide them from the white man he planted the whole thing full of prickly pears, but that his tribe never got anything out of the holes and have no idea as to what was obtained from them.

Working on the Farm "Doornrivier" No. 923, Zoutpansberg.—The country rock is old granite, the ridge in which the workings are situate is a quartzose segregation (pegmatite) in that granite. The main working is a square hole 35 yards long by 12 yards wide carried down in the solid rock with vertical wall sides to a present depth of 30 ft. to 40 ft. The bottom is all debris and as the rock rises up to within a couple feet of the surface and as there is little or no soil or debris over it, it looks as though all the dump had been thrown back into the hole. The west wall of the hole is the hanging wall of the big vertical quartz reef which forms the axis of the ridge. The faces in the north and south ends of the hole are still quite open and can be sampled, the rock is red coloured and silicious, very much fractured and seamed with a network of small secondary quartz veins. A little micaceous iron, certainly not more than 1%, is visible in some of the seams and in the rock, but is nowhere concentrated. The walls were thoroughly sampled but nothing was found to indicate what mineral was worked save a little micaceous iron.

Ancient Working on Lot No. 76, De Kaap, near Malelane.—This has been worked by modern natives for pigment, but the workings are so extensive that it is possible and even probable that they were started long before the era of the modern Bantu.

A bed of banded haematite makes a bold bluff on the east side of a kloof rising up at a very steep angle to a height of 700 ft. to 800 ft. The outcrop of the bed appears to be from 70 yards to 100 yards in width; the whole of this width from the bottom to the top of the bluff is covered with workings. Most of these are open quarries, but there are many irregular shafts and large cave-like openings which have been partly filled in. To get an idea of the size of these, the writer measured up one of the quarries and found that upwards of 15,000 tons must have been removed, and that quarry certainly did not represent one-tenth part of the total working on the hill.

Conclusions.—From the foregoing description it is obvious that the ancient workings of the Transvaal are important enough to deserve considerable attention. If it is said that they were made by natives then we must presuppose an entirely different native to the one we know, a native who knew the metallurgy of tin and bronze and who dealt in copper by the hundreds of tons. If on the other hand we consider, as the writer does, that these workings must be referred to an era long prior to the Bantu, then we are forced to the conclusion that the ancient miners of Rhodesia and the Transvaal were one and the same people. Who these people were it is yet too early to determine, but evidence is constantly being accumulated in Rhodesia to prove that they had some connection with Southern Arabia and that at a date long prior to the Christian era.

To enter into a discussion on this point is not now the object of the writer, who has merely wished to put these observations on record in order to aid in the accumulation of evidence which may some day settle the question.

The Chairman: We are much indebted to Mr. Trevor for his excellent paper; it is one of the most interesting we have had for some time. We have a large agenda, however, and discussion on it will have to be postponed until a future meeting.

NOTES ON HIGH DUTY GRAVITY STAMP MILLS.

(*Read at October Meeting.* 1911.)

By Peter N. Nissen (Member).

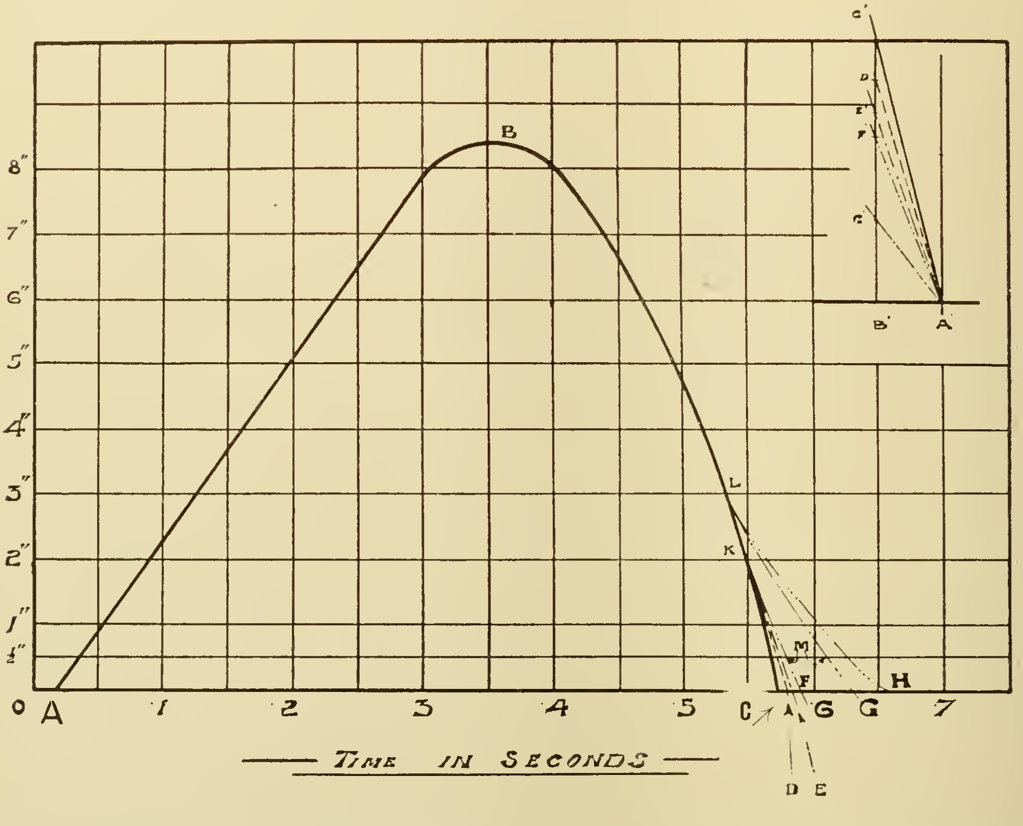
DISCUSSION.

Mr. W. Mackesy (*Member*): In commenting on the Nissen battery and its achievements, I should like first to say a few words about the stamp motion diagram. (Fig. 1.) The curve A, B, C in the figure is taken from T. K. Rose's Metallurgy of Gold. A, B is the lift and B, C is the drop. At the point C the stamp has come to rest upon half-an-inch of rock. At some point K the stamp begins to smash the rock upon the dies. At first the rock is loosely distributed, but as the stamp forces its way through the loose layer it forms a compact mass with enough resistant power to bring the stamp to a sudden stop. The first state of the rock might be roughly compared to screened coarse gravel, and the final state to sand and fine gravel, all interstices in the gravel being filled with sand. Such a layer would have large resisting power, and be able to bring the stamp to a sudden stop.

The first part of the crushing may be called "free" crushing, and the crushing done on the compact layer "choke" crushing, terms used by Prof. R. H. Richards in discussing the action of rolls. We would naturally expect a coarser product from "free" crushing. The slope of the curve B, L, K, C, that is the tangent of the angle which the curve at any point makes with the horizontal, represents the velocity at that point. Thus, as in the small figure, we can deduce the amount of work stored in the stamp at any instant. Now, at the point K the stamp begins to crush rock, and from K to C the crushing is "free" crushing. In doing this work the stamp should be losing velocity, and the curve should be getting flatter after the style of L, G. But, as B, L, K, C is an undisturbed parabola, we infer that the stamp is doing no work from K to C; and its efficiency as far as "free" crushing is concerned is zero. However it may be doing work in "choke" crushing.

Now I have set off K, D a curve representing the stamp having parted with 24% of its energy in "free" crushing. So at all events its efficiency is 24%. Further, suppose that instead of a 9 in. stamp of 63 sq. in. area, we try a $12\frac{3}{4}$ in. stamp of 126 sq. in., or just double the area. Allow that on the $12\frac{3}{4}$ in. die there is a layer of rock similar to that on the 9 in. die. The $12\frac{3}{4}$ stamp meets this at the same point K, crashes through it almost the same distance to E; thus having smashed through twice the area of a similar layer of rock for the same distance, the $12\frac{3}{4}$ stamp will have done double the amount of work that the 9 in. stamp did. Therefore, its efficiency in "free" crushing will be 48% instead of 24%, and to this I have set off the curve K, E. Now suppose on the $12\frac{3}{4}$ die we somehow managed to get a feed layer of the same thickness but twice the density. Then our "free" crushing curve would be K, M, F. However, as the density of the layer on the dies is twice that in the case of the curve K, E, the compact resistant layer would be about twice the thickness. Thus our curve would end at M. However, here the curve represents an efficiency of about 60%. L, G is drawn to represent the effect of a further increase of stamp area, and an extremely coarse grade of feed. L, K is the ideal for "free" crushing, 100% efficiency, and a soft blow like a Blake crusher.

It is evident that, with every rise in free crushing efficiency, the stamp has a smaller velocity when suddenly brought to rest; that is, the blow is softer. And on the occasion when I had the opportunity of seeing the Nissen stamp working, nothing impressed me so favourably as the comparative softness of its blows. It seems to me that it owes its efficiency in part directly to the large diameter as illustrated in curve K, E, and also to the fact that the larger diameter, giving an increased discharging power, allows a denser layer of ore on the dies as illustrated in curve K, F. I would point out that the coarse grade of issuing pulp also indicates that this battery gains its advantage in the "free" crushing part of its work.



Now, returning to the stamp motion diagram we see that in all the cases C, D, E and F from one-half to the whole of the energy of the stamp is passed into the compact resistant layer. Some of it will be used in crushing work, but the majority will be passed on to the foundations. The thickness of this layer is controlled by the feed tappet and therefore easily altered. It is of interest to enquire what connection there is between its thickness and the amount of energy utilised. The energy in excess of what the layer can absorb and pass on, must be used in crushing. The amount of energy the layer can absorb is the product of half the maximum total pressure into the elastic deformation of the bed of ore

$$\text{Energy} = \tfrac{1}{2}\, P_{\text{max.}} \times E.$$

now $E = P_{\text{max.}} \times \text{Thickness of bed} \times \text{Constant.}$

$$\text{Energy} = \tfrac{1}{2} P_{\text{max.}}^{\;2} \times \text{Thickness of bed} \times \text{Constant.}$$

as $P_{\text{max.}}$ is a more or less constant quantity.

$$\text{Energy absorbed} = \text{Constant} \times \text{Thickness of bed.}$$

This means that if we halve the thickness of the compact resistant bed we halve the amount of energy it can absorb; and so increase the amount used in crushing, that is in "choke crushing. Here then we have the explanation of the dreadful practice of cutting down feeds during the last week of the month when the crushing is behind.

Now it is clear that in cutting down the feeds we are reducing the amount of work done as "free" crushing. Thus it is evident that the 9 in.

stamp does the greatest part of its work in “ choke ” crushing, owing almost certainly to want of shoe area and lack of discharging power. Increased discharging power would admit of a deep feed layer on the dies. And this deep layer might easily gain more in “ free ” crushing work than it loses in “ choke ” crushing.

Having arrived at the conclusion that, in the present 9 in. battery, a thin resistant layer is desirable we can realise the loss of efficiency due to uneven wear of the dies. Mr. Nissen seems to think that even wear is a virtue peculiar to his type of battery. However, I do not look at it in that light. The new heavy multiple stamp batteries all have a water feed in the form of jets playing gently upon the face of each die as in Fig. II.

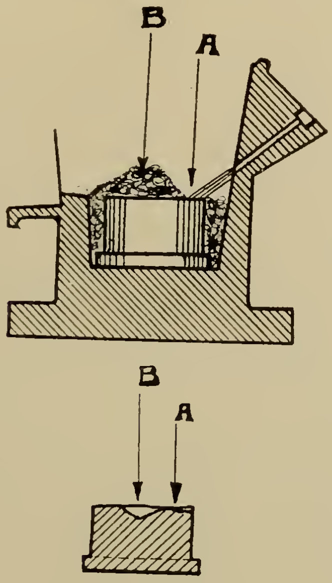


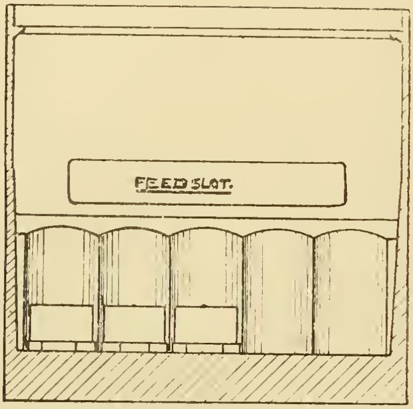
FIG. II.

Where the jet strikes at A it tends to wash the die bare and to make a little mound at B. Thus at the point B more work is done than at any other point on the die and more wear takes place. Thus we get a die worn into a hole at A and a hill at B, which is just the sort of thing that happens. Mr. Nissen's method of introducing the ore and water together down the chute is a good and old method ; and to it and a deeper layer of ore on his dies does he owe the better wear ; not, I think, to his one stamp design. There is more I should like to say upon this important point of obtaining even wear, but time forbids.

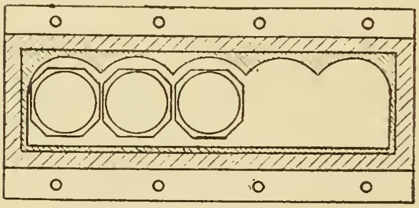
Now I come to what I regard as the weakest point in the Nissen stamp's action ; at each blow ore is broken and flung against the screen. The oversize has to find its way back on to the die in a very short space of time under the action of gravity alone. One can thus understand, as Mr. Nissen tells us, that very soon a limiting diameter of shoe is reached. In the multiple stamp battery, on the other hand, each stamp as it drops acts as a positive feeder to the adjoining stamps. This action will be fully realised if we consider the action of a battery with the out of date order of drop 3, 2, 4, 1, 5 ; the ore being fed at No. 3 the centre stamp. No. 3 drops, crushes and flings ore towards each side as well as discharging some. No. 2 drops, catches the wave of travelling ore, flings some of it back on to No. 3, which now has a covering of ore on it and some of it on the No. 1. No. 4 now drops flings ore on to No. 3, the centre stamp which now is fully covered, and some on to No. 5. Thus we see that No. 3 is now fully and positively covered with ore by the adjoining stamps after every blow. No. 1 and No. 5 now drop, but being end stamps they can clear themselves only in one direction. Thus the two end stamps receive an undue proportion of ore to crush ; No. 1 somehow, probably owing to the wave action, receiving more than No. 5. With the best order of drop this action occurs but to a less extent, and in most mills one can feel the distinctly softer blows of the two end stamps in any mortar box.

I fancy some millmen attach but little importance to the discharging power of a battery. However if this view was correct it would be a simple matter to increase the crushing power of any battery. The power a stamp is capable of giving out is the product of its weight into the number of feet fallen through per minute. As the time of lift is equal approximately to the time of fall it is easy to prove that the space fallen through per minute varies as the square root of the length of drop. By lengthening the drop from 8 in. to 12 in. and running at a correspondingly slower speed the stamp would fall through a 23% greater distance. Thus the power available in a 1650 lb. battery at 12 in. drop would be equal to that of a 2,000 lb. battery set to an 8 in. drop. Now, if up battery practice of 20 to 30 years ago is considered, it is found that long drop and slow speed was at first the standard practice. However, it was found by repeated tests that running with an 8 in. drop at 90 to 100 drops gave double the stamp duty. This was in spite of the fact that the available power was lessened in obtaining this increase. Not only did the increased speed give a greater number of individual splashes per minute, but at 90 to 100 drops there seems to be hit upon the periodic time of a series of waves that ripple from end to end of the box. These waves look, and are, far more efficient for keeping in suspension and discharging crushed particles than the individual splashes of a slower speed, which the splash of the Nissen stamp so much resembles.

There is another method by which increased discharging power has been given to the battery, that is by making the mortar box as narrow as possible and so reducing waste spaces. A narrow box confines and intensifies the splash, which, after all, is the only power which rids the box of crushed material. In connection with this a sketch (Fig. III.) of a mortar box liner which I designed some 18 months ago is given.



SECTIONAL ELEVATION.



SECTIONAL PLAN.

Proposed Mortar Box Liner.
Provisional Patent, Transvaal, 16-1-12.
FIG. III.

The idea is to form on the back of the box curved projections to partly fill up the waste spaces between adjacent stamps. By doing so an increased discharging power and efficiency should be obtained. Further, the projections should partly check the tendency of the rock to travel to the ends of the box and so allow more equal distribution of work among the stamps. If ever it is tried I shall await results with great interest.

In conclusion, although the writer believes that the multiple stamp battery can be made a better machine than the Nissen stamp, yet nothing can take from Mr. Nissen the credit of a real step in advance.

Mr. G. A. Robertson (*Member*): The author states that the lack of control over the feed to a five stamp mortar box is responsible for broken shoe necks, broken stems, worn guides, and uneven wear of dies. The writer's experience is that broken shoe necks were introduced with the heavy stamp. With light stamps of a running weight of 1,050 to 1,250 lbs. broken shoe necks were almost unknown. The introduction of stamps of a running weight of 1,650 lbs. led to the introduction of special steel shoes and the fact that guarantees given by different firms have not been fulfilled leads one to the conclusion that this question of shoe neck breakages has not been solved. A logical method of dealing with the problem is shown in the author's Fig. 1, namely, increased diameter of head and a thicker shoe shank. Stems break more often with heavy stamps than with light stamps and the Nissen stems 10 ft. long by 5 in. diameter would certainly run longer than the average 5 stamp construction. Provided, however, that the stems of the Benoni Consolidated mill which are about 7 ft. long and fit into the heads just below the cam shaft were not liable to be drawn out of the heads, that design is in my opinion an improvement on the long head of even diameter. The Benoni mill heads are tapered towards the stem and the heads run in the lower guides. If jar or vibration to the stem arising from impact of the shoe on the die is responsible for the stems breaking, it is a fact that the vibration on the stem decreases the further away the insertion of the stem in the head is from the shoe.

Individual iron guides, as advocated by the author, have many advantages over wooden ones. The metallic guide is quite a recent innovation on the Rand. Carrying stamp milling to a fine art, it is an impossibility to maintain true alignment of stems with the five hole wooden guide generally in use. With heavy stamps and high stamp duties, it has always been a puzzle to me to know why mill designers adhere to the system of 10 cams on one shaft. With light stamps a 5 stamp cam shaft runs longer than a 10 stamp cam shaft, besides possessing other advantages from the millman's point of view. When we come to heavy stamps with 10 cams on one shaft the breakages with medium priced cam shafts are enormous. With special material, however, for this type of shaft which costs £52 10s. each, good service is obtained. The Nissen cam shaft carrying 4 cams ought to last longer than the ordinary construction. Caps on the cam shaft bearings have as a rule been considered an unnecessary evil. The objections sometimes raised against the 5 stamp cam shaft is that one end of the shaft vibrates considerably in its bearing. That is true to a certain extent, but if a cap of good design is used the cam shaft can be kept wonderfully steady. The caps on the Nissen stamp and those in the Benoni mill only require to be seen to be appreciated.

The Nissen reinforced wood king-post seems an economical improvement over the heavy timber design. I am rather inclined to think however that with the increased vibration of heavy stamps and the possibility of a further increase in weight that wood will be superseded by steel or reinforced concrete. Tight joints are easily made with steel, but I have yet to visit the stamp mill made of wood where I could not detect heated joints. With a reinforced concrete structure there are no joints and consequently no beam joint troubles. I believe a mill in West Africa is now in course of construction, in which wood is being replaced by concrete. The large screen area provided in the Nissen stamp is greatly responsible for its high efficiency.

The generally accepted principle that the screen area with ordinary stamps is sufficient for the discharge of the pulp when crushed is not founded on scientific and experimentally ascertained facts. Dr. Caldecott in his paper "The Development of the Heavy Gravitation Stamp," quotes from C. De Kalb and points out that as 40% of the mortar box area is available for discharge it is ample for the small stream of pulp yielded by five stamps. H. W. Macfarren in his book "Practical Stamp Milling and Amalgamation" ridicules the idea of individual mortars and increased screen area. C. O. Schmitt proves to his own satisfaction that, with a heavy stamp, ample water supply and coarse screen, all the pulp is removed from the mortar as soon as the stamp has done its work. Mr. E. J. Laschinger says that with coarse crushing there is sufficient screen area with ordinary mortars. Some of these gentlemen were no doubt alluding to fine crushing stamps, but with coarse crushing and heavy stamps, and without going into details at present, the idea that the five stamp single discharge mortar provides sufficient discharge area is a fallacy which I hope is now exploded.

As regards the author's figures of capital expenditure for Nissen and ordinary stamps it will be noticed that as the author goes carefully into every item of the Nissen stamp construction, one could take it for granted that in the Nissen mill no construction whatever of an experimental nature would be put up and the finished mill would be in good running order and condition. Any millman who has had local experience of new mills knows that when the construction department is finished with their share of the work it does not follow that the mill is ready to run or is going to run well or that because all wearing faces are new and a mill is built in accordance with the needs of practical experience, milling costs ought to be the lowest for the first six months. The reverse is generally the case. The whole Nissen stamp construction is the outcome of close observation in practice. Experiments with stamps during the last five years have been numerous. These local experiments however have been conducted with the view of discovering something, whereas the Nissen stamp has to my knowledge been advertised in mining papers for the last five years. This only goes to show that in this field of specialisation mill designing has never been carried far enough.

NOTES ON ABSORPTION OF GOLD BY AMALGAMATED COPPER PLATES, CAST IRON RETORTS AND AMALGAM TRAYS.

(*Read at December Meeting, 1911.*)

By Prof. G. H. Stanley, A.R.S.M., F.I.C. and M. Thornton Murray, M.Sc.

DISCUSSION.

Mr. H. A. White (*Vice-President*): I heartily congratulate the authors upon this valuable and interesting paper and our Society upon the fact that we have strong hopes, warranted by previous papers from both the writers, that these "Notes" form part of a series of contributions to our knowledge of important technical details of gold metallurgy, especially valuable as coming from so well-equipped a source. To be candid, I must admit the feeling that professors who undertake no original research whatever are merely grinding dead men's bones to provide a daily bread for their students, and my welcome to such papers as this is all the more cordial from the proof they offer that our successors in the thorny path have living lights to guide them.

The authors object to the statement, almost invariably made when a new reduction plant is started, that gold is absorbed by the amalgamated plates. They prove that this statement is terminologically inexact and their proof is complete and sound. However there yet remains no doubt that this oft made statement, though inaccurate in wording, conveys to those for whom it is intended a substantial truth. Gold is necessarily "absorbed" in the *setting* of amalgamated plates and if the best work is to be done a considerable amount of such gold cannot be recovered till the plate is out of use. Something between 50 and 100 oz. of amalgam is retained against the daily scrape for each month's run before steaming. As this is usually done every three or four months, where conditions permit, a considerable "absorption" is thus accounted for. I admit that quite a number who should know better believed that a large amount of gold actually sank into the copper in the form of a solution in mercury, but this superstition will not, I trust, survive the reading of these Notes.

It may be interesting to briefly enumerate some of the sources of shortfall in the first month's output of a new modern plant. We may begin at the mortar box, where some little concentration takes place, then the launder to the elevating plant picks up a little gold in any small crevices that exist, the tailing wheel or pumps, especially the former, has been known to gather quite a rich coating, the tube mills have been known to yield as much as 500 oz. of gold upon removal of the lining when worn out and even silex liners can easily accumulate 200 or 300 oz. of amalgam. Beside the tube mill plates, already discussed, the succeeding launders and traps will not yield their full share to the first clean up. If in addition to these sources we consider that all the rock crushed cannot be completely treated in the time, that a fair amount of gold is in the working solutions, that a large amount of partly coated zinc must be left in the extractor boxes, that new furnaces and pots absorb some gold and that some valuable by-products formed are not immediately treated, it is clear that the usual statement to shareholders is considerably more than mere soothing-syrup. Indeed a considerable capital is necessarily locked up in this way, though this is seldom provided for in the estimates framed. I agree with the authors as to the probability of liquid amalgam being the means of conveying the gold into minute cracks in the retorts; where however suitable trays are used, no overflow can take place and in such cases the retort will not be, found to contain much gold, though the authors figures certainly warn us to cut out a piece for assay and not to rely upon appearance only. I would add, however, that at the Co-operative Smelting Works such bonanzas as referred to in this paper are not met with, and though gold content of the retorts bought is not taken into account, we certainly do not find any unmanageable surplus of gold in our outputs. In fact our manager after a good deal of experience in breaking up old retorts deprecates any enthusiasm about further supplies.

The fact that heat enormously increases the fluidity of most liquids has a considerable bearing upon this subject, and it is very instructive to examine a brick from the bottom of the ordinary pan furnace used in the lead smelting of zinc-gold slime. The minute cracks, visible only under the microscope, are yet filled with metallic lead, for though the pressure is small the lead was red hot with a corresponding decrease of surface tension and viscosity. A similar pressure of cold mercury would not so penetrate the brick if applied for years on end. It is a well known fact that a filter paper moistened with such a liquid as paraffin is impervious to water and if the authors could apply this principle and discover for us a liquid with which we might soak the bottom of a pan furnace, thus avoiding the infiltration of gold bearing lead and so evading a tiresome source of "absorption" of that precious metal, they would be performing a valuable service worthy of them and of the reputation this paper so ably sustains.

ACCIDENTS IN TRANSVAAL MINES.
(With Special Reference to Falls of Ground.)

(Read at November Meeting, 1911.)

By James Chilton (Member).

DISCUSSION

The President *(communicated)*: I read Mr. Chilton's paper with peculiar interest, as it has been my endeavour during my mining work to take care of those under me, whether white or black, and it has been my experience that if this spirit is manifested in the Heads, it is carried through the entire body of workmen in a mine.

Few realise what an accident costs, apart from loss of life or limb—the dislocation and disorganisation of work—the laying off of men through nerves caused through any accident—are often very serious, and I maintain that prevention is better than cure, and it is always better to be safe than sorry. It seems very strange, and yet it is none the less true (and it was brought home vividly to me by the late Mr. J. S. Fisher, Inspector of Mines), that, as a rule, the most dangerous mines were the safest, that is to say, the least number of accidents occurred on them, because supervision and care was regular and constant—not cursory and fitful, as on the "safe" mines. There is no doubt in my mind that if every one in a mine, from the manager to the miner, did his share in trying to prevent accidents by proper supervision, reporting and repairing, the accident rate would be minimised. But "*The*" great cause is the far too frequent change of men, from highest to lowest, in a mine. How can anyone take a particular interest in his section of work if his stay is to be but short!! When, in Europe lately, I visited numbers of mines both in England and on the Continent—some of them considered dangerous—but there, accidents were few, because their changes of personnel were few. It was most interesting, and to a Transvaal mining man most strange, to find from these old grey-headed employes that they had spent a life time on one property, and had no thought of leaving it except to retire. How different to South African experience!

Then again, the old experienced skilled miner has to a great extent gone, and in his place has grown up a rock breaker and native overseer, who does not know by instinct when to scent unsafe ground. The miner was careful and took a pride in his work. The present man takes chances, for he is here to-day and elsewhere to-morrow. In England the mine takes in boys who go from one step to another and learn all the necessary branches, as in every other trade. Here, a man gets a blasting certificate in six months, having perhaps been in one stope only, and poses as a skilled miner.

During vertical shaft sinking I had few accidents, and I put this down to getting good steady experienced men, whether miners, engine drivers or timbermen, and I kept them. Each man knew the unsafe points in the shafts and watched and examined them and took a pride in keeping them in order. Systematic cleaning of timbers, tightening of bolts and careful examination was done and the same care exercised as with a railroad

With pardonable pride I quote an extract from the Government Mining Engineer's Report, dated 30th June, 1910:—

> "Shaft sinking is one of the most dangerous branches of mining, and the death rate amongst shaft sinkers is usually very much greater than amongst trammers, stopers, developers, etc., and that three persons only should have been killed during the sinking of the South Rand shaft to its present depth of 2,254 feet forms a record. No doubt a certain margin must be allowed for the element of chance, yet I consider that the low death rate is to be attributed to the great consideration given by Mr. Saner and those associated with him in the sinking of the South Rand shaft and the Turf Mines shaft 4,100 ft. deep to the care of the lives of the men working in these shafts and the safety measures so rigidly enforced."

The enforcement of these safety measures was not at the expense of speed or cost, as these two pieces of work compare favourably in both respects with others on the Rand.

In the outside districts, where the mining is just as dangerous, but perhaps where there is not the same "hustle," the accident rate is very low. This is, I feel sure, due to the fact that there are few changes; and also that permanent white workmen lead to long service natives. Each knows his particular job and keeps his end up, the native having learnt to put trust in his boss. A steady stayer with ordinary ability is of much greater value to a mine than a brilliant man who comes and goes like a rocket. There is no better muscular machine than the South African native, and when once he is shown what to do and how to do it properly, it will be done day in and day out, and if a miner takes the trouble to instruct his gang in the perils of bad hanging and misfired holes, his accident list is low.

The author is to be congratulated on bringing forward this paper. I only hope that it will encourage other mining members to emulate him.

Mr. J. E. Vaughan, Acting Inspector of Mines, Natal (*Member*): Mr. Chilton's paper is welcome, as drawing attention to a subject which has not received the notice in the past that it deserves.

While agreeing with the author that the accident and death rates on the Rand are appalling, I do not consider it fair to make comparisons between the Rand and England. In England, underground workers are miners, in the Transvaal only about 3% of the persons employed below ground are skilled miners; the remainder are mainly ignorant savages.

In the English collieries, no person is allowed to work alone at the face until he has had two years' experience of such work under the supervision of skilled workmen; on the Rand, owing to the dearth of competent miners, one often finds a man with only six months' experience of underground work, not only working alone at the face, but having sole charge of from 30 to 40 natives. Again a comparison with England on the tonnage basis is misleading, in England the bulk of the tonnage is made up of coal, in the Transvaal of reef from the Rand mines; more labour is required to get a ton of reef than a ton of coal, and in consequence more accidents are likely to occur.

I have not been able to get recent literature on the subject, but a comparison of the death rate based on the tonnage crushed, would be interesting if made between the Transvaal and some other field on which coloured labour is employed, such as Mysore.

The author states that the Transvaal deep levels are more dangerous than the outcrops, and, judging by the death rate, this would appear to be so. I can see no sufficient reason for the death rate on the deep levels being higher than that on the outcrops; the mining methods followed are the same. I agree with the miners and attribute the higher death rate on the deep levels to the "rushing" methods which are adopted. A huge output is required from a deep level, and the mine managers have to secure that output, or go. In consequence every official has the fear of "losing his job" unless the promised output is secured; the underground work is rushed, risks are taken, and the accident rate naturally goes up. Why is it that on the outcrop mines, where there has been no great increase in the monthly tonnage crushed, the accident and death rates keep low; these mines are re-opening old stopes, the ground is often worked out on three sides of them; therefore they should be more dangerous than the deep levels and yet they have fewer accidents.

It is often argued, that the best men keep to the outcrops, and it is difficult to get good men in a deep level, and therefore fewer accidents occur on the outcrops; I think the real reason lies in the fact that the lives of many of the outcrops are too short to allow for an increase in stamps and tube mills, and consequently the men are not rushed to put up the tonnage. One often hears that such a mine captain "is a good man, he is a hustler," it naturally follows that he hustles the shift bosses, and the men are hustled. The shift boss is not usually polite to the man who is late in starting his boys. Some stopes can be examined in an hour, but there are many in which the hanging and footwalls are so treacherous, and the stopes so difficult to get about in, that two hours will not suffice for a proper examination and the removal or securing of dangerous hanging and footwall. The mine officials recognise this, but some of them make no allowance for it, and do not regard the careful man with the favour he deserves. It seems natural that as the workings get deeper, so the roof pressure increases, but I very much doubt whether this increase is so great as is thought; personally I think there is more pressure on the roof supports at a depth of 300 feet in an outcrop where much ground has been worked out, than there is in a new deep level mine at 3,000 ft. Are pillars being left more frequently or of much greater size in a deep level than in an outcrop, or are greater precautions being taken to hold up the roof? I think not.

When the outcrops are exhausted and the deep levels nearly so, extraordinary pressures must be expected; but at present I am convinced that much of the flaking of pillars which is often pointed to as proving the great weight the pillars are under, is due more to weathering than to weight.

The author states that owing to the time occupied in getting to the working places and to the abolition of the dinner hour, the work has now to be compressed into nine hours instead of ten; I do not agree with him. The time spent in the shafts on a deep level is less than on an outcrop, on account of the fast hauling in vogue on the deep levels; in any case the time spent in getting to the working place is little when compared to the duration of the shift. In the old days when men came up for dinner, their watches went fast about 11.30 a.m., and there was no great competition to catch the first skip down after dinner; while on descending, arguments started in the boarding house were thrashed out on the explosives boxes, until the appearance of an official broke up the discussion; so that the time for dinner usually meant a loss of two hours.

The author remarks that to drill the round over, and complete a bench, requires greater speed and higher skill than was necessary in the early days. Then men were not asked to get a round out in a day, and two four-hole benches in a stope was considered a fair day's work; nowadays a manager points with pride to increased footage driven and increased hand and machine drill efficiency. These increases are often gained at the expense of an increased accident rate.

Further on in the paper, the statement is made, that when an extra hazardous job has to be undertaken, old and experienced natives are taken as assistants, and in consequence the accident rate is increased among them; I think the increased accident rate among old mine boys, is due to their efforts to "show off" before new arrivals, by so doing they often place themselves in great danger with disastrous results. Few accidents occur in really dangerous places, the reason given for this is, "we put on good men, give them plenty of timber, *and don't hurry them.*" If sufficient good men were available, if they were given plenty of timber and were not hurried, throughout the mine, our accident rate would soon come down to reasonable proportions.

The suggestion of lectures to miners seems excellent, but I don't think the lectures would be successful—the good practical miner is convinced that he has nothing to learn, and has no use for theory. After a day's work in the mines, few men will attend a lecture. The author advocates the employment of hanging wall inspectors. I used to advocate the same, but am now convinced that they are of little use. The appointment of a hanging wall inspector tends to make the stoper careless, by relieving him of a responsibility he ought to take; the man who works in a stope, is the proper person to examine it. A hanging wall inspector, to be of any value, must be an active man and a thoroughly competent miner, with an appetite for work, because the more thoroughly he examines a stope, the more work he will have to do in it, and if it is a difficult stope to get about in, the work may not be pleasant. To examine stopes properly and at frequent intervals, will require a very large number of hanging wall inspectors. A competent hanging wall inspector would do well on contract, and in consequence good men will not take on the job, except at a higher rate of pay than that given; in the past hanging wall inspectors have often been poor miners, or novices at mining. I do not agree with the appointment of a small army of Government Inspectors to examine the mines; the mining companies should keep their mines safe, so why should they not appoint their own inspectors?

I notice in the Chamber of Mines Report for 1910, that in that year the amount of compensation paid for accidents, was (one hundred and sixty thousand) £160,000 per annum; apart from reasons of humanity, it would appear sound policy to endeavour to reduce this amount. "Experts" of all kinds are very numerous on the Rand, but we have not yet heard of the appointment of an Accident Expert. It may be agreed that the Mine Inspectors should be the Accident Experts, but it must be remembered that Inspectors of Mines have not the power of getting behind the scenes, which a representative of the houses would have. A mine inspector may notice a mine getting into a dangerous state, but he cannot tell whether it is the fault of the manager, or due to the policy he has to carry out. A mine inspector has to appeal to a magistrate to support him, whereas an unofficial inspector would have the consulting engineer to back him up. It is a very difficult thing to prove to a magistrate that such and such a system of mining is dangerous and should be stopped, whereas a few minutes talk with a consulting engineer will cause him to look into the matter, and if necessary rectify it.

Systematic timbering would prove very expensive, and in only a few cases, of any value. A stone weighing a few pounds will do enormous damage in a steep stope, and unless the timbering is so close as to be impracticable, systematic timbering will not prevent such small stones falling. Again, stopes in the same mine vary so, that it would be impossible to lay down any rule for timbering which could apply throughout a mine. Systematic timbering in a coal mine is a different proposition, the working places are flat as a rule, and most of the timber is drawn and used again; although systematic timbering has not increased the costs in coal mines, it would do so very considerably on the Rand, where the timber would usually have to remain, once it was put in. Rather than adopt systematic timbering, it would be better to adhere to the following conditions in stoping:—

Steep Stopes.—Beat the bottom of the stope face well back, and have boxholes at frequent intervals, so that the shovel boys are protected from falls of rock by the overhanging portion of the stope face; it is among shovel boys that the greater proportion of fall of rock accidents occur. Stull at frequent intervals, keeping the stulls as close to the face as possible and well loaded.

Flat Stopes.—These should be well packed, and if the hanging is bad, the packs should be kept well up to the face; the argument against this is that the fine is blown into the pack and lost, but if loose dirt is shovelled up against the pack this loss will not amount to much.

In both flat and steep stopes pillars must be left, but where and of what size depends on local conditions, and this must be left to the discretion of mine officials; but when a mine captain has hardened his heart and given instructions to cut a pillar, he should cut a big one; small pillars lose so much in the cutting that when completed they are neither useful nor ornamental, but sources of danger. Timber should be used more freely than it is now, and the timber bill of a mine should not be too closely criticised if it is high.

We often hear that sticks cannot be put in near the face, because they will be blown out by the first blast; this trouble is much exaggerated, as a stick if properly put in it will take a lot of knocking out. If sticks are blown out they should be replaced at once, but miners often start their boys and then replace blown out sticks.

Explosive accidents are enormously high, and in the majority of cases preventable. If a man carefully examines his working place, he should find the old holes, but with the rushing methods now adopted, men sometimes do not examine their places carefully, and so holes are overlooked; explosive accidents in winze and shaft sinking are more easily forgiven, but with care many of these could be avoided. Miscellaneous accidents, such as falling down shafts, found dead in shafts and stopes, struck by skip, etc., etc., are very often due to the ignorance of the native. The policy of keeping new boys in gangs, is a wrong one, they wander about the mine like lost sheep, climb over fences and get into danger, because there is no one to warn them not to do so; new boys should be distributed among experienced natives.

I am convinced that the accident and death rates would be reduced if the following conditions prevailed:—

1. If mine officials and miners were less "rushed" than they are at present.
2. If more attention were paid to accidents, and less to low costs.
3. If more care was taken of inexperienced natives underground.
4. If the mine officials would insist on the regulations being observed.

No one will contend that the natural dangers on the mines of the Witwatersrand are so great that they cannot be surmounted with care and skill.

The Government Mining Engineer's Report gives a list of the mines which have the highest and lowest death rates. Why is it that some mines figure more than once in the same list. It cannot be entirely a question of luck. Much depends upon the manager of a mine, changes of management are usually associated with an increase or decrease in the death rate; a successful manager is expected to get an increased tonnage and low costs, but how rarely do we find that these are associated with a low death rate. We have got so used to accidents in the past that we have grown callous; it was and may be still, the custom on many mines in England to stop work for the day as soon as a fatal accident occurred, and again on the day of the funeral; possibly if that excellent custom prevailed on the Rand, we should realise more fully the gravity of death, and have fewer accidents. It may be argued that I have viewed the question entirely as an Inspector of Mines, whose view is naturally that the accident rate must be diminished; to argue that to carry out successful mining on the Rand necesitates the present appallingly high death rate, is a serious reflection on the capabilities of our engineers.

Mr. D Rees (*Member*): The author's paper is a very valuable contribution, and he should be congratulated on directing the attention of the mining members of the Society to the above subject. As most of the papers read at this Society deal more or less with chemical and metallurgical subjects, a paper on a mining subject comes as a pleasant change to the mining member.

I note that the author in comparing the Transvaal accident and death rates with the English rates has not even mentioned the different conditions existing in both countries, therefore a word or two in that direction would I think not be out of place.

The majority of miners in the English mines are trained from boyhood to earn their livelihood in that capacity: the first thing a boy is taught as he enters the mine are the dangers appertaining to the calling. It is so thoroughly grafted on his young mind that after gaining a little experience he is always on his guard. The experienced miner considers it his first duty on entering his working place at the commencement of every shift to thoroughly examine the roof and sides. The majority of miners in the Transvaal are ignorant unskilled natives, with one white man in charge of about twenty or so of them, and often enough that white man is a cosmopolitan who has served the time required by the Mining Regulations to qualify for a blasting certificate, as an assistant to a developer on the rock drill, and has not even an elementary knowledge of how to ascertain the condition of the hanging and footwall, for that is not required to qualify for a blasting certificate, as the certificate specifies that it is no guarantee as to the ability of the holder. Thus the conditions are entirely in favour of the English mines, where one skilled miner has only one unskilled man in his care, and a man hardly ever gets charge of a place of his own unless he has served as an assistant for about two years, and then he has only himself to care for. And, again, coal mining is very different to metal mining; in almost every coal mine there is sufficient debris or waste to pack the excavated space as the working face proceeds, the goaf or pack is generally kept within about 4 ft. of the coal face, thus there is much less hanging exposed than there is in a stope in a metal mine.

I quite agree with the author that most accidents from falls of ground occur in districts where the roof is good; we need only refer to this (the Pilgrims Rest) district for proof, as there is hardly a mining district in the Transvaal where the ground is more broken and the accident rate so low. And to compare the Transvaal accident rates with the English rates on a tonnage basis is very misleading, as the great majority of mines in England are coal mines, and the majority of mines in the Transvaal are metal mines and as a ton of ore takes much more getting than a ton of coal, it naturally follows that there are many more men employed here per million tons produced.

As I have not seen the mining statistics of England of late years, I do not wish to qu stion the accuracy of the author's figures, but if it is correct that the death rate in England for 1910 was 1·116 per million tons raised, then the average has been reduced wonderfully in the last five or six years, as the average death rate in mines under the Coal Mines Act, in England, Scotland and Wales in 1904 was 4·3, and in 1905 the average was 4·6 per million tons of minerals raised.

The increase in accidents in the Transvaal since 1903 is partly due in my opinion to the fact that owing to the great quantity of ore that has been extracted from the mines on the Rand there must be in some of the older mines vast areas of excavated ground unsupported, which causes a great pressure on all the mine in addition to the increased roof pressure caused by the increased depth, and as the measures get less steeply inclined with depth, the superincumbent strata lie more heavily than in steeply inclined measures. I do not agree with the assumption that the increase is due to the "speeding up," as there cannot be more rush and hurry in the Transvaal mines than there is in the English coal mines where the average annual output per person employed is about 300 tons.

Everyone will agree as to the seriousness of the situation, and that it is essential that something be done to remedy this. The author suggests education in mining safety methods which everyone would consider a very good idea, but here again figures are very discouraging. In the last ten or twelve years the opportunites and facilities for the miner to obtain education in mining have advanced immensely in England and as a result of that there are hundreds if not thousands of miners holding first class, and especially second class mine manager's certificates working at the coal face today in that country. In spite of that according to Table I. of the author's paper the death rate there has gone up ·56 per thousand in the last seven years. And there has also been an appreciable improvement in the facilities for the miner to gain a better knowledge of his profession on the Rand of late years, and here also the rate has gone up a little.

I thoroughly agree with the author in advising the appointment of roof inspectors, and in order to avoid the division of the responsibility I suggest that the shiftboss be responsible for this, and that he should examine every working place in his section before the commencement of each shift, and that he should commence his round at such time that he would be able to return to the top of the shaft, or some other suitable place, in time to see every miner that works in his section before they proceed to their work, (the same as the fireman or deputy in England) and that he should instruct every white miner as to the condition of his place as he goes on shift, and if some timber was required in any particular place he should instruct the timber-man to proceed there at once.

The author is mistaken when saying that a detached slab of hanging wall over 3 ft. thick cannot be detected by tapping with a hammer. It is true that a large piece of detached hanging cannot be detected with a small hammer or sampler's pick, but if a heavier hammer or pick is used he will find that a very large piece is easily detected by tapping and placing one hand against it.

Systematic timbering I do not think is very practicable on the Rand under the existing conditions, neither do I believe that it would improve matters if adopted, as often a greater amount of timber would have to be set than would be actually required, and with each blast half of this would be blown out, and would again have to be reset according to law, and whilst doing that the timberman and his assistants, (if the stope was steeply inclined) would be exposed to overhanging rock that was loosened in the face by the blast in addition to any dangerous hanging wall that might be in the vicinity.

If the President of the Mining, Civil, and Mechanical Institute for the current year says, that systematic timbering has not increased the working cost in the coal mines of Great Britain, neither has it decreased the death rate per thousand in that country according to the author's Table I., then I fail to see what has been gained. Thus the only remedy that I can suggest is compulsory packing where possible with sand etc. and compulsory daily roof inspection.

Mr. Wm. Cullen (*Past-President*): At the last meeting I made some remarks on the changes which were continually taking place among the white employees on the mines, and it has struck me since that it might give the question a more practical turn if I gave the figures for the Dynamite Factory. These I have got out and they have proved of interest even to me. The number of white employees in the factory varies from 255 to 270 year in and year out, but owing to the encouragement given to white labourers these figures are a bit high. I find that during the last financial year of the Company the number of changes among white employees was 80 = 30% made up as follows:—

(1) Dismissed	22
(2) Left of own accord ...	53
(3) Died	2
(4) Killed in explosion ...	3
	80

These figures are admittedly high—much higher than usual as will be seen presently—and they call for a little explanation, Most of Class 1 were dismissed for drunkenness or breach of factory regulations. Class 2 embraces all those who were taken on temporarily for urgent construction work caused by the explosion, also quite a large number who left because the explosion scared them. The largest proportion were, however, the "Weary Willies" who wanted employment and not work. I would like the newspaper advocates of white labour to try a practical experiment with the unskilled white labour available here. One does find a good man here and there but only after a lot of searching. Classes 3 and 4 speak for themselves.

I come now to the average period of employment, details of which are given in the following table :—

16 years and over	...	3
15 ,, ,, under	16	6
14 ,, ,, ,,	15	6
13 ,, ,, ,,	14	1
12 ,, ,, ,,	13	5
11 ,, ,, ,,	12	4
10 ,, ,, ,,	11	7
9 ,, ,, ,,	10	34
8 ,, ,, ,,	9	13
7 ,, ,, ,,	8	7
6 ,, ,, ,,	7	14
5 ,, ,, ,,	6	20
4 ,, ,, ,,	5	24
3 ,, ,, ,,	4	12
2 ,, ,, ,,	3	24
1 ,, ,, ,,	2	26
Under 1 year's service	...	57
		263

I find that 100 or about 37% of our employees have been with us for 7 years, and even including all those under 1 year's service (57), a large number of whom are so-called white labourers, the average period of employment is 5 years 3 months. Figures even better than these can be shown by Mr J. W. Quinn.

Dr. D. Macaulay, M.L.A. (*Member*) : It is rather presumptuous on my part to address you on a purely mining and technical subject, which is outside the scope of my ordinary avocation. But Mr. Chilton's paper deals with the question of mine accidents in such a comprehensive manner that it forms a suitable corollary to the discussion which occupied the attention of the Society some years ago on "Safety Measures in Mining." I therefore feel disposed to say a few words about it, although it may seem presumptuous on my part to do so, more particularly after hearing the remarks of Mr. Cullen and the appropriate remarks of Mr Saner. I wish to touch on one aspect of the question, and more or less to crystallize the vague statements made by Mr. Chilton himself, by Mr. Cullen and Mr. Saner. They are all agreed as to the paramount importance of the changes that are perpetually taking place on the mines. This fact has also struck me in my own work, and within my own knowledge as to the accidents which have taken place on the mines. The Government Mining Engineer has been drawing attention to this fact year after year, and he has been classifying the accidents according to the best of his ability. But he has been like the voice of one crying in the wilderness. I would like to pay a tribute to Mr. Kotzé not only for the readiness with which he supplies information, but for his evident desire to do everything in his power to enforce the regulations and improve the working conditions on the mines.

I have long thought that this perpetual changing on the mines had something to do with the accident rate, which, unfortunately, is the highest of any metalliferous mines in the world. My present object is to give statistical proof of this. We have to take for granted that owing to the conditions of employment, the native labour supply changes about 100% per annum, and, as Mr. Kotzé says in his report, the natives are little more than semi-savages, and at the best, if they remain with us, they require constant supervision. Seeing that they are continually changing like this, it is all the more necessary that the skilled supervisor should be more or less constant and should be very skilled indeed. I asked Mr. Kotzé, who readily complied, to call for a return showing the changes in white employees on the mines and the result is so astounding—I might say appalling—that it is absolutely necessary that public attention should be drawn to it in order, if possible, to put an end to it, because so long as the present system, which has arisen from so many causes on the Rand—so long as this system continues, a system which is economically and socially unsound, so long we may expect neither industrial efficiency nor political stability, and I think it is absolutely necessary that you who are interested in the industry should try to put a stop to such a system. I have returns here, supplied by the Government Mining Engineer, showing the changes on some 53 mines during the period of six months. The average total of white employees on these mines was in the neighbourhood of 22,000 per day. I think you may be surprised to hear, that during six months there were on those mines 17,745 changes among white employees. I ask you to consider for a moment what possible supervision can you get of the army of semi-savages who are carrying on the manual work of the mines if such a condition of things is going on, and if those changes continue. I believe Mr. Cullen has asked the Government Mining Engineer for another very interesting return. The changes are not restricted to the lower grade of the mines; they are going on from top to bottom. As a fact, the conditions on the mines to-day might be described by the old lines :

Big fleas have little fleas upon their backs to bite 'em ;
Little fleas have lesser fleas, and so "ad infinitum."

And so it goes merrily on. I think it behoves us here, who are members of a scientific Society, whose sole aim in the past has been not self-glorification but the interests of the employers and our comrades, to put an end to this appalling state of affairs. I do not propose to enter into the causes of the changes. I do not blame the management, and I do not blame the men entirely. I have not the slightest doubt that the blame could be scientifically apportioned to several decimal points. That aspect of the question you know better than I, and I would suggest that you should adopt the same plan, as you did with "Safety Measures in Mining," and keep the discussion open for a long time, so that you will get a full expression of opinion from members of the Society. I think it must be gratifying to you, as it is to me, that the agitation which we commenced in 1905 and carried on with your permission in 1906, is now bearing fruit, and that, so far as certain health aspects of the mines are concerned, the authorities are going to do everything possible to make the mines as healthy as they ought to and can be. I have not the slightest doubt that if you adopt the same policy in regard to the diminution of accidents you will have similar happy results. I may give you at a later date some of my own ideas. Meanwhile, I wish to put before you the astounding changes on the mines. While those changes go on, the death rate will keep up. The most important fact is that the Government Mining Engineer has been able to trace a direct relationship and parallel between the changes on the mines and the accident rate, that is, the changes amongst the whites. I will give you this table grouping the chief producing mines and showing the percentage of changes per month in the white labour force and the total death rate from accidents (including white and coloured workmen) on the mines during the half-years ending June 30th and December 31, 1911, respectively. (These figures cover 53 producing mines).

Changes per month per cent.		Death-rate from accident. Jan.-June.	July-Dec.
0 to 5	...	3·47	0·93
6 ,, 10	...	2·85	2·83
11 ,, 15	...	4·10	4·17
16 ,, 20	...	4·34	3·80
21 ,, 25	...	4·96	5·00
26 and over	...	5·10	5·08

You will notice that in some cases the changes are more than 26% in *one* month. The average is 80% in *six* months.

A Member: In what period?

Dr. Macaulay: Six months, but the changes are going on daily, of course. I will leave those figures to your consideration.

The meeting then closed.

Contributions and Correspondence.

TABLE OF PEBBLE LOADS OF DIFFERENT DEPTHS IN A TUBE-MILL 22 FT. IN LENGTH AND 5 FT. IN INTERNAL DIAMETER.

By W. R. Dowling (Vice-President).

Depth of Pebble Load.	Cubic Feet of Pebbles in Load.	Weight in Tons of Pebble Load. (105 lb. per cubic foot.)	Relative Volume and Weight of Pebble Load as percentage of total internal capacity of mill.
12 in. above axis of mill	323·0	16·96	74·8
11 ,, ,, ,,	314·4	16·51	72·8
10 ,, ,, ,,	306·1	16·07	70·9
9 ,, ,, ,,	297·3	15·61	68·8
8 ,, ,, ,,	288·3	15·14	66·7
7 ,, ,, ,,	279·8	14·69	64·8
6 ,, ,, ,,	270·6	14·21	62·6
5 ,, ,, ,,	261·4	13·72	60·5
4 ,, ,, ,,	252·7	13·27	58·5
3 ,, ,, ,,	243·5	12·78	56·4
2 ,, ,, ,,	234·1	12·29	54·2
1 ,, ,, ,,	225·4	11·83	52·2
Level with axis of mill	216·0	11·34	50·0
1 in. below axis of mill	206·6	10·85	47·8
2 ,, ,, ,,	197·9	10·39	45·8
3 ,, ,, ,,	188·5	9·90	43·6
4 ,, ,, ,,	179·3	9·41	41·5
5 ,, ,, ,,	170·6	8·96	39·5
6 ,, ,, ,,	161·4	8·47	37·4
7 ,, ,, ,,	152·2	7·99	35·2
8 ,, ,, ,,	143·7	7·54	33·3
9 ,, ,, ,,	134·7	7·07	31·2
10 ,, ,, ,,	125·9	6·61	29·1
11 ,, ,, ,,	117·6	6·17	27·2
12 ,, ,, ,,	109·0	5·72	25·2

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Genesis of Iron Sulphides.—*General Principles.*—"The genesis of ores is essentially a chemical problem. It involves in each individual case the formation of one or more minerals. Barring those rare instances in which the geologist may be able to observe a mineral in the actual process of formation, the conditions of formation have to be inferred from the data obtained by field observation, supplemented by what chemical information is available. At the present time our chemical knowledge of minerals generally embraces something of their behaviour toward certain reagents, and often little, if any, more. Accurate data on the genesis of most of them is still wanting. This will have to be accumulated by actually forming the minerals in the laboratory and studying the conditions in systematic fashion. The questions of first importance in mineral synthesis concern, of course, the conditions of temperature and pressure and the composition of the systems in which the minerals have formed.

By laboratory study, certain limits to these conditions may be set, outside of which the mineral cannot exist, though in general the limits will be wide. Thus if it be learned that a certain mineral cannot exist above 450°, it is known only that the mineral in nature must have formed somewhere between this point and the lowest temperature which prevails in the earth's crust. It is quite probable that we shall be able in many cases to narrow this range of temperature by the study of paragenesis. Thus if we could determine the temperature limits of two different minerals which, on account of their intergrowth or otherwise, could be shown to have formed at the same time, it would be certain in that particular instance that the minerals formed inside the temperature range which is common to both. Thus far, we have been able to set these temperature limits for comparatively few minerals, chiefly those which are capable of existing in more than one crystal form. Here the temperature is often the most important condition determining *which* crystal form shall appear, and this characteristic of the mineral may be taken as an index of the temperature which prevailed in its genesis. In a similar way, the composition of original solutions has to a slight extent been determined, since it has been observed that crystal form is sometimes determined by the composition of the solution in which the crystal grew. For the determinations of pressure limits, there is at present no method.

In drawing inferences regarding the genesis of natural minerals from experiments in mineral synthesis, the investigator must be constantly on guard lest he mistake secondary for essential conditions. It is therefore necessary for the chemist who works in this field to keep in constant touch with the geologist.

The Sulphides of Iron.—In this paper will be considered the application of some of the above principles to the mineral sulphides of iron which form one of the most important classes of the sulphide ores. The frequent association of these minerals, pyrite, marcasite, and pyrrhotite, with other valuable minerals of the sulphide group would indicate that the knowledge of the conditions of the genesis of the former might be applied to the latter as well. Also the association of the different sulphides of iron with one another have suggested several interesting problems of more particular nature which will here be considered.

Pyrite and Marcasite.—Pyrite and marcasite, the disulphides of iron, often occur in such geological formations as to lead to the belief that they were precipitated from cold surface solutions. Typical deposits of this character are those of the Mississippi Valley. Geologists have reasoned that these and similar deposits have been precipitated from sulphate solutions because surface waters have frequently been observed by them to carry ferrous and ferric sulphates which have formed by the oxidation of older deposits of pyrite or marcasite through the action of atmospheric influences:

$$FeS_2 + 7O + H_2O = FeSO_4 + H_2SO_4.$$

Observation has gone a step farther. Recent pyrite has been observed in wooden conduits, and on the twigs of trees in hot springs, and the fact that the same mineral occurs on coal, and sometimes associated with asphalt, has led to the conclusion that the ferrous sulphate has been reduced in some way by carbonaceous matter to FeS_2. A disulphide obviously could not be formed by simple reduction, but a more complicated process in which ferrous carbonate would be one of the products may be conceived. The following equation in a very imperfect way conveys this notion:

$$7C + 4FeSO_4 = 2FeS_2 + 2FeCO_3 + 5CO_2.$$

Omitting details, it may be stated that experiments which have been directed toward the formation of pyrite and marcasite through the agency of carbonaceous matter, have been entirely without result. A possibility of synthesis is suggested by the fact that pyrite and marcasite are commonly associated with other sulphides of a simpler nature, such as sphalerite and galena which are directly precipitated by hydrogen sulphide. Experiments have in fact shown that both pyrite and marcasite may be obtained by the action of hydrogen sulphide on ferric sulphate solutions. It is well known that ferric sulphate is directly reduced by hydrogen sulphide to ferrous sulphate with a simultaneous precipitation of sulphur, but it had not previously been noticed that the further action of hydrogen sulphide and sulphur on ferrous sulphate slowly gives rise to a dark precipitate having the composition FeS_2. This reaction, which may be represented by the equation: $FeSO_4 + H_2S + S = FeS_2 + H_2SO_4$, takes place even at ordinary temperature, though quite slowly. A microscopic examination of the product proves it to be distinctly crystalline, though the crystals are minute. If the reaction is allowed to go on at a sufficiently high temperature in a sealed glass tube, say at 200° C., the crystals grow large enough to measure, and it has been found that they are partly marcasite—a mineral which had not previously been made artificially. A method was devised by H. N. Stokes for the analysis of mixtures of natural pyrite and marcasite which we have used to great advantage in studying the laboratory products. It consists in the oxidation of the sulphides by a dilute solution of ferric sulphate. The products of the reaction are ferrous sulphate, sulphur, and sulphuric acid for both minerals, but the relative quantities of the products are decidedly different. Only 14% of the sulphur in marcasite is thus oxidized to sulphuric acid, the rest being precipitated as free sulphur, while 56% of the sulphur in pyrite is oxidized. It is thus possible to determine in any mixture of the two sulphides the percentage of pyrite and the percentage of marcasite. As was stated above, the action of hydrogen sulphide on a mixture of ferrous sulphate and sulphur gives in general a mixture of both pyrite and marcasite, and the application of Stokes' method has proved that there are two essential conditions which determine the proportions of the two. These are temperature and acidity. The higher the temperature during the reaction, the greater is the percentage of pyrite obtained. It is also true that the lower the acidity of the solution, the greater is the percentage of pyrite formed, until, when the solution becomes neutral or alkaline, the product is practically all pyrite. The equation above shows that it is not possible to maintain the solution neutral when hydrogen sulphide acts on the ferrous sulphate and sulphur, for sulphuric acid is one of the products, but if we substitute suspended ferric hydroxide for the ferrous salt, the solution remains neutral (if we waive the question of the weakly acid nature of hydrogen sulphide itself). Another way in which we can command the same essential conditions is to heat pyrrhotite or ferrous sulphide with free sulphur and a water solution of hydrogen sulphide which contains a very little sodium bicarbonate. This solution dissolves and carries over the sulphur to the pyrrhotite, thus forming FeS_2 in a practically neutral solution, and the result is pyrite.

An alkaline solution of sodium polysulphide precipitates from a ferrous salt an amorphous mixture of ferrous sulphide and sulphur, but these combine slowly under the influence of heat, and probably, even at the ordinary temperature, with the formation of pyrite.* These data enable us to draw some important conclusions relating to the formation of these minerals in nature:

1. We know that the mineral marcasite can be formed by the action of hydrogen sulphide on ferric sulphate solutions, or, what comes to the same thing, a solution containing ferrous sulphate† and hydrogen sulphide to which the air finds some access. If the solution contains as much as 1% of free sulphuric acid and the temperature is not above 100°, the product is all marcasite. It is also not improbable that pure marcasite can form at a higher temperature if the solution is still more acid. Now, these conditions agree strikingly with what we find about the genesis of marcasite from field study. It is always a surface mineral, presumably formed from surface solutions at *low* temperatures. It is also a matter of common knowledge that such solutions when they contain iron usually carry it in the form of sulphate, together with sulphuric acid, since they result from the oxidation of an older body of pyrite or marcasite.

2. We know that pyrite forms from solutions which are neutral or alkaline by the action of alkaline polysulphides on ferrous salts and by the action of hydrogen sulphide on ferric hydroxide. Both reactions are essentially the union of ferrous sulphide and sulphur in a neutral or alkaline medium. Coming again to the deductions from field observation, it is found that pyrite is always deposited by hot waters instead of marcasite, and also that deep veins which formed under comparatively high temperatures contain pyrite, never marcasite. Now, hot terrestrial waters are practically always alkaline, not only because silicates are hydrolyzed by hot waters with the necessary formation of an alkaline solution,‡ but because if a strong acid were allowed to act on rocks at the high temperatures which prevail in the depths of the earth's crust, chemical action would soon neutralize it by the decomposition of carbonates, silicates, and the like.

3. Pyrite and marcasite form together when hydrogen sulphide acts on ferric sulphate unless the percentage of free acid and the temperature are sufficiently low. The quantity of free sulphuric acid must be about 1% at 100° to insure the formation of marcasite free from pyrite, while at 300° it must be considerably more, if indeed it is possible to get a product free from pyrite at that temperature. Pyrite and marcasite are very commonly found together in nature—often in the same hand specimen and sometimes intergrown. Although in some cases the two minerals probably belong to different periods of deposition, it seems entirely probable, in view of the laboratory results, that in other cases they are contemporaneous.

Relation of Marcasite to Pyrite.—When marcasite is heated to a temperature of 450° C., under conditions which preclude oxidation, it changes to pyrite. Not only does its colour become yellower and its density higher (the density of marcasite is 4·889, that of pyrite 5·02), but its behaviour with a ferric sulphate solution proves conclusively that it has become pure pyrite; for while at the start the ferric sulphate oxidized only 14% of the sulphur of all the marcasite it decomposed, now it oxidizes 56%. In a comparatively simple way, it can be shown that heat is set free during this change, which proves, of course, that marcasite possess more energy than pyrite. Pyrite cannot be changed directly into marcasite, that is, the change is irreversible and marcasite is what is called a monotropic form. The question of the chemical constitution of the two forms will not be discussed here, except that it may be remarked that they *appear* to be polymorphic forms in which the difference is one of crystalline structure rather than of a chemical nature. More important for the problems of ore deposition is the significance of the temperature at which the change occurs. It goes on at 450° very slowly and could not be detected at 400° in a period of four hours. It will be observed, therefore, that marcasite cannot have formed in nature above 450°, and any minerals which can be shown to have been precipitated at the same time with marcasite are subject, of course, to the same limitation. Pressure cannot be invoked to qualify this statement, for a pressure of several thousand atmospheres was found not to accelerate the change. We may also conclude that a paramorph of pyrite after marcasite would show that the original crystal had been subjected to a temperature the lowest limit of which lies somewhere between 300° and 450°, because at 300° marcasite crystals were found not to be changed to pyrite, either by acid or alkaline solutions.

Pyrrhotite.—This mineral can be readily formed by the decomposition of pyrite in hydrogen sulphide at temperatures above 575°. The higher the temperature is carried, the more sulphur is lost, and a product formed at high temperatures, say at 800° to 1,100° C., takes up more sulphur when heated in hydrogen sulphide below that temperature. Although it is somewhat aside from the main purpose of this article, which is to discuss certain problems of ore deposition, it may nevertheless be of sufficient interest to remark in regard to the mooted question of the true chemical formula for pyrrhotite, that this mineral is really variable in composition, at the same time preserving its homogenity; in other words, ferrous sulphide can dissolve a quantity of sulphur depending on the temperature like any other solubility. At 600° the maximum quantity dissolved is 6%. We may, if we please, represent such a solid solution by the formula $(FeS)_yS_x$.

Under the experimental conditions thus far tested (heating pyrrhotite in dry H_2S) a solution containing only about 6% of sulphur can be obtained, the addition of more sulphur changing the pyrrhotite into pyrite. The pyrrhotite of nature doubtless formed from solution, yet it is remarkable that the highest percentages of sulphur in the natural mineral range in the neighbourhood of 6% dissolved sulphur, corresponding to about 40% of total sulphur.

Geological observations lead to the conclusion that vein forming solutions containing, doubtless, polysulphides, change pyrrhotite to pyrite, and phenomena are observed about contacts which lead to the belief that the reverse action, the decomposition of pyrite into pyrrhotite and sulphur, occurs. In the neighbourhood of contact intrusive masses, where the sulphides of iron occur, pyrrhotite is found close to the contact where the temperature was highest, and pyrite in the colder zones. It is commonly held by geologists that in nature pyrrhotite was formed from the same solutions as silicates like olivine and augite. The experimental difficulties are at present too great to permit attack of this question.

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* At the lower temperatures (100°) amorphous disulphide appears to be the first product.

† It is not essential that the sulphate be used; any other ferrous salt may be substituted.

‡ This of course postulates the presence of some alkali or alkaline earth metal in the rocks, but some of these metals are practically always present.

In conclusion, I wish to express indebtedness to J. L. Crenshaw, John Johnston, and Esper Larsen, in collaboration with whom the experimental data for this article were worked out; and to F. L. Ransome, W. H. Emmons, and especially to W. Lindgren, for geological data."—E. T. ALLEN, *Mining and Scientific Press*, Sept. 30, 1911, p. 413. (H. A. W.)

THE DECOMPOSITION OF NITRIC ACID BY LIGHT· "The authors have investigated the decomposition of nitric acid by light, and find that the reaction is reversible, the products of decomposition slowly recombining more or less completely in the absence of light. The extent of the decomposition and recombination was illustrated by a series of curves. They have proved that pure anhydrous nitric acid decomposes very slowly in the dark, giving the same products of decomposition.

Discussion.

Dr. VELEY expressed his satisfaction that the authors had amply confirmed the conclusion drawn from certain elementary experiments by Manley and himself (*Phil Trans.*, 1908, *A*, *191*, 367) that the vapour of nitric acid rather than the liquid is decomposed by sunlight. The observations of the authors that the reaction, $4HNO_3 = 2H_2O + 2N_2O_4 + O_2$, is reversible on alteration of conditions, appeared to be of especial interest, and was in accordance with results obtained in the study of other reactions in which nitric acid took part.

More information might possibly have been obtained from the authors' results if sunshine records had been given; although it was shown that the actinic effect of the sun was greater in August than in May, a result to be expected, yet the difference found between the effects in May and March was almost inappreciable, a result contrary to general experience.

Prof. T. TURNER asked for information as to the method whereby the pressure in the tubes had been estimated, and how the gases were collected and measured. It was also suggested that by means of a capillary tube closed at the upper end and inserted inside the closed tube, a continuous series of pressure observations might perhaps be taken, instead of only determining the final pressure at the end of the test.

Dr. SCOTT asked whether the authors had considered the possibility of traces, especially of hydrochloric acid and of iron, acting as catalysts. These and other substances which might act thus might conceivably come from the glass apparatus used.

The curves of the pressures obtained with the various proportions of liquid and vapour space were remarkable in the case of a reversible action.

Mr. REYNOLDS, in reply to Dr. Scott, stated that it was possible that traces of iron and chlorine too small to be detected by the ordinary reagents were present in the acid employed through contact with glass. If such traces affected the results, it was, however, curious that the phenomena were so regular when different samples were employed, and that in clear silica tubes rather greater decomposition occurred than in glass; however, as only the vapour was decomposed, it was unlikely that iron affected the phenomena."—WILLIAM CLARKE REYNOLDS AND WILLIAM HENRY TAYLOR, *Proceedings of the Chemical Society*. Dec. 15, 1911. p. 306. (F. W. W.)

THE POROSITY OF IRON AND ITS RELATION TO PASSIVITY AND CORROSION.—"The author showed that: (1) The surface of iron is slightly porous, so that when the metal is immersed in certain solutions the latter are absorbed to a minute extent. (2) The passivity induced by immersion of iron in alkaline solutions, such as those of sodium and potassium hydroxides, is due to absorption of small quantities of these alkalis within the pores of the metal. (3) There are thus more kinds of passivity than one."—JOHN ALBERT NEWTON FRIEND, *Proceedings of the Chemical Society*. Dec. 15, 1911. p. 311. (F. W. W.)

DIAMONDS, HOW AND WHEN THEY WERE MADE BY NATURE.—"The accepted theory that diamonds were produced from organic carbon is now shown to be erroneous. The question is, were diamonds formed before or after organic matter?

The new theory places their origin before, and during an epoch of the earth's history when its atmosphere consisted, for the most part, of carbonic acid gas and carbon monoxide.

The metallic elements of the earth were then in a molten state, liquid and semi-fluid iron flowed freely in various directions, and it must have happened in solidifying that cavities and fissures of various sizes were filled with carbon monoxide and carbon dioxide.

These carbon compounds being enclosed and imprisoned by the surrounding iron, became compressed by the closing together of the sides of the cavities; so that a thousand cub. ft. were compressed into the space of, perhaps, one-fourth of a cub. in., and so highly condensed that the oxygen united chemically with the iron, forming oxide of iron and free carbon, which crystallised by the effect of the enormous pressure produced during the cooling and contracting of the outer matrix.

Diamonds in their native surroundings are found associated with iron; in fact, it is impossible to find a diamond without a trace of iron, when found in river beds and other places, without blue earth or any ferrugenous matter, it proves the immeasurable time that has elapsed since the disintegration of the matrix, which belonged to the molten period of the earths' evolution and million of years before animal or vegetable carbon existed.

For diamonds to have been produced from organic carbon the whole of the elements of the earth would again have had to become nebulous with a temperature of not less than 5527° C.

With the exception of diamonds, carbon does not and never has existed as a separate element at a temperature below 5527° C. It was certainly one of the first to unite with oxygen and form carbonic acid gas, from which all forms of carbon and their compounds have been directly or indirectly evolved.

Graphite and coal are accounted for as a result of organic matter; but the origin of the diamond must certainly be placed in pre-organic times."—ROBERT B. COCKER, *Chemical News*, Vol. 104, No. 2713, p. 248. (J. G.)

THE STABILITY OF THE DOUBLE OXALATES OF SODIUM AND NICKEL, AND SODIUM AND COBALT.—"On boiling nickel oxalate, $NiC_2O_4,2H_2O$, with an approximately 2*N*/solution of sodium carbonate, a green solution is obtained; when this is filtered from the residue, and allowed to remain for about twenty-four hours, green crystals separate, which consist of the double oxalate of sodium and nickel, with but a small amount of impurity. The green solution cannot be crystallised by evaporation aided by heat, as the salt is partly decomposed on heating with water or with sodium carbonate solution, forming a pale green precipitate, but leaving a solution still green in colour. The green solution is also obtained when sodium oxalate and nickel oxalate are boiled together with water, and also when nickel carbonate is boiled with sodium oxalate solution, but in the latter case crystals do not form easily.

Three specimens of the crystalline compound were prepared, one (*a*) by boiling 20 gm. of nickel oxalate with 100 cc. of a 2*N*-solution of sodium carbonate, another (*b*) by boiling 15 gm. of nickel oxalate with 100 cc. of a 4*N*-solution of sodium carbonate, and a third (*c*) by boiling sodium oxalate and nickel oxalate in molecular proportions with water. A green solution was obtained in each case; it was filtered, and allowed to remain for a few days, when the crystals separated, and were washed several times with cold water and dried in the air. On analysis:

(*a*) Found $Ni = 13{\cdot}68$; $Na = 10{\cdot}41$; $C_2O_4 = 41{\cdot}18$; $H_2O = 34{\cdot}73$.*

(*b*) Found $Ni = 13{\cdot}62$; $Na = 10{\cdot}83$; $C_2O_4 = 41{\cdot}18$; $H_2O = 34{\cdot}37$.*

(*c*) Found $Ni = 13{\cdot}77$; $Na = 10{\cdot}79$; $C_2O_4 = 41{\cdot}43$; $H_2O = 34{\cdot}01$.*

$Na_2C_2O_4,NiC_2O_4,8H_2O$ requires $Ni = 13{\cdot}82$; $Na = 10{\cdot}83$; $C_2O_4 = 41{\cdot}44$; $H_2O = 33{\cdot}91\%$.

The results agree sufficiently closely with one another to warrant the conclusion that the same compound is formed in each case, and that its composition is properly expressed by the formula

$$Na_2C_2O_4,NiC_2O_4,8H_2O.$$

In order to test the effect of a large excess of carbonate, 100 c.c. of a 4*N*-solution of sodium carbonate were boiled with three gm. of nickel oxalate; a pale green solution was obtained, from which no crystals separated after five days; later sodium carbonate crystallised out, leaving the solution still green. This result agrees with that obtained on boiling the double oxalate with a solution of sodium carbonate.

The reaction which takes place is obviously a balanced reaction, and may be represented by the equation:

$$Na_2CO_3 + 2NiC_2O_4 \rightleftharpoons Na_2C_2O_4,NiC_2O_4 + NiCO_3,$$

the actual conditions of equilibrium depending on the relative masses of the sodium carbonate and nickel oxalate reacting.

As, however, the most interesting case is that of the action of a 2*N*-solution of sodium carbonate, 5 gm. of nickel oxalate were boiled with 50 c.c. of 2*N*-sodium carbonate, the liquid filtered while hot, the residue thoroughly washed with successive quantities of cold water, and the filtrate and washings diluted to half a litre. Analysis showed that this contained 2·2 gm. C_2O_4 and 0·245 gm. of nickel; these amounts are 87·5% of the C_2O_4, and 15·2% of the nickel in the nickel oxalate taken. The residue left on the filter was found to contain oxalate.

Cobalt oxalate, $CoC_2O_4,2H_2O$, is decomposed in a similar manner, but the amount of double oxalate produced is not so great. Five gm. of cobalt oxalate was boiled with 50 c.c. of 2*N*-sodium carbonate, and filtered and washed. The filtrate was found to contain 2·37 gm. of C_2O_4 and 0·117 gm. of cobalt; in this case practically all the C_2O_4, in the original oxalate enters into solution, but only 7·3% of the cobalt, as compared with 15% of the nickel. The residue consisted entirely of carbonate.

The double oxalate of cobalt and sodium is easily prepared by boiling together cobalt oxalate and sodium oxalate with water. The pink solution deposits crystals on keeping, which on analysis gave the following result:

Found: $Co = 13{\cdot}89$; $C_2O_4 = 41{\cdot}18$.

$Na_2C_2O_4,CoC_2O_48H_2O$ requires $Co = 13{\cdot}88$; $C_2O_4 = 41{\cdot}41\%$.

It is therefore obvious that neither nickel oxalate nor cobalt oxalate can be completely converted into carbonate by the action of a sodium carbonate solution, and that in each case a soluble double oxalate is formed, having the composition given by the formulæ already deduced, this double salt being stable in the presence of excess of sodium oxalate." JOHN WALLIS DODGSON, *Proceedings of the Chemical Society*, Nov. 11, 1911. (F. W. W.)

QUANTITATIVE SEPARATION OF IRON FROM MANGANESE.—"Pyridine is added to a slightly acid solution of the ferric and manganous salts until no further precipitation takes place, the mixture is then boiled for 10 minutes and filtered. The precipitated ferric hydroxide is washed with a hot saturated aqueous solution of pyridine and then with boiling water. Precipitation of the iron is complete, whilst all the manganese remains in solution. With relatively large quantities of manganese, however, it is advisable to dissolve the precipitate in hydrochloric acid and re-precipitate. Metals precipitated by hydrogen sulphide from acid soluions should preferably be first removed. If zinc be present the ferric hydroxide must be dissolved and re-precipitated with ammonium chloride and ammonia."—J. A. SANCHEZ, *Bull. Soc. Chem.*, 1911, 9, 880-881.—*Journal of the Society of Chemical Industry*, 1911, p. 1283. (J. G.)

NOTE ON THE ASSAY OF FORMALDEHYDE.—"The amount of formaldehyde in strong commercial solutions of about 37% strength can be determined by means of a solution of potassium cyanide instead of hydrogen peroxide. ½ cc. of the sample is weighed into a stoppered conical flask of about 150 cc. capacity, and 100 cc of a standard solution of potassium cyanide (approximately N/10) are added. The whole is well shaken and added to a mixture of 40 cc. of N/10 silver nitrate and 10 cc. of 10% nitric acid in a 200 cc. measuring flask. The smaller flask is washed into the measuring flask which is then filled up. After shaking, 100 cc. are filtered off and the excess of silver titrated by means of N/10 thiocyanate solution. The volume of thiocyanate solution used is doubled and subtracted from 40, giving the equivalent of uncombined potassium cyanide in cc. of N/10 silver nitrate. This is subtracted from the corresponding equivalent of the total potassium cyanide added, and the difference multiplied by 0·003 gives the weight of formaldehyde in the sample." E. ELVOVE, *Amer. J. Phar.*, 1911, 83, 455-71.—*Journal of the Society of Chemical Industry*, 1911, p. 1280. (J. G.)

THE ESTIMATION OF WATER IN SOAP.—"The usual methods in use at present for the estimation of water in soaps often lead to anything but correct results. The method here described suffers from none of the drawbacks of these methods, and as far as tried has given good and consistent results. 1 grm. of the sample of soap is weighed from a weighing bottle into a 200 cc. conical flask, 50 cc. of absolute alcohol added, and the flask warmed until all the soap has dissolved. This is best done by adding a fragment of porous tile, and heating on the water bath under a reflux condenser. The solution is allowed to cool somewhat, and filtered through a filter paper into another 200 cc. conical flask, the first flask being washed out with about 20 cc. of absolute alcohol, so as to convey all undissolved solid on to the filter, which is then washed with absolute alcohol. The filter paper with the residue is dried in the steam oven, cooled in a desiccator, and weighed. The weight of alcohol-insoluble matter in the soap is thus found. The filtrate is allowed to become quite cold, and 5 gm. of anhydrous sodium sulphate added. The flask is corked and allowed to stand for at least 12 hours (best overnight). The solution is then filtered into a 200 cc. conical flask which had previously been weighed along with a fragment of porous tile. The sodium sulphate remaining in the flask and on the filter is washed two or three times with warm absolute alcohol. The alcohol is evaporated off on the water-bath, and the residual soap dried in a steam oven for 15 minutes. The flask and soap are cooled in a desiccator, corked, and weighed. This weight plus the weight of alcohol insoluble residue, subtracted from the weight of soap taken, gives the weight of water. Hence the percentage of water is calculated. A necessary point to observe is that the solution before addition of the sodium sulphate does not deposit any soap jelly. If this happens more alcohol must be added. The total volume of alcohol should not exceed about 110 cc." R. H. FITZPATRICK. *Chemical News*, *104*, No. 2713, p. 247. (J. G.)

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* By difference.

MODIFIED COBALT REACTION FOR THE DETECTION OF SMALL QUANTITIES OF HYDROGEN PEROXIDE.—"The following manner of carrying out the cobalt test for hydrogen peroxide enables very small quantities of the latter to be detected. 2 cc. of a solution, consisting of a mixture of equal weights of a 1% solution of cobaltous chloride and of a solution containing 1·6 parts of borax, 20 parts of glycerol, and 100 parts of water, are placed in a test-tube and the solution to be tested is poured on its surface. If hydrogen peroxide be present, a brownish zone appears at the junction of the two liquids. A similar reaction is given by perborates, and also by persulphates on heating or on the addition of a few drops of sodium hydroxide solution."—M. LEUCHTER, *Chem.-Zeit.*, 1911, *35*, 1111.—*Journal of the Society of Chemical Industry*, 1911, p. 1250. (J. G.)

DETECTION OF GOLD WITH TITANIUM TRICHLORIDE —"Titanium trichloride may be used for the detection of gold. When a few drops of the aqueous solution are added to a very dilute solution of gold chloride, an intense violet colouration is formed. One part of gold in 30,000,000 parts of water may be thus detected." — A. STAHLER and F. BACHRAN, *Ber.*, 1911, *44*, 2906-15.—*Journal of the Society of Chemical Industry*, 1911, p. 1261. (J. G.)

METALLURGY.

THE AIR-LIFT AGITATOR.—"Early in the year 1880, the Tonite Powder Co., a British corporation, built a powder factory in Contra Costa county, California, to supply the Pacific Coast States with 'tonite,' a safety powder made from gun-cotton and barium nitrate. To wash the gun-cotton and remove the last traces of acid a pneumatic agitating device, identical in theory and similar in design to the agitator now known as the Pachuca or Brown tank, was built. The original idea came from England and how long it had been previously used in the English factory of this company I have been unable to ascertain.

Referring to Fig. 1, accompaning, the general dimensions and appearance of the tank will be seen. It was built of redwood, like a truncated pyramid, with the smaller base down. It was 16 ft. high, the upper base being 8 ft. square and the lower base 3 ft. As first constructed no central tube was used. Air under a pressure 12 to 15 lb. per sq. in. was admitted into the bottom through a nozzle, and agitated the mixture of gun-cotton and water with which the tank was filled to within about four feet of the top. This tank was operated with fair results until about 1882, when William Letts Oliver, the manager for the Tonite Powder Co., conceived the idea of using a central tube, as shown in the illustration. A tapering tube 18 in. at the top and 12 in. at the bottom, 7 ft. long, was constructed, with its lower end 2 ft. above the bottom of the tank. The nozzle of the air-pipe was extended to reach just inside this central tube. The agitation was very much improved by this tube, and was found to be best when the larger diameter of the tube was at the top. After this tube was introduced, the charge would settle in the bottom when agitation was stopped to permit decanting, and the air did not have sufficient force to set the charge in motion again. To overcome this a jet of air was admitted through a pipe entering at one corner, as shown. This effectively broke up the cake and started the charge. This tank is still in use for agitating gun-cotton at the works of the California Cap Co., which purchased the plant of the Tonite Powder Company..

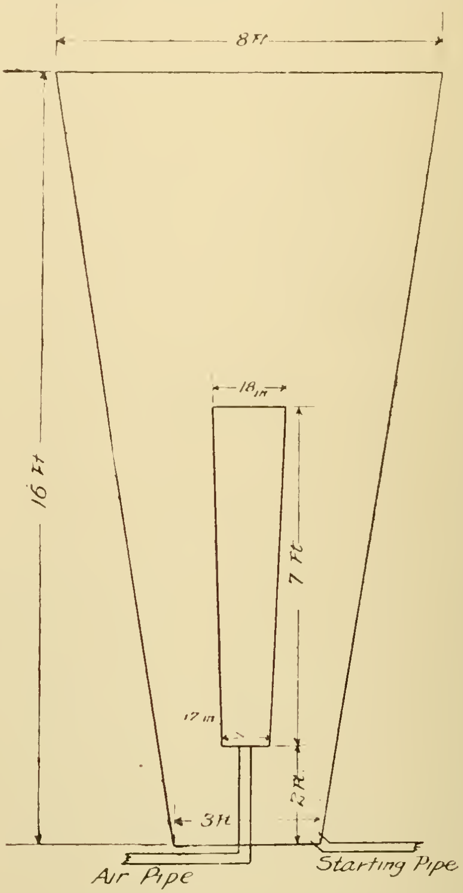


FIG. 1.

In the late eighties, Mr. Sommers, of the University of California, discovered that sulphur bichloride (SCl_2) had the extraordinary property of changing the nature of animal oils, and the discovery was believed to be of great commercial value. However, the reactions set up were of such violence that difficulty was experienced in getting a proper mixture. He consulted Mr. Oliver, who suggested using an agitator similar to that described above. In 1890, the American Lucol Co., of which Mr. Oliver was manager, erected a plant at Stege, to manufacture lucol, a composition of fish oil, kerosene, and sulphur bichloride, an excellent substitute for linseed oil. Fig. 2 shows the general scheme of agitation employed. A redwood tank, 16 ft. high and 8 ft. diam. was erected. The central tube was 18 in. diam., 8 ft. long, and 1 ft. above the bottom of the tank. This tank was filled with fish oil above the central tube, and the air admitted. Sulphur bichloride and kerosene were then run in simultaneously from tanks arranged as shown in the sketch. Considerable care had to be exercised in mixing these ingredients, as the sulphur bichloride and fish oil mixed alone made a hard rubbery mass, while the kerosene and fish oil emulsified. To prevent this, the mixing of the ingredients had to be very rapid and thorough. An air pressure of 12 to 20 lb. was used in the central tube.

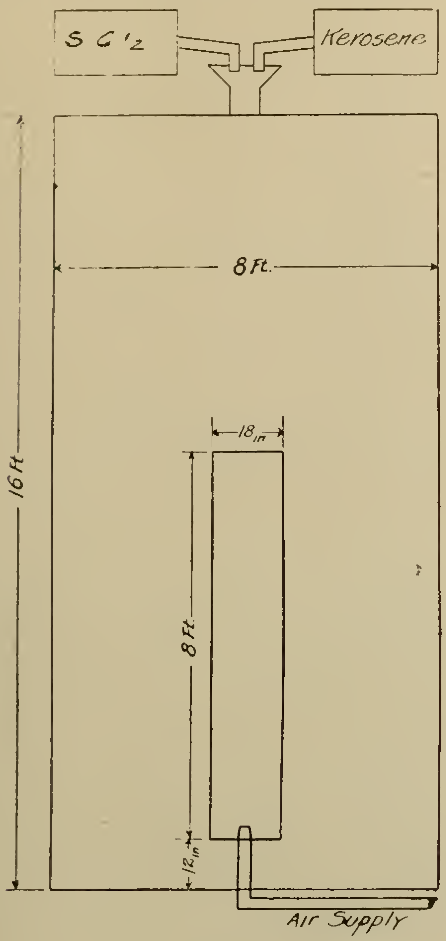


FIG. 2.

The operation of this agitator was so satisfactory that in 1894 four similar tanks were constructed in a new plant for the American Lucol Co. at Cateret, New Jersey. These tanks were steel, with 18 ft. sides and 10 ft. diam., with 16 in. central tubes 12 ft. long, as shown in Fig. 3. These tanks were used continuously until the American Lucol Co., went out of existence in 1895. A novel means for determining the rate of agitation was employed in this process. Small bits of coloured paper were introduced in the tank one colour at a time, and a sample taken from the different parts of the tank gave a visible indication of the remarkable efficiency of the agitation. These pneumatic agitators antedate the experiments of F. C. Brown at Komata Reefs by some years. According to his contributions to the *Mining and Scientific Press* Sept. 26, 1908, the first pneumatic tanks were installed early in 1902, and the central tube was not introduced until 1904. Pneumatic agitators with central tubes had been employed at the North Star Mines, Grass Valley, California, previous to 1904.

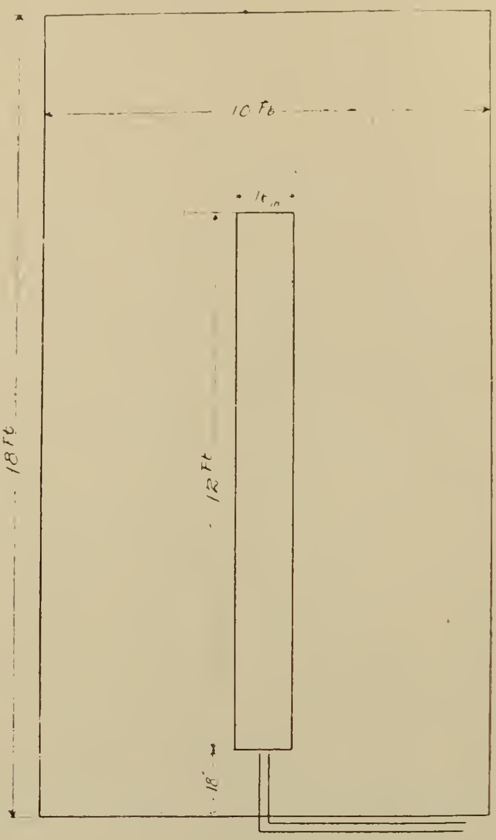


FIG. 3.

During the summer of 1903, Edwin Letts Oliver, metallurgist for the North Star Mines Co., Grass Valley, California, and son of W. L. Oliver, began experimenting with electrolytic processes for recovering the metal content direct from cyanide treated pulp without separating the solutions or using zinc. In this work thorough agitation for the pulp was necessary, as well as some means of bringing the pulp into the electrolytic field of action. A steel tank with 5 ft. sides and 20 in. diam., with a 45° conical bottom was constructed. A 2 in. central tube was used with the lower end 5 in. above the apex of the cone, and its upper end 10 in. below the top of the tank. The nozzle of the air-pipe extended into the central tube. A spreader board was suspended over the top of this tube. The amalgamated copper and iron electrolytic plates were suspended in the annular space, surrounding the upper end of the central tubs. Fig. 4 shows the general arrangement of this agitator. This tank was constructed in August 1903, and proved sufficiently efficient to warrant the construction of additional tanks.

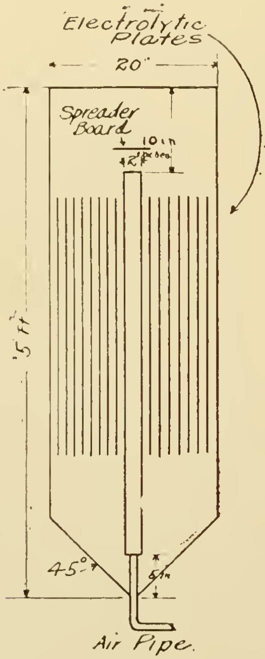


Fig. 4.

In Dec. 1903 two additional tanks were built of wood, with 12 ft. sides and 60° cone, and 6 ft. diam. The central tube extended from 1 ft. above the apex of the bottom to about 18 in. from the top of the tank. A spreader board 10 in. diam. was suspended 6 in. above this tube. The first tube used was 3 in. diam., but this did not give sufficient agitation and was replaced by an 8 in. tube. The nozzle was placed inside the tube at first, but was gradually shortened until it was inside the 4 in. discharge pipe, and just below the apex of the cone. When the central tube choked, as it did occasionally before the nozzle was shortened, the spreader board was removed and a cover board attached. A lever was placed on top of the tube. This lever was held under a cleat at one side of the tank and forced up tight by a screw device on the opposite side. Air under 80 lb. pressure was then turned into the air supply pipe, quickly and effectively breaking up the compacted pulp. Fig. 5 shows the general arrangement of these tanks.

A few weeks after these two were completed a third tank was built. In the spring of 1904 twelve larger tanks, six for the North Star and six for the Central cyanide plant were built, 8 ft. diam., 14 ft. sides, with 60° cones and 8 in. central tubes with spreader board and cover board for breaking the pulp when the tube became choked, as shown in Fig. 6. The pulp originally sent to these tanks was concentrator tailing crushed through a 30-mesh screen at the battery. Later, to increase the capacity of the cyanide plant, the sand was classified out, and leached. The electrolytic plant worked as described until Jan. 1905, giving approximately 75% recovery. This process, which is described fully in U. S. Letters Patent No. 784,120, worked beautifully in theory.

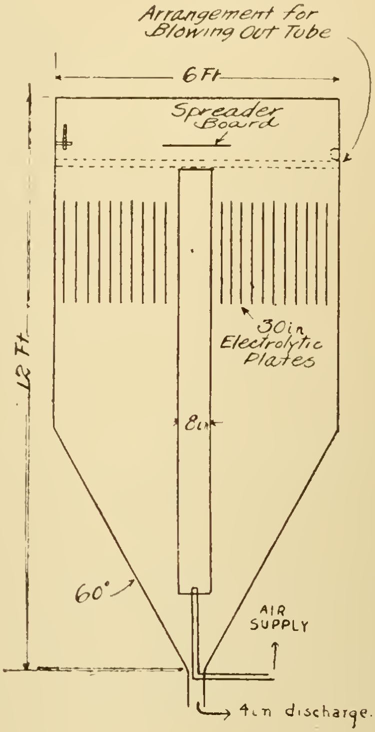


Fig. 5.

Copper plates were used for cathodes and ½ in. iron plates for anodes, 1000 sq. ft. of cathode per agitator being used. The pulp was agitated in a solution containing 0·05% KCN and 0·05% CaO. No salt was needed to increase the electric conductivity of the solution. Contact was made between the cathode and current supply by specially shaped mercury cups, which permitted the mercury to spread over the plate, thus constantly refreshing the mercury surface. The layer of amalgam was formed on the copper plates as in ordinary amalgamating practice. The most economical density was found to be about 50c. gold per sq. ft. Agitation was intermittent, 6 hours being the usual length of time given to each charge. For cleaning up a smaller tank was used. Once a month the copper plates were transferred to the clean up tank. The direction of current flow was reversed, the amalgamated copper plates becoming the anodes and planished iron plates were used as cathodes. Clear cyanide solution was used for electrolyte. In precipitating a current density of 0·05 amperes per sq. ft. of cathode was found to give best results. In the clean up tank, however, this was raised to 20 amperes per sq. ft. the object being to strip the copper plates as rapidly as possible. Usually three or four minutes sufficed to strip the copper plate of amalgam, which was then removed and re-amalgamated with fresh mercury.

The amalgam formed on the planished iron cathodes during the clean up as a thick greyish sponge, ¼ in. or more thick. With such a thick layer of amalgam, more or less would shake off in handling the plates.

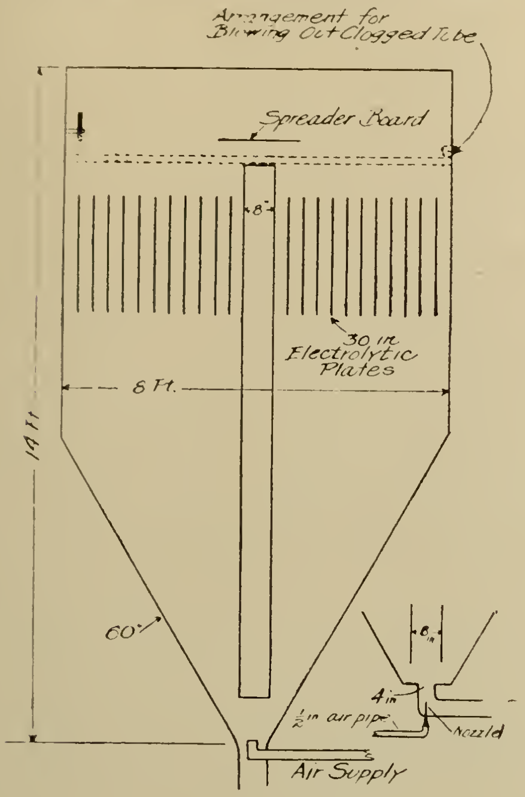


Fig. 6.

The clean-up of the sludge in the bottom of the tank usually yielded two or three hundred dollars. When a sufficiently thick layer of amalgam was formed, the plates were removed, the amalgam wiped off, and excess mercury removed by squeezing through canvas in the usual way. On retorting, the bullion ran approximately 400 gold, 350 silver, and 250 copper, when all parts of the equipment were operating to the best advantage. The clean up could be controlled by the current density, weak currents scarcely affecting the mercury, and depositing copper only on the iron cathode. A stronger current dissolved and deposited some silver and mercury, while a very strong current density formed a deposit high in gold, silver, and mercury, but low in copper. Complete separation of gold and silver was never attempted. The most satisfactory working was obtained when depositing an amalgam retorting a bullion of the composition given above. The electrical equipment used for this work consisted of a generator rated at 1,200 amperes, 5 v., driven by a 10 h.p. motor.

When leaching the sand was begun, new conditions were met, as the solutions from the sand vats were precipitated by zinc. Barren solution from the zinc boxes was pumped into the electrolytic tanks. The zinc in solution was precipitated with the gold and silver. When the plates were cleaned up, and the amalgam retorted, the gold-silver-copper-zinc bullion resulting was a brass, having the colour of good gold bars. The weight, however, was light. In one instance the assay value of a beautiful bar was guessed from $5 to $12 by different employees. The assay showed $3. This bar would have turned any gold brick artist green with envy. The iron oxide, ferrocyanide, and other ferric and ferrous compounds, of which considerable quantities were formed, were at first discharged in the creek. After the plant had been in operation some time, an assay was made of these compounds. This showed a value in excess of $200 per ton. After this all scrapings and rust were saved, even broken and discarded plates yielding $75 per ton. In Jan. 1905, Acheson graphite was substituted for the iron anodes in the treatment tanks. The same current density was employed. Following the introduction of the graphite, an attempt was made to cyanide concentrate, but the sulphides attacked the graphite, disintegrating it very quickly. The experimental work here described was carried on at the North Star mill. During the summer of 1905 the three original agitators were transferred to the Central mill of the North Star Mines Co., for treating concentrates. In Oct. of that year the Central plant was enlarged by the addition of sand leaching.

Early in 1906 electrolytic precipitation was abandoned. Pneumatic agitation was retained, however using the same agitators, but removing the electrolytic plates. Decantation in settling vats was employed for separating solution and slime, a number of additional vats being erected for this purpose. During the time of these experiments, about 30 months, over $200,000 was recovered at a good profit. The cyanide consumption was high, running about 1½ lb. per ton of ore and about the same consumption of lime. The 'holdover' was high; more than twice that of zinc boxes. The anodes, whether iron or graphite, were always troublesome, recovery never checked within 10% of assays (these showing good extraction); while the whole process required extreme care in manipulation.

With a view to finding an economical method for handling the slime Mr. Oliver, early in 1907, begun a series of experiments with filtration devices. The first drum filter was built April 1907, having a diameter of 20 in., and 12 in. long. This worked splendidly, as it was turned by hand, and never operated longer than ten or fifteen minutes without stopping. Assays showed excellent extraction, and the quantity of material handled showed the possibilities, if a machine of large size were constructed.

In June 1907 the first full working size filters were put in operation. Two filters with drums 6 ft. long and 6 ft. diam. were built at this time. The filters as originally constructed did not work satisfactorily. The cake would not discharge properly and a thin impervious coating would form, preventing a new cake from building. When an attempt was made to remove the discharged or the barren cake by scraping, much difficulty was experienced with the tearing of the canvas. Different schemes were tried with varying success, until the plan of holding the filter cloth in place by a wire winding was adopted. This method of construction solved the last difficult mechanical problem encountered. This experimental work was also done at the North Star mill. Early in 1908 four filters with drums 10 ft. diam. and 7 ft.

long were built for the North Star cyanide plants. To make room for these machines, three of the agitators described above were removed, and the capacity of the remaining three increased by raising the sides 6 ft.; making the total height 20 ft. The central tube was not lengthened, and the spreader board was retained, as it gives a better distribution of the slime, and throws the heavy sand to the outer edge of the tank.

The accompanying illustrations show the six original agitators, three of which are still operating at the Central mill of the North Star Mines Co. The filters first installed are also working, not a single change in construction having been made. Further descriptions would lead to the discussion of present-day devices, so that the notes are submitted as an indication of a few of the difficulties encountered in the metallurgical advances here recorded. Through the courtesy of those responsible for the work described in this article, I have been permitted to examine correspondence and drawings covering the erection of the apparatus here illustrated and described."—J. W. SWAREN, *Mining and Scientific Press*, Sept. 30, 1911, p. 409. (H. A. W.)

CRITICAL MOISTURE IN TUBE MILL FEED.—"The thickest discharge that I have been able to obtain from a cone fitted with a suitable diaphragm contained 28% moisture. Now, battery-sand containing 28% moisture is perfectly fluid, and will run quite easily down a 25% grade; but slime, 80% of which will pass a 200 mesh, containing 28 or 30% moisture, is 'cake,' as no one will need to be reminded who relies on this property for the satisfactory operation of leaf-filters.

One cannot feed a 22 ft. or 24 ft. tube mill with about 70 or 80 tons of sand per day at 30% moisture without producing a discharge which is noticeably viscous, the loss in fluidity being indicated by a rapid rise in the power necessary to drive the mill. If the feed is increased to 200 or more tons per day (I am speaking of tough quartz), the discharge of the mill is still quite fluid at 34% or 35% moisture, and it becomes advantageous to increase the time of passage of the pulp through the mill by reducing the dilution to a point depending on other circumstances.

At present, my experience suggests that, within wide limits, the best results should be obtained by keeping constant the quantity of liquid added, so that the dilution falls and rises as the tonnage rises and falls. One limit is supplied by the mill in which the tonnage has been reduced until the percentage of moisture has risen above 40%, and the other is suggested by the state of the mill referred to by Mr. Dowling, which could not crush 400 tons per day at a low dilution, as the stream of fluid was not sufficiently strong to wash out the fine sand accumulated in the mill. The latter is a condition of which I have no experience. I have intended for some time to investigate this question further, and shall look forward to the publication of the results of the series of experiments referred to by Mr. Bell. If these are not already concluded, I would suggest that the results be reduced on a basis that shall assign 'metallurgical value' to different pulps according to the area of surface exposed. It seems to me that the calculation of relative mechanical efficiencies from tables of 'Mechanical Equivalents' brings the fine-grinding machines, such as the tube mill, into a discredit that is not entirely merited. Granted that any system one may adopt for the calculation of the mechanical values of pulps will show a rapidly decreasing mechanical efficiency as the material treated becomes finer, I wish, nevertheless, to deprecate the habit of considering a metallurgical device as though it were a power-station. The series of mechanical equivalents calculated by Mr. Stadler tempts one to its use, as there seems only small scope for error in the value it assigns to the finest sizes 'including washed slime.' If, however, we tabulate, side by side with Mr. Stadler's equivalents, the relative area of surface exposed per unit of weight, we find an enormous increase in the value to be given to the very sizes in which the rise of mechanical value has become so slow. This has been sufficient to convince me of the futile nature of Mr. Stadler's method of comparison.

Although we are not yet able to make sizing-tests on pulp finer than 200-mesh, there are many ores in the world whose successful treatment depends on the presence of a proportion of 300, 400, or even 1,000-mesh material in the pulp as treated. Any grinding machine that produces such material in large quantities will be given credit for it in any fair method of comparing efficiencies.

Convinced that the function of the diaphragm was merely that of a mixer, I abandoned the type of plate developed by Mr. Caldecott some time ago and now use a round plate with holes drilled in it each slightly smaller than the size of the discharge to be used. For instance, for a 6 ft. cone, I use a plate of 14 in, diameter, with 12 holes each 1¼ in. diameter. Eight inches above this comes a solid plate rivetted to angle-irons, so as to leave an annular space of 11½ in. With such a cone, it is quite easy to get a thick and perfectly steady discharge, and yet keep the bed of sand about a foot below the top of the cone, thereby improving the classification effected."—A. QUARTANO, *The Mining Magazine*, Sept. 1911, p. 224. (A. R.)

ALLOYS: NEW CAPABILITIES OF THE UNCOMMON METALS.—This is not intended as a paper describing accomplishments, but is aimed at contributing, if possible, to increase the interest of experimenters in new alloys. I want to call attention to the possibilities in the way of useful discoveries. It frequently happens that one manufacturing company produces a new metal or alloy for some particular use, which, owing to lack of general study, may remain their sole use for a long period of time. Probably those who produce the special cerium lanthanum-iron alloy used in the various types of friction cigar lighters have not been able to study very thoroughly the application of this alloy to other uses. So also, those who use the very modern metals, tantalum, tungsten and molybdenum, have been engrossed with applications to a single field and have hardly been able to look carefully into others. It is often through the interchange of information across the gap between quite remote fields that useful advances are made.

Probably few realize the rapidity with which new metals are coming into use, particularly in alloys. There are, in all, approximately 50 metallic elements, though most of our important industries employ but a few of them. Some of these, in the metallic state, have market prices which are not yet controlled by the cost of production nor by the infrequency of occurrence, but rather by the lack of development of a utility. Beginning with gold, which we may assume is the one element whose exchange value depends upon its commonness in nature plus its cost of production, and passing over iron, copper, lead and zinc, whose values may be said to be well fixed by occurrence and costs of production, we soon reach other metals, for which a new demand might well greatly reduce the cost. Among these are many whose ores occur in abundance. In the case of this type of element the interest attached to research work is doubly great.

It is highly improbable that the cost of copper will ever be greatly changed by the discovery of new uses. This is because the world's supply of the ore is pretty well known, the demands are high and the costs of production of metal from ore have been so studied that further reduction will probably only be of about what I like to call the second order of magnitude. This was not true of the metal aluminium a few years ago, and it is still possible that considerably wider uses and reduction of production costs may be developed in its future. There is apparently a much wider divergence between the occurrence of aluminium in nature and its price in metallic state than in the case of copper. In case of aluminium, only selected and purified ores are used at present, while other compounds of it occur everywhere in nature. On the other hand, in the case of copper, ores containing even less than 2% of copper are worked for the metal. Aluminium may thus still be considered in the transition state, a state long ago passed by copper and iron and not reached by some of the metals considered below. We have all been witnesses of the interesting advance of aluminium. From 1860-74 its properties were becoming generally known. The inefficient process by which it was then reduced from its ores made it impossible to sell the metal below $10 per pound. Advance was then made so that in the 80's the price was about $5. There was not a great demand for it at this price. In the year 1907 something like 26,000,000 lb. of this metal were made in America alone. The price is now about 20 cents per pound.

The element calcium, which a few years ago was listed only as museum specimens and at several dollars per gram, was sold in 1908 at $1·50 per pound, and could certainly be sold for a small part of this price if a greater use could be found for the metal. Its ores are everywhere and it is only awaiting commercial use. It slowly decomposes water, giving hydrogen, and it differs from the alkali metals in producing such a feeble alkaline solution that it is generally harmless. It ought to serve as a good deoxidizer and should be a very cheap metal. It is not fair to relegate it to a list of useless metals. History of the metallic arts points to there being no such list.

Thallium is an element quite similar to lead, but probably possessing some property which will some day warrant its exploitation. It is softer and heavier, and could be obtained in quantity if a demand were created.

The elements chromium, molybdenum, tungsten and tantalum, the three later now obtainable in wire-form, are tempting elements to study in mixtures with others. Who knows the useful properties of a chromium bronze, for example?

Tellurium has long been an apparently useless metal and any market price is fictitious, as there is but little isolated in metallic state. It is not necessary that a great use, such as a substitute for zinc in brass, should be found for it. Our industries are so great that if a pound of tellurium added to the ton of aluminium was of benefit to the latter, the production of the necessary tellurium would be a real industry.

Consider cobalt a moment. The world's rate of supply of ore has been greatly augmented. It may take time to actually realize a greatly reduced cost of metallic cobalt, but we ought, notwithstanding, to realize it when uses have been developed. Our natural impulse in such a case is to try direct substitution of one metal for another in some well developed use. Cobalt, for example, might replace nickel in most uses when the cost fell below that of nickel, but this is a second-order use. A first order use would be the supplying of a want which every metal previously supplied distinctly less perfectly.

In this connection, an interesting alloy of cobalt and chromium has just been described by Elwood Haynes in the October number of the *Journal of Industrial and Chemical Engineering*, and it is altogether probable that technical use will soon be made of it.

Many tons of metals are annually consumed as resistance wire for electrical purposes. At one time iron was the element most used. German silver replaced it in some cases where a lower temperature coefficient was needed and the increased cost was permissible. Now there are a dozen or more special alloys for this particular electrical use. The new ones have far out-classed the old in most of those properties for which the electrical engineer uses them. In such alloys nickel, chromium, manganese and others are now used by the ton.

Silicon, which in 1900 was a curiosity and sold for 40 cts. per gram, is now a necessary component of special iron alloys and of high-grade transformer iron, and the world uses thousands of tons of the alloy annually. Silicon is now sold at about 5 cts. per pound. The use of this element in other alloys is still quite limited.

Boron, still a quite expensive material in the elementary state, is coming into commercial use in assisting in the making of solid copper castings of high electrical conductivity.

Vanadium seems to be a young wonder-working metal. Its use has increased very rapidly in the past few years, but the quantities consumed are not known to us. As several companies are producing the iron alloy, it is safe to assume that it is being sold by the ton. The price for the metal in the alloy is not far from $5 per pound.

Cadmium is a beautiful metal in many respects, and it is certainly awaiting use. It is whiter and less crystalline than zinc, and doubtless the high price of nearly a dollar a pound keeps practical workers from trying it in their experiments. It should be produced as cheaply as aluminium, if there were a good demand for it.

Titanium is an element long subject to neglect. It is a high-melting ductile white metal, which at present is only separable from its ores at high cost. It exists in many cheap ores widely distributed in nature. It is now apparently coming into use in steel manufacture, particularly for railroad rails, and for this purpose it is fortunately unnecessary to isolate the pure titanium from its ores, an iron titanium alloy being produced directly. What will happen when the pure element has been tried in special fields can only be surmised.

If one omits the common alloys, brass, bronze, solder, etc., and considers only possible alloys of two metals, and still confines himself to twenty of the common metals, like vanadium, manganese, chromium, boron, etc., he is interested at once to recognize that there must be one hundred and ninety different pairs or binary alloys. When, in addition, the effect of varying proportions in these alloys is considered, it becomes evident that the field of alloy-research is truly a large one. Many of the alloys apparently still unstudied are those which melt at extremely high temperatures.

The time is always ripe for the study of new alloys with new tensile strengths, elasticities, colours and wearing powers. The automobile and the aeroplane have forced the aluminium and iron alloys to make rapid strides, and it is natural that we should want to inventory our possibilities. The physical chemist has started along the way of a systematic co-ordination of certain properties of binary, and in a few cases tertiary alloys. He has shown how to plot a few freezing points of 2-metal and 3-metal mixtures and to construct therefrom curves showing not only all possible melting points of those metals, but what may be expected in the way of segregation and structure and such effects as caused by annealing or quenching.

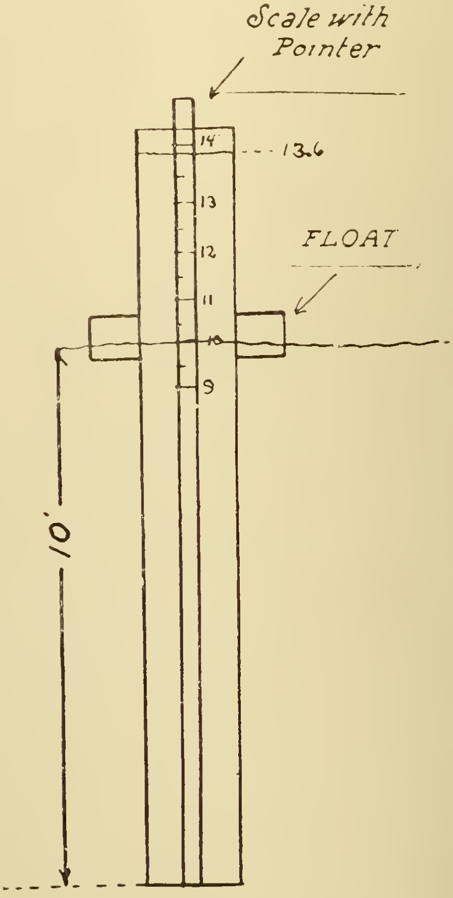
He has found that there is a solubility of metals in one another which varies just about as the solubility of substances in water varies. Metals may be melted together and well mixed, but the quality and permanency of the mixture is determined by just such solubility laws as control ordinary solutions. We know that in some cases well-mixed melted metals will separate into two layers if allowed to remain even a few moments in molten condition at low temperatures. Such a pair are zinc and lead. The two separated layers contain both metals, no matter what the temperature, but the quantitative composition depends on the temperature.

The other extreme of metal solubility is found in such a case as zinc-cadmium, the two components going into solution in all proportions and remaining in solution at all temperatures.

Man first used the metals as he found them; then, as he reduced them from the ores, and finally, when specific requirements became more and more exacting, he not only brought into use previously unused metals, but also greatly modified the old familiar ones. For a harder iron he used steel, a carbon alloy; for a harder steel, or one capable of cutting iron more readily, he added tungsten, nickel, chromium, and other metals. For permanent magnets molybdenum was added; for high electrical resistance nickel, chromium, etc., were added; for low electrical hysteresis, silicon and aluminium were added; for toughness in springs a little vanadium was used, and for wearing qualities titanium is now introduced. These are only a few of the successful alloying experiments with iron. They will probably be repeated with other metals, such as copper, zinc and aluminium, where the cost of the base metal is is not high. On the other hand, the study of those metals which have not yet advanced to a stage where first order cost reduction is possible, is equally interesting."—W. R. Whitney, *Mining and Engineering World*, Sept. 30, 1911, p. 643. (A. R.)

Measuring Specific Gravity in Agitators.

"Take a piece of 3 or 4 in. pipe some 14 ft. long. Sink it in the agitating-vat so that the lower end of the pipe will be exactly 10 ft. below the surface of the pulp. Measure the height of the solution inside the pipe (in feet). This divided by ten will give the specific gravity of the pulp. For example, suppose that the lower end of the pipe is exactly 10 ft. below the surface of the pulp and that the solution in the pipe stands 3·6 ft. above the surface of the pulp, or 13·6 ft. above the bottom of the pipe. This measurement divided by 10 gives 1·36 as the specific gravity of the pulp. If it be desired to refine the measurements, the scale can be calibrated according to the specific gravity of the cyanide solution instead of feet and tenths. This is based on the fact that the inside column must balance the outside column. The weight of the inside column is the area by height by weight of a cubic foot of solution. This must balance the weight of the outside column, which is area by height by weight of a cubic foot of solution times the specific gravity of the pulp. Eliminating the equal factors in the above and having the pipe 10 ft. deep, the matter is reduced to the height of the solution in the pipe, measured in feet, divided by 10, this giving the specific gravity of the pulp. The pipe could be arranged on a float so that the bottom of the pipe would be 10 ft. below the level of the pulp in the vat. A small float with a pointer could be placed inside the pipe. The whole arrangement could be placed inside the vat and left floating around. The pointer would indicate the specific gravity at any time. This would be especially advantageous in continuous agitation, one specific gravity tester in each tank. If the ore foams badly a shield 12 to 18 in. diameter placed around the pipe will allow the operator to see the surface of the pulp in the tank." [Note by Editorial Committee.—It is evident that some form of filter must be used to cover bottom of tube.]—C. F. Spaulding, *Mining and Scientific Press*, Sept. 16, 1911, p. 358. (H. A. W.)



SPECIFIC GRAVITY TESTER.

Slime Filters.—"Slime, from the view point of the chemist, is a colloidal hydrate, while the term, to the metallurgist, expresses a condition or a state of fineness. Material that was at one time considered unleachable and impermeable, can now be readily filtered, and slime is now purposely made because the very fine grinding facilitates solution of values in gold and silver in certain chemicals, such as KCN. As a rule, the metallurgist classes as slime all material that passes a certain sized sieve, usually from 100 to 200 mesh, while 150 is now most commonly recognized. This limit gives in practice the best results, when the two economic factors—cost of reducing the size of particles and time required for solution of the values—are taken into account.

Until recent years slime constituted the bugbear of the metallurgist and millman, and their aim in crushing was to keep its production to a minimum, on account of mechanical difficulties involved in leaching out the values from such products. Improved methods of agitation and filtration have recently, however, rendered the successful extraction of values possible. Nowadays, when the metal values in gold or silver ores are finely disseminated, the tendency is to do away with sand treatment, and to "slime" everything. This is rendered imperative on account of the necessity of separating each particle of metal from the particles of gangue enclosing it.

In order to make the slime amenable to percolation, it is necessary to suppress the formation of colloids as much as possible. The formation of these can be greatly hindered by grinding in the presence of such soluble compounds as will cause the colloid maker to coagulate on the surface of the mineral grains. Such compounds are lime and chlorides and sulphates of calcium and magnesium. The action of lime in assisting the settling of slime has long been made use of in cyanide practice.

The mechanical devices for filtering slime may be conveniently divided into two main classes, *i.e.*, by the application of a plus or a minus pressure. The machines using pressure filtration appear to be all intermittent in action, while on the other hand filters of the vacuum or suction type may be either intermittent or continuous in their action. In the latter case, no stoppage is necessary for discharging, as formation and discharge go on simultaneously in different parts of the machine. A further subdivision might be made, having reference to the material used as a filtering medium, whether of cloth or sand, or, as in the Just process, a silica sponge.

Pressure Filters.—To this class belongs the ordinary filter press, so largely used in West Australia and other closed-vessel filters, such as the Burt, Kelly, Blaisdell and Moore.

It is scarcely necessary to here describe the ordinary filter press; but a brief description of an improved form of it, the Merrill automatic discharge press, will not be out of place. By this improvement, it is not necessary to take the press apart for discharging. It may therefore do much toward preventing the relegation of filter presses to the background, in favour of filters of the vacuum type.

Merrill Press at Homestake.—At the Homestake, where these presses were first installed, there are 26 26-ton presses in operation, handling 1,600 tons of slime daily. The slime averages 80 cents per ton in value, and it is treated at a cost of 32 cents, the extraction being 90%. The pulp passes to the presses through a pipe-main at a pressure of 30 lbs. Leaching takes place within the press, the slime being aerated between each treatment with solution, and the whole operation, exclusive of filling and discharging, occupies about 6 hours.

The low value of the slime entirely prohibited agitation and filtration in separate vessels, and the cost of operating the old type of filter press was prohibitive. The filter press being admirably adapted for carrying out treatment and filtration in one operation, it was necessary to devise mechanical means for reducing the cost of operation, and the Merrill press was the outcome. When the press is filled with slime air is admitted at a pressure of 26 lb. This removes excess of moisture, and aerates the slime cake. The first aeration occupies about 2 hours, and is succeeded by treatment with strong solution (·12 to ·14 % KCN) draining, aeration, strong solution, draining, weak solution (0·07 to 0·075 KCN) draining, aeration, second weak solution, draining, wash, discharge. The complete cycle takes about 9 hours. By an arrangement of valves, the air, water, strong or weak solution, may be admitted from either side and drawn out at opposite sides. The slime cake is loosened by a water jet within each frame and sluiced out through a passage at the bottom and away by a launder.

The consumption of cyanide on the Homestake plant is 0·3 lb. per ton of slime. The filter cloths last from 6 to 12 months. The lime, which deposits on them, is removed by pumping dilute HCl. through the press. Whether this automatically discharged press can compete under average conditions with the vacuum filter, will depend largely on local conditions, such as the site, utilization of gravity for various operations, water facilities, water supply and the permeability of the material under treatment as well as the rapidity with which it yields up its values. At the Homestake, the conditions are extremely favourable; and, in fact, it was to meet these conditions at a low cost of operation, that the press was evolved.

The other types of filter presses consist of cylindrical vessels in which are suspended a number of filter leaves, generally of similar construction to those used in the ordinary Butters or Moore vacuum filters. In the Kelly and Burt systems, the cylinders are inclined to the horizontal, so as to admit of discharge of slime cake by gravity.

Burt Press at El Oro.—At El Oro, where the Burt filters were first introduced, each unit has 18 filter leaves, suspended in a steel plate cylinder, set at 45°, with a door at the lower end. The leaves are constructed of coconut matting, and heavy canvas filter cloth around a pear-shaped pipe frame in a somewhat similar manner to the vacuum types of filter leaves.

Each leaf is connected through the top of the cylinder shell, with a 4 in. pipe, which serves as a solution header. Each filter leaf has its own petcock and valve in front of the header for testing the filtrate and cutting out a leaf. The leaves have each an effective filter surface of 13·5 sq. ft. The cylinders are 20 ft. long by 42 in. in diameter, and are built of 5-16 in. steel plate. They are supported on C.I. heads, and have a door at the bottom, which is easily operated by a hydraulic cylinder suitably placed. The slime-settling vats are at an elevation, which gives an effective pressure of 60 lb. per sq. in. at the filters.

After the cylinder doors have been closed, the slime valve are opened. The cylinders fill with pulp under pressure, and the filtrate is forced through the leaves and solution pipe into a vat, from which it is pumped to the zinc room. After about 9 minutes the slime-valve is closed, and a valve is opened at the lower end of the cylinder to allow of egress of excess slime, while at the same time compressed air is admitted at upper end of cylinder, forcing out excess of slime and driving moisture out of cakes, as well as forcing the last of filtrate through to solution vat. After about 8 minutes, air and solution valves are closed, and compressed air is admitted to inside of filter leaves, and knocks off the cakes, which drop out through the door at the bottom A jet of wash water from upper end of cylinder clears out the last of the slime cake. Each cycle takes 20 minutes. The cakes are 2½ in. to 3 in. thick, carrying about 30% moisture. Slime coming from the settlers contains about 80% water. Each filter treats 120 tons to 130 tons dry slime per 24 hours.

If a water wash were given, the time would be increased to from 45 to 50 minutes. The cost is very low, 1·5 cents per ton of dry slime.

Vacuum Filters.—To the intermittent type of suction filters belong the Moore and Butters type, the latter being the type in use on the Lake View and Brownhill mines in Kalgoorlie. Time does not, however, permit of a detailed description of these filters, and it is of more interest to describe less well-known types, which have not received any application in West Australia.

To the continuous type belongs the Ridgway filter, which should also be well known in Kalgoorlie, and the Oliver filter, first introduced at the North Star mine, Grass Valley, California, and since introduced with considerable success in a number of recent American plants.

The filter consists of a drum, 11 ft. 6 in. diameter by 8 ft. wide, revolving on a horizontal axis in a box or tank. The outer surface of the drum is divided into sections, and covered with the filtering medium. The drum is partly submerged in the pulp to be filtered (kept from settling by air agitation), and, as it revolves, a vacuum applied to the section builds up a cake of slime on the surface. As the sections leave the pulp, they are dried by the vacuum, and then a water wash, in the form of a drip, is applied, and drawn through the slime by the vacuum. Just before the section enters the pulp again, air, under pressure, is admitted to the chamber, and the cake is discharged. The cycle is then repeated, the whole work being continuous and automatic. The drum revolves once every 4 minutes. The cost of operation in Mexico, where native labour is cheap, is 12·23 cents per ton of dry slime filtered.

Another type of filter which appears to have a future, makes use of a porous silica sponge as a filtering medium. This sponge is strong enough to support heavy weights, and is so evenly porous that air can be passed through it with practicallly little resistance, while the pores are so minute that no solid matter however finely divided, can pass through it or even into it. This porous diaphragm is in actual use for agitation of pulp, while the sponge has been proposed for use as a filtering medium, and its applicability has been demonstrated by experimental tests.

The simplest type of silica sponge filter now being made consists of a table, the top of which is constructed of this porous material, under which is an air-tight pan, connected with an exhaust pump. The pulp being run over the table, the solution is drawn through, leaving a dry cake of the desired thickness. Wash is applied, and the cake washed in the same way. When the cake is finally dried, the table is tilted to a vertical position, and the cake blown off by compressed air. The discharged cake contains less than 23% moisture.

Another form of filter uses the sponge on the sides of a wheel. The filter wheel revolves slowly in a tank containing the pulp. The formation of the cake takes place on the lower half of the wheel, the upper half being used for drying the cake. The solution is withdrawn by vacuum through the hub of the wheel. Instead of washing the cake on the filter, it is peeled off by a knife, and drops into a mixer, which reduces it again to pulp, with a minimum of wash water. This pulp goes through a number of such wheels in series, until it is finally washed.

The list of filtering machines in actual successful operation has by no means been exhausted in this paper, and as the progress in this branch of metallurgical science is particularly rapid, we may from time to time expect further additions to the already long list of slime filters."—B. H. MOORE, (*Science Society of West Australia*) *Mining and Engineering World*, Oct. 7, 1911, p. 693. (A. R)

ELECTROLYTIC OXYGEN IN CYANIDE SOLUTIONS.—"There are two conditions generally prevailing upon the earth—those within atmospheric influence, tending toward oxidation, and those away from atmospheric influence, tending toward reduction. Practically all mineral substances from mines of any depth are in a reducing condition. Since the cyanide process, in order to dissolve silver or gold, requires that the prevailing conditions under which it operates shall be oxidizing, and the material usually acted upon being of a reducing character, it becomes necessary to supply oxygen to the solution carrying the cyanide. This oxygen is usually supplied through the medium of dissolved air in the solution, or by the use of one of various chemical compounds, which upon combining with the solution or the ore give off a part of their oxygen. In a dissolving race between the oxygen of the air and the reducing agents from the ore, if the reducing agents predominate, cyanide will not dissolve the gold from the ore. In many cases it will dissolve some of the gold, because in a mass of irregular shape some of the gold particles might be exposed upon the outside surface of a particle of rock ; but if the solution had to penetrate through cracks, the side-walls of which were lined with reducing-agent-producing material, before the solution carrying oxygen could reach the gold it would have lost its oxidizing power. For this reason in many cases cyanide solutions will produce only an incomplete extraction of the gold or silver present.

It occurred to me that since water is composed of hydrogen and oxygen, if it be decomposed by the electric current, the hydrogen would bubble away and the oxygen would be carried by the solution. This was tried in December 1908 upon an ore carrying amorphous iron sulphides from which all the gold could not be dissolved by cyanide with simple air-agitation, no matter what the cyanide strength or how great the time. The process was tried first in an inverted bottle with the bottom cut out, the air being forced in through a glass tube in the cork to agitate the pulp. Two lead plates were inserted in the agitated pulp at the top. These plates were about 4 in. long and 0·5 in. wide, and $\frac{1}{16}$ in. thick. Through them was passed the current of an incandescent lamp, which gave about 0·25 ampere of current. The results were excellent. The value of the ore was $4 per ton. It was ground in a tube-mill so that 60% passed a 200-mesh screen. The value of the tailing, after 48 hr. agitation with air alone, was $1·25 ; but after agitation for 2·5 hr. with air and electrodes as described, the tailing was reduced to 40 cents. This typical result was verified many times, with uniformly good results.

In testing the solutions, a 2 lb. solution of cyanide is test 10. The alkali is tested on the basis of ten points over and above the alkali due to the cyanide, test 10 being a 2 lb. solution of caustic soda. The reducing agents were tested with a 1% solution of potassium permanganate, 1 cc. of which in 10 cc. of the solution, after acidulating, equals test 10, it being much easier to keep track of these solutions by simple numbers than by keeping the records in pounds per ton.

Numerous tests were made in order to determine a proper electrode. Lead was found to be the best material. Many other substances, such as carbon, worked very well, but with the alternating current, there being no consumption of the lead electrode, lead proved most satisfactory.

The following tests upon the working solution show the effect with iron and lead electrodes. All the tests were made at the same time and with the same solution, using a direct current of 0·25 ampere.

Lead electrode.	KCN.	Alkalinity.	Dbl.	Reduction.
Before	8	+1	5	6
After 4 min. ...	12	+2	0	4
After 10 min. ...	12	+3	0	3
Iron electrode :				
Before	8	+1	5	6
After 6 min. ...	$7\frac{1}{2}$	$+6\frac{1}{2}$	0	3
After 12 min. ...	6	$+7\frac{1}{2}$	0	2

(Showing destruction of the cyanide.)

There seems to be a regeneration of cyanide, and the process is certainly cheaper than any added oxidizer or even air-agitation of the solution. I found by numerous experiments that an alternating current was as effective as a direct, and had the additional advantage of giving no deposit on the electrodes at lower current-density, and with lead there was no consumption of the electrodes even where the ore-pulp flowed over the electrodes.

As finally used in practice in January 1909, a battery, supplied with alternating current, was placed in the barren sump. This battery consisted of 18 plates in series, each plate 6 in. by 6 in., with 110 volts between the two. The plates consumed 15 amperes, and produced sufficient oxidizing effect, or whatever other effect it may be, to keep the solution in condition to treat daily 40 tons of this ore. These plates, made of $\frac{1}{8}$ in. sheet lead, were built so as to form hollow rectangles in section, the rectangle being 6 in. high, 6 in. long, and 1·25 in. wide inside. The two ends were lapped at the top and holes punched. The plate was bolted to a paraffined plank 1 in. by 6 in. on the top side ; 18 of these plates were connected in series. The distance between any two plates was $\frac{1}{8}$ in., and, of course, the current would travel principally across the $\frac{1}{8}$ in. gap, instead of around the $1\frac{3}{8}$ in. gap, from plate to plate. Lead wires were used from the surface of the solution down to the plates. The ore was ground in the tube-mills so that 60% would pass a 200 mesh sieve. Previous to using the batteries in the sump, the extraction in the tube-mill was 20% during grinding ; after the batteries were used, the extraction in the tube-mill was 75%. The effect of the batteries seemed to build up in the solution gradually and to lose from the solution gradually when the operation of the batteries was discontinued.

During two months in the fall of 1910 the mill was working coarsely-ground, partly-oxidized ore carrying considerable sulphide. The water at the hydro-electric plant was low, and the use of the batteries was discontinued because the mill was driven by steam, and no arrangement had been made to supply alternating current from any but the hydro-electric plant. During this time the tailing on $4 ore went up to $1.25 per ton, and immediately after the rains gave sufficient water to drive the hydro-electric plant, the tailing assays diminished until 20 cent. per ton was reached on identically the same ore with the same head-assay ; moreover, the reducing agents dropped from 16 to 4. The time occupied in getting the working solution up to this condition was two weeks. I consider that this process owes its value almost entirely to the presence of oxygen due to electrolysis, putting the solution ahead in the race for the reducing agents and causing the gold and silver to dissolve in spite of the reducing agents. However, it does not stop the reducing agents from dissolving also, and although it produces solution of the gold in spite of the reducing agents, it does not help precipitation, and if the reducing agents are not decomposed by the batteries—and all of them are not—they build up in the solution rapidly to a point where zinc-shavings will not precipitate the gold.

Of course, in practice the cyanide solution contains reducing agents of many kinds. The electrolytic action seems to reduce the influence of some, but not all of them. For example, I experimented on some highly-graphitic ore, and whether the normally-poor extraction was due entirely to the graphite or not, I do not know ; but the solution, after electrolyzing, gave a very much better extraction than before electrolyzing. The action seems to decompose the sulphocyanides and the soluble sulphides, but not the alkaline sulphides and all of the many others always present. A test on the electrolyzed solution 18 months after the batteries were installed showed :

	KCN.	Alkalinity.	Dbl.	Reduction.
Before	8	1	0	15
After 10 min. electrolysis	8	1	0	15

showing that the solution remained practically the same, or was electrolyzed as much as was necessary. However, testing some of this same solution further by placing a piece of gold leaf upon its surface, and allowing it to float, the gold leaf was dissolved in 71 min. on the working solution and in 50 min. on the re electrolyzed solution, showing that the additional electrolysis, although it had no apparent effect on the solution, gave an increased dissolving rate. Grease in the ore or on the surface of the barren sump seemed to dissolve very rapidly in the treated solution and slowly in the untreated solution. The tanks were tarred on the inside and coated with black oil outside, and more or less grease was frequently floating upon the surface of the solution where this effect was noticed.

Since the installation of this process it has treated successfully at this plant 25,000 tons of ore of all kinds, oxidized, partly oxidized, and sulphides. Previous to the use of the batteries, in treating sulphide ores, the average cyanide consumption was 1 lb. per ton, in some months running as high as 1·1 lb. After the use of the batteries the average was 0·45 lb., running for some months as low as 0·23 lb. per ton of ore treated. I tried using batteries in the agitated pulp and in the solution, and found the result to be just as good if the plates were inserted in the barren sump as if inserted in the agitated pulp. The original lead plates placed in the barren sump are still there and in operation. They cost about $4 to insert originally, and were inspected after 26 months of practically continuous service, and are today just as good as when they were first put in use. I have applied for no patents on this process and do not expect to, and anyone is free to use it. It should be a cyanide-saver, an accelerator, and a general solution-purifier."—F. H. ALDRICH, JR., *Mining and Scientific Press*, Oct. 14, 1911, p. 483. (H. A. W.)

BALL-MILL PRACTICE.—"Metallurgists will agree that Kalgoorlie practice is, as far as dry crushing with ball-mills is concerned, far ahead of other districts in efficiency and low working costs.

In my opinion we have not yet had sufficient proof that wet-crushing ball-mills are as efficient as heavy gravity stamps on hard dense ores, although they show to advantage when compared with light gravity stamps and would probably compare favourably with the heavy stamp on soft friable ores. I take it that the heavy stamp is suitable for hard ores, not soft ones.

On p. 81 of Warnford Lock's 'Gold Milling' the author states that a Krupp mill had a greater output than a 10-stamp battery, but does not give details.

The following is an instance of a No. 5 Krupp wet-crushing ball-mill working on hard dense quartz in comparison with light gravity stamps. In the third column I have put the output of the Simmer Deep-Jupiter mill, having taken the data from Caldecott's paper on heavy gravity stamps in Vol. xix. of the Transactions of the Institution of Mining and Metallurgy.

	Ball-Mill.	Light Gravity Stamps.	Simmer Deep-Jupiter Heavy Stamp.
Weight of balls or stamps	1,000 kilo 2,200 lb.	10 of 850 lb 5 of 750 lb	1,600 lb.
Drop of inches ...	4	6	8
Screen in aperature ..	0·0312	0·0312	0·028
Revolutions or drops per minute ...	24	102	96
Tonnage per 24 hours	30	30	8·31
Grading:			
On 30 mesh %	33·4	—	—
,, 40 ,, ,,	11·8	—	—
,, 60 ,, ,,	19·9	—	31·7
,, 80 ,, ,,	7·8	—	—
,, 90 ,, ,,	—	—	16·5
Through 90 ,, ,,	—	—	51·8
On 100 ,, ,,	1·0	—	—
,, 150 ,, ,,	0·6	—	—
Through 150 ,, ,,	25·5	—	—

I should not like to contract to erect a No. 5 ball-mill in 24 hours with 28 men, for to my mind the building of a ball-mill is not so easy as the author would lead one to believe. He reminds me of the miner's wife whose husband was grumbling about going to work on the first shift of the week at night. The good lady remarked: "Why there is only tonight, to-morrow night, and the night after that, and three more nights, and you are finished."—DONALD F. FOSTER.—*Mining Magazine*, October, 1911, p. 304 (A. R.)

CUPELLATION.—"A paper giving details of experiments carried out to determine the relative value of the best known brands of cupels regarding silver loss, conditions of surface, etc., after cupellation. The conclusions arrived at are:—'Under similar conditions, with about 10 gm. of lead and 40 mgm. of silver, the average percentage losses in different cupels were as follows:—

	Silver Loss Per Cent.
Morganite	1·99
Cassite	3·09
Brownite	2·89
Bone Ash	2·36
Cement	3·38
Equal parts cement and bone ash	2·95

It was further determined that cupels made of different grades and sizes of bone ash give the same percentage loss of silver within the limits of experimental error." RAYMOND C. BENNER and MINER L. DARTMANN.—*Journal Industrial and Engineering Chemistry*, No. 11, 1911, pp. 805-7. (J. G.)

MINING.

NORTH STAR MINING METHODS.—"*Timbering in the Stopes.*—The method of mining is overhand stoping with stulls for supporting the roof, and subsequent filling of the stopes with waste from the vein. The novelty in mining is getting the ore out of these narrow stopes, about 2½ ft. wide in the upper workings and four feet in the lower levels. It is a problem in mining quite similar to that facing the engineers of South Africa, but one solved in quite a different way. As in South Africa, the level interval has been increased; first it was 10 fathoms, later 100 feet, and finally, below the 4,000 ft. level, 300 ft., but with this greater interval intermediate levels have to be driven on account of the irregularities of the deposit. With an incline shaft it will probably be found cheaper to maintain the 100 ft. interval and give up the intermediate levels.

The diabase and the granodiorite stand well, but are costly to drive in, as neither will chip in drilling holes but must be pulverized. In the stopes, the stulls are put in at intervals of six feet, but in few places do they take any weight to speak of. In a few wide parts of the vein, improvised sets are put in, but in general the posts are not over four feet long. As little short timber is used about the mines these posts are put in without any head boards, but are sharpened at the bottom; otherwise the initial creep of the ground would break them. When the weight does come the bottom of the post burrs, and by the time that the creep has ceased the top of the post has been compressed, so that there is an even pressure throughout its section. When the posts do fail, they break in the middle.

The posts are sawed from skinned logs of second growth spruce and pine, cut near by and bought by the company, delivered at the mine, for 3c. per running foot for pine and 6c. for spruce logs that scale six inches at the small end. It is because the timber is cheap that the company uses so many posts in the stopes; few take any appreciable weight until late in the life of the stope, for the initial creep seems to be long delayed, and probably in many stopes does not come until long after they have been filled and abandoned. The posts serve mainly to assure the miners that there is no weight coming on the roof. No doubt this could be done quite as well by using fewer posts, or possibly an economy could be effected by using head boards and knocking out posts as the filling progressed, but some means of indicating the condition of the roof is probably necessary, as it appears probable that the diabase yields suddenly when overloaded and comes without much warning. Before stoping has progressed far, a raise is put up to the level above, in order to provide good ventilation.

Piston and Hammer Drills used.—The ore is drilled either by Torpedo, 2¼ in. piston drills or

Waugh air hammers, but gradually the air hammer drills are displacing the piston machines in all but the flattest stopes. The piston drills bore about 30 ft. of hole in an eight-hour shift, and the air hammer drills probably average somewhat more, although the flatness at which the hole has to be bored impedes them greatly. The Torpedo drills are used in driving all drifts, and in shaft sinking, although they are one man machines. When the rock is especially hard, two of these one man drills are used in a drift, one on each side of the bar. The Torpedo drills have the inlet and discharge ports at the ends of the cylinder, so there is no wire drawing, and the full pressure of the air is working on the piston without any appreciable back pressure to resist it; the drill, therefore, bores rapidly, and as it has a tappet-operated value, positive in its action, a front head is never broken; in fact, the only breakage which does occur is in the occasional giving way of a side rod. The miners drill all the time on both shifts, as the blasting is done by a special crew. Each machine will average about six 5 ft. holes per shift, and the holes drilled by a machine will break, on the average, 5½ tons of ore.

North Star Mining Costs.

	Cost per Ton of Ore.	
Surface.	North Star Shaft	Central Shaft.
Engineers and mechanics ...	$0·09	$0·09
Blacksmiths	0·19	0·13
Carpenters	0·01	0·01
Brakemen	0·15	0·05
Landers	0·15	0·02
Electricians	0·01	0·01
Sundry	0·03	0·05
Total surface expense per ton of ore raised	$0·63	$0·36
Total surface expense per ton of ore and waste raised ...	0·37	0·30
Underground		
Officials	$0·17	$0·10
Foremen	0·08	0·03
Bosses	0·36	0·19
Pumpmen	0·14	0·07
Timbermen	0·20	0·10
Stoping	1·22	0·61
Drifting	0·21	0·14
Raising	0·12	0·07
Station cutting	..	0·06
Station tenders	...	0·10
Carmen	0·77	0·22
Shovellers	0·33	0·49
Tool boys...	0·07	0·06
Sundry expense	0·02	0·03
Total underground cost per ton of ore raised	$3·69	$2·27
Total underground cost per ton of ore and waste raised ...	2·18	1·84
Total cost per ton of ore raised	4·32	2·63
Total cost per ton of ore and waste raised	2·25	2·14

The Blasting Crew.—Special men are employed to do the blasting, partly because there will be less stealing of specimen rock by the miners, but mainly because it is much more economical, as ordinarily a miner will take an hour to take down his machine, load his holes and blast. The blasters come on shift at 10.30 p.m. and work eight hours, the first two hours being taken in capping the fuse, making up primers and getting the powder to the different levels where it will be wanted. The morning shift comes on at seven and works until 3.30 in the afternoon, while the afternoon shift comes on at four and works until 12.30 at night.

In order to guard against the stealing of rich ore, specimen bosses go into the mine the first thing in the morning and proceed immediately to the faces, where, from the condition of the face before blasting it seemed probable, the night before, that specimen rock would be broken. The rich pieces of ore are taken to boxes on the different levels and locked up until they can be taken to the surface. To guard further against stealing, while coming off shifts, the miners are made to change in the company's change house.

Handling Ore in Flat Stopes.—The problem at the North Star mine is to get the ore cheaply out of the stopes. Many ways have been tried since the mines were first worked. The first-used way was to shovel the ore from man to man down the stopes, each miner having to shovel to the man below him, the ore that he broke in the shift. As a result, the miners at the bottom of a breast had to shovel all the ore broken by the miners above, so that soon the work was lagging at the bottom and the broken ore from above was lying where the miners below wanted to drill. This trouble was obviated by putting shovellers in the stopes, but it was expensive to work the ore down to the level by shovelling it, as in the low places a man could not throw the ore on an average of more than four or five feet down the stope. Gravity planes and loading into cars in the stopes were next tried, and after a time given up, while shaking chutes were experimented with, but the chutes did not work well and were soon scrapped.

The practice of loading the ore into cars and the use of gravity planes were then resumed, and in time the "go-devil" brake arrangement was devised to aid in the lowering of the cars down the plane. The go-devil is a three-wheeled block for lowering cars that is equipped with a braking device working on the inside faces of all 3 wheels around which the rope passes. This go-devil can be fastened to a stull by a drill or bolt, while the short lever arm is lengthened in the stope by putting a piece of pipe over it. With the first go devil brakes, ton cars were used, running down the stopes on rail tracks and dumping into a pocket at the bottom, holding four or five tons. Side tracks were turned off horizontally near the face so that the cars could be run to the broken ore. Of course, a horizontal face was carried in the stopes where the go-devils were in use, but it was expensive to move these lateral tracks, as the face was worked away from them, so that much of the ore had to be shovelled twice to get it into the go-devil cars.

The first experiment tried to overcome this difficulty was the use of a derrick, consisting of a long arm clamped loosely to a machine bar above a tightly fastened collar. On the end of the bar was a bucket into which the ore was shovelled. On the bucket was tied a rope used for tipping the bucket when it swung over the go-devil car, and also for pulling the bucket and arm back to the face again. Unfortunately the derrick proved almost as fascinating as a steam shovel; the men wasted too much time in watching it dump, to do much shovelling, so the derrick was sent to the scrap heap.

Roller Wheelbarrows in Use.—Next a three-wheel barrow was devised, having loose handle arms, fitted with brake shoes so that by pushing down on the handle the shoveller could brake the wheels when going down hill. This barrow required a three-plank track which took time to lay and was hard to keep even enough so that the barrows would run on it without tipping. Moreover, the loose arms made it impossible to tip the barrow until the handle was raised quite high in the air—rather difficult to do in a low vein—so in time the three-wheel barrow went the way of the other devices. About two years ago the idea of using small low trucks mounted on rollers instead of wheels occurred to Richard Barry, the mine foreman. This idea has proved to be a most excellent one, for by means of the small trucks the cost of getting the ore out of the stope has been reduced several cents per ton.

This small truck, locally called "a roller devil," looks like a toy. Indeed when one first sees a man working with one of them he thinks of the pictures of Mansfield Mines, and the little German boys with their wagons hauling the ore out of the stopes; but by means of these trucks, the go-devil brake and the inclined planes, a man is able to shovel and run down to the ore pocket at the bottom of the stope an average of six tons of ore per day, while he also shovels and sorts out another four tons of waste.

In blasting, 40% dynamite is used. The broken ore is not cleaned up carefully after each blast; the cleaning of the stope is left until the end, when the floor in the richer stopes is being swept. The waste that is broken with the ore when it is needed in another part of the mine, is loaded into ton-cars and lowered by go-devils down a track into the other stope. Generally the floor in one part of the stope is cleaned carefully and the waste filled in roughly to make a pack wall to support the roof, in the same stopes nearby where it is broken.

The Building of Pack Walls.—Along the level these pack walls are built carefully and rival in excellence the dry walls built underground by Mexicans. When a high wall has to be built, as in places where the roof is 15 or 16 ft. high, wood is not used to tie in the walls, as the wood would rot; instead pieces of worn, go-devil rope are used. These are anchored in the planks by piling large rocks on top of them after they have been twisted around some rocks, then the rest of the rope is brought forward and laid in the centre of the front wall of the pack, and the next tier of stones laid on top of it. As the holding power of the rocks on the rope is great, it is practically impossible, when the roof settles, for the pressure to push out the wall, when it is so tied to the centre of the pack.

The timbering is done by special timbermen, while the tools are gathered by nippers. In the stopes about three tons of ore are mined per man working in them, while an average of 1·6 tons is mined per man employed underground. These figures refer to tons of ore hoisted; not to tons of ore and waste broken. Most of the tramming is done by mules, and side-dumping cars having a capacity of one ton are used throughout the mine. These side-dumping cars are much easier to dump and their use does away with the temptation to put up raises so as to hold out under the tracks on the level above.

The Cost of Mining.—Before mentioning costs, probably it will be well to give the rate of wages in the district. Head timber men are paid $3·25 per shift, timbermen $3, skip men $3, head blasters $3·50, blasters $3, miners $3, carmen $2·50, shovellers $2·25, and nippers $2. The men can get board and rooms for from $26 to $30 per month.

The cost of driving a 6 × 7 ft. drift along the vein is about $14 per ft., including the cost of track and air pipe, as well as all other expenses, while the cost of putting up a raise on a vein, including the cost of putting in the sill timbers and air pipes, is, owing to the flat dip, about $20 per ft. As the ground stands well, few of the drifts or raises have to be timbered; consequently that cost does not figure in the statements herein given.

The differences of cost in working the North Star or upper levels where the vein is $2\frac{1}{2}$ ft. wide and the central or lower levels where the ore is four feet wide are somewhat revealed by the statements of the amounts of ore and waste hoisted. During 1909 there were hoisted through the North Star shaft 21,710 tons of ore and 15,114 tons of waste, while at the Central shaft there were hoisted 69,934 tons of ore and 16,233 tons of waste.

The cost of handling the ore at the North Star shaft is greater because it is hoisted up the incline in one-ton cars and then trammed by the landers directly to the mill bins. The greater area covered by the stopes that are being worked in the North Star, where the mining is scattered, owing to the longer time that the upper workings have been furnishing ore, causes the cost of bossing to be considerably more than at the Central workings, where the stopes cover a much less area on the veins, as they are larger and are on fewer levels. The same remarks apply to difference in cost of the tramming at the two shafts. With this exception that some old stopes are being finished in the upper workings, there is no reason other than the difference in the width of the stopes for the difference in the cost of the mining at the two shafts."—CLAUDE T. RICE, *Engineering and Mining Journal*, Nov. 18, 1911, p. 991. (A. R.)

REPORT ON INVESTIGATION INTO THE COMPOSITION OF GASES CAUSED BY BLASTING IN MINES. (CORRIGENDA AND ADDENDA).—"Some of the figures given for the nitrogen content of collodion cotton in the original reports* have been found to be incorrect, and the amended results show that in seven collodion cottons examined, the nitrogen content varied from 11·27 to 12·16%. This correction necessitates alterations in the calculations for oxygen balance in the various explosives. In twenty samples of various brands of gelignite, gelatin dynamite and blasting gelatin, three were found to be theoretically underoxidised, the greatest deficiency being 1·47% of oxygen. In those which were theoretically overoxidised, the highest excess of oxygen was 6·13%. When the paper wrappers were taken into account in the calculations, eleven of the explosives were found to be underoxidised, the greatest deficiency being 3·63%. Determinations of the velocity of detonation were made, the lowest figures being 1708 metres per second for one of the gelignites and 4100 metres per second for one of the blasting gelatins. The view is advanced that in gelatinised nitroglycerin explosives containing collodion cotton sensitiveness to detonation is due to free nitroglycerin distributed throughout the mass of the explosive, in intimate contact with the main mass of the colloidal mixture of collodion cotton and nitroglycerin. It is considered possible that this nitroglycerin is very slowly absorbed into the mass of the colloid, and there exerts its power of solution upon particles of collodion cotton not completely dissolved, and that when this absorption takes place the explosive becomes less plastic and less sensitive. The general experience in Western Australia is that an explosive which, on landing from Europe, is soft and plastic will, after a lengthy storage in the magazine, become stiff and more rigid to touch, whilst the common experience is that such explosives explode only under excessive stimulus, and are very apt to leave unexploded portions in the bottoms of the bore-holes in mines. These facts, in the author's opinion would indicate that an explosive which, brought directly from the factory, contains a certain proportion of free nitroglycerin and is therefore quick and sensitive, becomes inert on storage. When an increase in inertness takes place it is further often accompanied by fall of heat test and it is suggested as an explanation of this that an extended solution of collodion cotton due to further absorption of nitroglycerin liberates small quantities of acids from the fibres of the collodion cotton which have not been extracted by washing in the factory. The heat tests immediately after manufacture are higher than those obtained when the explosive is landed in Australia, and although the voyage through the tropics has been advanced as a possible explanation of this, no physical change has been given as a possible key to the series of phenomena observed."—E. A. Mann.—Chief Inspector of Explosives in Western Australia, *Journal of Chemical Industry,* 1911, p. 1281. (J. G.)

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* *Journal of the Society of Chemical Industry*, 1911, 447.

Mine Accidents in the United Kingdom.—" The general report by the Chief Inspector of Mines on the mining industry for 1910 has made its appearance. Fatalities in the mines of the United Kingdom have fallen fairly regularly since the Institution of Government inspection in 1851, as the following quinquennial averages of fatalities underground per 1,000 up to 1905 show; 5·149, 4·628, 3·791, 3·995, 2·736, 2·709, 2·312, 2·042, 1·704, 1·473, 1·411. Since the five years ended 1905, however, a steady and very serious increase has taken the place of the improvement hitherto witnessed, and we get the following fatality rates per 1,000:—

1906	1·424
1907	1·455
1908	1·467
1909	1·621
1910	1·917

Last year was, we hope, exceptional on account of the two severe explosions which added considerably to the loss of life, but until some stringent measures are taken for dealing with the dust danger it is impossible to say that they are necessarily so. Apart, however, from the question whether 1910 will prove exceptional from the point of view of deaths from explosions, we have to face the fact that fatalities under every class of accident were heavier last year than in the five years preceding. The following are the actual figures:—

	1910.	1905-9.
Explosions	496	125
Falls of ground	625	551
Shaft Accidents	89	78
Miscellaneous (underground)	391	345
Surface	153	134

Some slight allowance must be made for the increase in numbers employed, but the lesson remains that a substantially increased number of men were killed in each category of mining accident last year. The record is one of increased carelessness the growth of which, as the previous table shows, has been manifesting itself for several years past. In terms of ultimate responsibility, therefore, an analysis of the position means increasingly careless administration of mines in this country. The underground fatality rate in Cornwall was again very heavy last year. The following is the record brought up to date from last year:—

	Cornish Mines.	All Mines
1902	1·15 ...	1·37
1903	1·45 ...	1·35
1904	2·00 ...	1·35
1905	3·70 ...	1·52
1906	3·63 ...	1·42
1907	2·39 ...	1·46
1908	2·78 ...	1·47
1909	3·34 ...	1·62
1910	3·26 ...	1·91

London Mining Journal. Dec. 9, 1911. p. 1185. (A. R.)

MISCELLANEOUS.

Oil-Mixed Portland Cement, Mortar and Concrete.—" In October 1909 I made the discovery that oil in considerable quantities could be combined with wet Portland cement paste by a simple mixing process. It was noted that when oil was added to wet cement it disappeared completely when mixed for a few moments, and, furthermore, that there was no apparent separation of the ingredients upon setting. Experiments were at once begun with various kinds of oil, to ascertain which was best adapted for the mixture. After these preliminary inquiries were completed, tests were conducted, or are now in progress, in order to investigate the properties of oil-mixed cement, mortar and concrete. The materials used consisted of Portland cement, river sand and ¾ in. crusher-run gneiss. The oils selected for use were seven varieties of fluid residual oil, varying in viscosity from 2·5 to 65 and in specific gravity from 0·893 to 0·945, and one sample of cut-back oil asphalt. The cement complied with the specifications recommended by the special committee on Concrete and Reinforced Concrete. Of the sand 17% was retained on a No. 10 screen and 96% on a No. 80. Of the stone 28% passed a ¼ in. mesh screen. The sand contained 37% voids, the stone 43%.

Time of Setting.—The addition of oil retards both the initial and final set. The effect of 10% of oil is to increase the time of initial set about 90%, and the time of final set about 60%.

Crushing Strength.—Concrete, like mortar, when mixed with oil, shows a decrease in strength, that containing 10% of oil having roughly 75% of the strength of plain concrete when stored in water. Air-cured concrete, mixed in proportions of 1 : 3 : 6 and 1 ; 3 : 5, containing 10% of oil, has 65% and 75%, respectively, of the strength of the plain concrete mixtures. At the completion of the one-year tests on compression, a marked increase in strength is shown over similar tests at six months. The next tests will be made at the expiration of two years.

Toughness.—Cylinders of concrete, 6 in. in diameter and 6 in. high, were tested in the Page impact machine under blows of a 10-kg. hammer, falling upon a 5-kg. plunger from successively increasing heights of 1 cm. The results of this test indicate that the toughness or resistance to impact of oil-mixed concrete differs but slightly from that of plain concrete.

Modulus of Elasticity.—Tests for the modulus of elasticity were made on specimens of the usual size, 8 in. in diameter and 16 in. high. The results indicated that the addition of oil to concrete has very little effect on its stiffness, the initial modulus of elasticity, with a few exceptions, showing but little change. The effect of storing in water is to increase the modulus of elasticity to more than that of air-cured specimens, and is as true with oil as without it.

Permanent Set.—In the endeavour to discover the effect of oil on the elasticity of concrete, micrometer readings of permanent set were taken after releasing the load to its initial value of 500 lb. In 1 : 3 : 5 concrete, containing as much as 10% of oil, the permanent deformation existing after the application of 500 lb. per sq. in. was less than that in the specimen containing no oil. This was true of both air-cured and water-cured specimens. The elasticity of 1 : 3 ; 6 concrete appears to suffer with the addition of oil, the permanent set of the 10% oil mixture, air-cured, being 0·00076 in. per inch of length, as against 0·00019 in. for the plain concrete specimens. The corresponding ratio appears to exist for the water-cured specimens. The permanent set of the 10% oil mixture was 0·00017 in. after the application of 500 lb. per sq. in. Here, again, the beneficial effect of storing in water is well marked, the permanent set being very small for water-cured concrete, when compared with that cured in air.

Absorption.—In order to determine the waterproofing qualities of oil-mixed concrete, compared with those of a plain mixture, absorption tests were made on 1 : 3 : 6 concrete cylinders, 6 in. in diameter and 6 in. high. The cylinders were dried to constant weight in an oven, after being cured 15 days in air. They were then completely immersed in water, and were weighed every day until they had attained constant weight. The addition of oil decreases greatly the percentage of absorption, the cylinder containing 10% of oil absorbing 1·7% of water, based on the dry weight, while 6·25% was absorbed by the cylinder containing no oil. In almost every case the maximum absorption was attained within one day after immersion, and furthermore, the rate of absorption seemed to be inversely dependent on the quantity of oil added. It will be noted that there is approximately only 0·5% difference between the absorption of the cylinders containing, respectively, 10% and 25% of oil. The inference to be drawn from this fact is that 10% of oil is nearly as effective as 25%, as far as waterproofing is concerned.

Four vessels of 1 : 3 mortar, each 2½ in. high, about ½ in. thick, and having an outside diameter of 8 in., after hardening in moist air for one week, were immersed in water to a depth of 5 cm. Vessel No. 1 was made with no admixture of oil. A damp spot appeared inside about 1 min. ofter immersion. In one hour it was completely water-soaked, the moisture having climbed by capillarity above the level of the surrounding water. Within a few days after immersion, water had penetrated the plain mortar vessel until its level inside was the same as that outside. Vessels No. 2, 3, and 4 contained, respectively, 5%, 10% and 20% of oil. After an immersion of six months, the interiors of the oil-mixed vessels are as dry as they were on the day they were placed in the water. A corresponding series of tests was made on mortar vessels, made to contain water, with like results. The plain mortar vessels leaked, and those of oil-mixed mortar were perfectly tight. These results show with certainty the waterproofing qualities of oil-mixed mortar and concrete.

Permeability.—The following tests were made on specimens of 1 : 3 mortar, 3 in. thick and 6 in. in diameter: Plain 1 : 3 mortar, at the age of 28 days, under a pressure of 30 lb. showed dampness in about ½ hr. Under a pressure of 40 lb., applied for 24 hr., the leakage amounted to 146 cc. Specimens of the same mixture, but containing 5% and 10% of oil No. 4923, remained perfectly dry for 24 hr., under a pressure of 40 lb. A ½ in. coat of 1 : 3 mortar, containing 10% of oil No. 4923, applied to the surface of a very porous concrete, was absolutely effective in preventing all leakage, under comparatively high pressure.

Bond Tests.—Tests were made to determine the relative strength in bond of oil-mixed and plain concrete. Specimens were made of 1 : 2 : 4 concrete, mixed with 23·5% of water, based on the weight of cement and dry sand, making a rather wet mix. Rods 12 in. long were embedded in the centre of cylinders 8 in. in diameter and 8 in. long, one end projecting about ½ in. through the bottom. The specimens were bedded in plaster of paris on a cast-iron plate mounted on a spherical bearing block. The test consisted in pushing the rods through the concrete, the point of failure being taken at the drop of the scale beam. Two kinds of bars were used, plain and deformed. All specimens were tested at 28 days. The results show that the effect of oil is to decrease the bond strength, the decrease depending on the quantity of oil added. It is evident that the bond between plain bars and concrete is affected so seriously by the admixture of oil that it would be extremely inadvisable to use such a combination. The bond of deformed bars is not affected so seriously. With an admixture of 10% of oil, which is as much as is necessary to use, the bond strength is 84% of that attained by the mixture containing no oil.

Service Tests.—During April and May, 1910, two bridge surfaces of oil-concrete were laid at Ridgewood, N. J. In the Borough of Richmond, New York City, during May and June, 1910, about 400 ft. of street surface was laid with different aggregates. About 300 ft. of street in Washington, D. C., was surfaced in June 1910. Numerous inspections of this work have been made, and, up to the present time, it is all in very good condition. About ½ mile of roadway, laid with 10% oil-concrete in the suburbs of Harrisburg, Pa., in the summer of 1910, is also reported to be in very good condition. A vault, 112 ft. long and 18 ft. wide, in the United States Treasury Department, was built in the autumn of 1910. The side-walls of this vault contain 10% of oil. Its roof was constructed of ordinary reinforced concrete with about 3 in. of 10% oil-concrete on top. For several months this roof was subjected to a head of water of several feet without showing any signs of leakage. Another vault, in the north end of the Treasury, on account of leakage, has never been available for storing anything of value. Its roof was treated with oil-concrete, and is now perfectly dry. Numerous floors in the sub-basement of the Treasury Department, and a floor in the Office of Public Roads, have been constructed with 10% oil-concrete, and have remained absolutely free from dampness. A tank, about 12 ft. long, 4 ft. wide, and 4½ ft. high, constructed in the Office of Public Roads during the winter of 1910, has been continually full of water, and has not shown the slightest signs of leakage.

Summary of Conclusions.—The following tentative conclusions as to the effect of the oils used in cement and concrete may be drawn from the foregoing investigations:

1. The tensile strength of 1 : 3 oil-mixed mortar differs very little from that of plain mortar, and shows a substantial gain in strength at 28 days and 6 months over that at 7 days.

2. The times of initial and final set are delayed by the addition of oil, 10% of oil increasing the time of initial set by 90%, and the time of final set by 60%.

3. The crushing strength of mortar and concrete is decreased by the addition of oil to the mix, concrete with 10% of oil having roughly 75% of the strength of plain concrete at 28 days. At the age of one year the crushing strength of 1 : 3 mortar suffers but little with the addition of oil in quantities up to 10%.

4. The toughness or resistance to impact is affected but slightly by the addition of oil in quantities up to about 10%.

5. The stiffness of oil-mixed concrete appears to differ but little from that of plain concrete.

6. The results of tests for permanent deformation do not indicate that a law is followed by oil-mixed concrete.

7. Oil-mixed mortar and concrete containing 10% of oil have very little absorption, and, under low pressures, both are waterproof.

8. Oil-mixed mortar containing 10% of oil is absolutely water-tight under pressures as high as 40 lb. per sq. in. Tests indicate that oil-mixed mortar is effective as a waterproofing agent when plastered or painted on either side of porous concrete.

9. Bond tests show the inadvisability of using plain bar reinforcement with oil-concrete mixtures. With deformed bars the bond is not weakened seriously by the addition of oil in quantities up to 10%.

A patent has already been granted to me for mixing oil with Portland cement concrete and hydraulic cements giving an alkaline reaction. All citizens of the United States are at liberty to use this process without the payment of royalties."—L. W. PAGE, *Mining and Scientific Press*, Nov. 4, 1911, p. 580. (H. A. W.)

THE DIAMOND-CUTTING AND POLISHING INDUSTRY.—"*Varieties of Diamonds.*—Not all diamonds are what they seem. There are two kinds—the real diamonds called 'brilliants' and the 'roses,' the latter of which are used to meet the popular demand for diamonds in spite of higher prices. As a matter of fact, the only real difference between 'brilliants' and 'roses' lies in the number of sides they individually possess after having been polished in one of the many diamond factories. A 'brilliant' must usually possess fifty-eight 'sides,' but a 'rose' need only possess twenty-four 'sides.' Another, though slight, difference between them is that a brilliant is pointed at both ends, while a 'rose' diamond has one end flattened and the other pointed. In the cutting of diamonds, the rough crystals are first examined, and as far as possible they are chipped into shape by tiny wedges before being actually cut and polished on the revolving wheel. The various flakes thus chipped off are themselves cut and polished to the best advantage, to form 'rose' diamonds, while the bulk of the crystal is cut into a 'brilliant.' Obviously, then, roses are cheaper and also less bright than brilliants; but their judicious use, either with brilliants or with coloured stones, enables manufacturers to produce articles which readily take the public fancy, and which, by the trade, are used to replace wholly or in parts the all-brilliant articles of the earlier times, and very effectively, too.

Leading Diamond Markets.—Formerly Paris enjoyed the monopoly of the cut diamond trade, in view of the fact that the Antwerp and Amsterdam merchants at that time sought no direct market, but were content to sell the product of their factories to the merchants of that city. Certain French houses made enormous fortunes by thus acting as middlemen, for it was to Paris that the jewellers of the entire world came for their supply. To-day all this is changed, and it is to Amsterdam and Antwerp that the buyers come as the leading diamond markets of the world. In Amsterdam the diamond-cutting industry has been extensively carried on since the fifteenth century. In one factory alone a short time ago some 400,000 diamonds were cut and polished in one year alone, and from 300 to 500 men were daily employed; so we have been informed by the American Consul there, in one of his reports.

The First Process.—The first of the three processes through which the rough uncut diamond has to pass is that of splitting the stone. About twelve men are employed in this task, which is all done by hand, without the aid of machinery. This is not so lengthy a process as might be imagined, unless the stone happens to contain a great many flaws, for, as a matter of fact, an ordinary diamond weighing about 10 carats can generally be split in fifteen minutes, more or less. The great aim is to divide the rough, uncut diamond (containing flaws) into several pure, flawless stones, and in order to do this strong pressure is exerted on the weak spots, caused by flaws, by means of a diamond-tipped tool, which is held in the right hand. In the left hand is held a little frame, into which the rough diamond now to be split has been fixed. A considerable amount of pressure has to be exerted on the weak spot, and in a few moments the diamond splits into two portions. If by chance there are found to be any more flaws in either of these sections, the process is repeated until several pure, flawless diamonds lie on the rough wooden tables of the workman ready for the second process of cutting. Only in the case of very hard, difficult stones is machinery necessary. The work of diamond-cutting is a very delicate and difficult task, very trying to the eyes, too, for some of the diamonds are so minute that four hundred of them may weigh only half a carat.

The Cutting Process.—The tools employed in diamond-cutting are very similar in appearance to those used in the first process, that of diamond-splitting. In both cases diamonds are used as knives after having been firmly fixed into the end of pear-shaped tools. When the diamond has been sufficiently cut and rounded, it is then ready for the third and last process, that of polishing. This work needs a great deal of skill and time, and is performed by machinery, a steam engine supplying the motive power for all iron discs. The process consists in giving to the rounded diamonds what is termed in technical language, their necessary 'sides.'

Polishing.—When the diamond is ready to be polished it has first to be soldered into a pear-shaped frame made of zinc, and then dipped into a preparation of oil and diamond dust before being fastened, with three others, into a strong frame placed in close proximity to an iron wheel. This little disc is then set in motion, by connecting it with the great engine, and the wheel immediately turns at the rate of 2,000 revolutions to the minute. As it revolves, the four diamonds set in the adjacent frame are gradually polished by means of the friction produced on their surface by the particles of diamond dust still adhering to the oil into which they had been previously dipped. Only one portion of a diamond can be polished at once, and consequently when this part has been dealt with, the stone has to be resoldered, so that another portion of its surface may be subjected to the friction of the wheel. The process of re-soldering takes place about twenty times in the case of every brilliant, before it can be considered to be thoroughly polished, and to have acquired all the necessary fifty-eight 'sides.' The 'rose' diamond, with its smaller complement of twenty-four 'sides,' needs to be re-soldered only about six times.

Some Statistics.—About seventy establishments in Amsterdam cut and polish diamonds. They employ more than 10,000 people, of whom about 1,700 are cleavers and cutters, and 4,700 polishers, etc., the remainder being engaged in the office and in other work. Several establishments outside the city itself will swell these numbers considerably. Workmen in the Amsterdam diamond industry are composed of five classes, in the following order, as to the amount of wages received—Cleavers, polishers, turners, cutters, and sawers. Polishers and turners receive about the same wages. Some cleavers receive as much as £25 a week; formerly even more than that at times. From this maximum wages grade downward through the other classes to 25s. and 35s. a week for sawers. Cleavers split the diamonds; cutters take off the rough and sharp edges and corners and make the general shape of the stone; polishers polish the stones and make their facets; turners turn the diamonds around in the apparatus which holds them, so that the facets can be made; sawers saw stones which cannot be cloven or which it is more profitable to saw; sometimes a stone is cleft and the parts are then sawn, but very small stones cannot be sawn."—*South African Mining Journal*, Nov. 11, 1911, p. 344. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

NOTES ON THE MINERAL DEPOSITS OF THE ANGLO-EGYPTIAN SUDAN. By STANLEY C. DUNN, A.R.S.M., F.G.S., A.I.M.M., 1911. (Edinburgh: Oliver & Boyd.) Price 1s. 6d.

"These notes constitute the first bulletin issued by the Anglo-Egyptian Sudan Geological Survey. In a compressed form the bulletin forms an excellent introduction to the history of mining in the Protectorate. After a review of the pre-existing literature of the mining history of the country, a brief account is given of the exploration work done on the various concessions in the past five or six years. The copies of the reports of the various engineers employed in prospecting the concessions over which prospecting rights were granted by the Administration were filed with the legal department, and permission has been given the author (Mr. Dunn) to take extracts and summaries from them for the purposes of his handbook. Owing to the method under which the promotion of the various enterprises and their operations were conducted, little or nothing has appeared in public of the very considerable amount of exploratory work accomplished, and there are probably a good many people interested in the Sudan in recent years by whom this report will be perused with interest if not necessarily with satisfaction. Considering the area of the Sudan, it follows that the greater proportion of the country has still to be prospected, and, consequently, the value of an official statement of the results of the work so far accomplished is self-evident."—*London Mining Journal*, Nov. 25, 1911. (A. R.)

TECHNICAL METHODS OF CHEMICAL ANALYSIS.—Edited by GEORGE LUNGE, Ph.D., and C. A. KEANE, D.S., Ph.D. Vol. II. In two parts. (London: Gurney & Jackson.) Price £3 3s.

"These two parts form a welcome addition to the two (Vol. I.) issued in 1908 and reviewed in this journal. The high character of the earlier issues has been maintained, and the list of names of those who who have collaborated with Dr. Lunge in writing the German edition, and with Dr Keane in translating it into English, and particularly in adapting it to English requirements, justifies us in stating, as is indeed proved throughout the work, that this is not a mere collection of information, but is an authoritative work from which matters of doubtful utility have, so far as possible, been excluded.

The two parts, into which the volume has been divided for convenience, cover 1,250 pages, and include a subject matter index of about 35 pages and a complete name index, with a table of contents to each number. All these bear evidence of an editorial supervision, in pleasing contrast to the neglect commonly shown in giving the finishing touches to a technical work. It is seldom that a subject-matter index can be spoken of so favourably as in the present case, where the index is not merely a means of finding what one looks for, but is largely an indication as to the general subject-matter of the work. Although practically all is of interest to the mining man and metallurgist, even including the section dealing with fertilisers, where natural phosphates and basic slags receive attention, the principal matters (from the point of view of the readers of this journal) lie in the first part, which covers a large range of tests and assays for ores, metals and alloys, and which is particularly valuable in connexion with the rare metals now becoming so increasingly important. Such a metal as vanadium, for instance, occupies more space in Lunge than in probably any other work on general assaying. Similar remarks apply to tungsten, molybdenum, etc., and to the alloys in which they and many other rare metals play so important a part. One valuable feature of the book is the indication given in many cases as to commercial requirements and market conditions in addition to a mere statement of commercial methods of analysis. The book is, in fact, a guide, not merely to the matters of testing, but to the interpretation of the results."—*London Mining Journal*, Nov. 25, 1911. (A. R.)

THE LAY-OUT, DESIGN, AND CONSTRUCTION OF CHEMICAL AND METALLURGICAL PLANTS. By OSKAR NAGEL, Ph.D. 1911. (New York: The Author.) Price 8s. 6d. net.

"At the commencement of the book, choice of site, consideration of local and natural conditions, examination of ground for foundation purposes, selection of most suitable building materials and type of building construction, as well as all the necessary preliminaries and estimates, are considered in a general way; these same points are subsequently considered in detail in reference to many actually existing plants and works. The succeeding chapters, with the aid of nearly two hundred illustrations, deal at length with modern methods, covering in separate chapters plants for steel, spelter, gold, silver, copper, ore dressing, sampling, acid. alkali, and other chemicals, also bricks, cement, solid, liquid and gaseous fuel, paint, etc. In all the different plants which are considered, the general arrangement, lay-out, and methods of construction are fully considered; emphasis is laid on saving of labour, minimum of handling and transport, convenient grouping of various sections of the plant, economy and proper distribution of power, minimum insurance risk, and easy superintendence and management. All the before-mentioned points are of vital interest to those engaged in designing and construction work, and much, or all, of the future success depends thereon. The last chapter, dealing with the science of business, is, perhaps, too short, but is of great interest and sound commonsense."—*London Mining Journal*, Dec. 2, 1911. (A. R.)

A TEXT-BOOK OF THEODOLITE SURVEYING AND LEVELLING. By JAMES PARK. (London: Charles Griffin & Co., Limited.) Price 7s. 6d.

"This is the second edition of a book which on its original appearance won a well-deserved reputation, both for thoroughness and reliability, in the treatment of a difficult subject. Complete revision has been carried out, and numerous additions have been made. The result is a work of the greatest utility, embodying an able presentment of the principles of practical surveying and prescribing a solid two year's course of instruction for students in this branch. The first year's practical work consists mainly of the instrumental adjustments, the closed traverse, calculation of co-ordinates, the computation of areas surveyed in the course of field practice and plotting; the second includes the adjustment of instruments, triangulation, astronomical observations, levelling, contouring, setting out curves and plotting. Mine surveying is considered in a special chapter, the experience of the author in this class of work rendering him an especially able exponent of method."—*London Mining Journal*, Dec. 2, 1911. (A. R.)

A TEXT-BOOK OF MINING GEOLOGY. By J. PARK, Professor of Mining in the University of Otago. (London: Charles Griffin & Co., Limited.) Price 6s.

"The subject is dealt with in an essentially practical way, and the various chapters are replete with illustrations drawn from the experience of the author, whose knowledge of mining is evidently both broad and deep. The chapters on morphological classification, ore veins, and the dynamics of lodes and beds are excellent examples of didactics in the domain of applied science, and, if thoroughly mastered, will furnish a solid scientific grounding to the activities of the mining engineer. Chapter VI. deals with the vexed question of theories of vein formation, concerning which the best scientific opinion is at present divided. Mr. Park quotes the various authorities with so much impartiality that it is difficult to feel certain in what direction his own opinion, informed by a long mining practice, would be cast. The chapter on ores and minerals, considered economically, is an excellent exposition of the character and occurrence of the leading minerals of commerce, and gives an excellent grounding, upon which the young engineer may erect a substantial fabric of subsequent experience. Altogether, the book is a distinctly successful exposition of geological principles in their especial relation to mining practice, and it may be placed with absolute confidence in the hands of the young mining engineer."—*London Mining Journal*, Dec. 2, 1911. (A. R.)

Adams, Henry, and Matthews, Ernest R. Reinforced Concrete Construction in Theory and Practice. 8vo, pp 330. *Longmans.* Net 10s. 6d.

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Year-book of the Scientific and Learned Societies of Great Britain and Ireland, The. 1911. 8vo, pp. 382. *C. Griffin.* 7s. 6d.

Abstracts of Patent Applications.

C.) 52/11. Laucelot Ussher. Candle saving appliances. 30.1.11.

The present invention relates to candle saving appliances of the kind consisting of a cap which rests upon, and descends with, the upper end of the candle and forms therewith a receptacle for the molten grease around the flame.

(C.) 81/11. John Penhale. Improvements in purifying mine air. 8.2.11.

This is for the method of purifying mine air, which consists in spraying the place where the purification is desired with a solution of caustic alkali or alkali metal carbonate having oxygen gas dissolved therein.

(C.) 261/11. William Ross. Improved system of overflow launders for tanks used in the process of gold recovery or extraction from ores. 8.5.11.

This invention has reference to launders, troughs, or the like, for receiving liquids, or mixtures of solids and liquids, which overflow from vats.

It is proposed to have a vat overflow launder system which consists of a number of straight lengths of launder, supported or suspended inside the vat, and joined by suitable fittings in such a manner that the lengths from chords or a circle slightly less in diameter than the internal diameter of the tank.

Such a system may have one, two, three or four, separate sections, each section consisting of part of the whole number of chords. Each section may have a separate outlet, or one outlet may be common to two sections. At desired places, in place of the ordinary connecting fitting of two chords, a connecting fitting with an outlet branch may be put in.

It is proposed to make a section of an overflow launder of parallel launders, having different depths, placed with their tops on a horizontal plane, the object being to obtain a fall or grade towards the outlet.

What is claimed is the use of a combination of parts consisting of straight lengths of launder pairs of which lengths may be parallel, and numbers of which lengths may vary in depth from one another, made of wood, or steel or of pipes with longitudinal pieces cut from them; flexible joint connecting pieces; and adjustable hanger supports or wedge pieces assembled in a vat so that the lengths of launder form chords of a circle less in diameter than the inside diameter of the tank, to obtain an overflow launder which is easily adjustable to a level, easily erected or dismantled, and capable of being standardised so that numbers of launder lengths and joint pieces may be made to form the overflow launder system of vats of widely varying diameters.

(C.) 573/11. Charles Hoyle (1), Harry Aldebert Guess (2). Improvements in classifiers. 20.10.11.

This invention relates to improvements in classifiers and separators adapted mechanically to effect a complete separation of the extremely fine particles of material, commonly known as slime, from the coarser particles, which are commonly known as sand or concentrate, and also to provide means for slowly but continuously discharging the coarser or heavier material in such a manner that it may be thoroughly washed and the lighter and finer material completely separated therefrom.

The classifier consists of a settling box of which the sides are preferably vertical, whilst the ends are inclined, one of them preferably having a steep pitch, and the other having a more gradual pitch and being extended a considerable distance above the level of the sides and other end.

The classifier is provided with an overflow pan, with a dressed overflow edge, arranged in the settling box so that its top shall be slightly below the top of the sides, and is provided also with an endless scraper conveyor for removing the coarse material from the settling box. This scraper conver extends across and engages the sloping extended end of the settling box.

In operation the feed is entered near the steep end of the box; the slime and fine sand overflow into the overflow pan; the sand settles to the bottom of the box and is then drawn by the scraper conveyor up the inclined end which is of sufficient length to permit most of the water, or other liquid, which is drawn out of the settling box, or delivered from the supply pipe, to drain off.

Means are also provided for delivering a quantity of water, or other liquid such as a solution of cyanide or other chemical, against the coarse material which is being dragged up the inclined bottom, there by thoroughly washing same and removing all of the slime or fine material.

(C.) 590/11. Ernest Clayton Johnson. Improvements in grinding mills. 27.10.11.

This specification relates to an improved form of tube mill, the interior of which is lined with chilled iron sections with longitudinal corrugations. The grinding is carried out by a number of round rods extending the full length of the cylinder which will roll on one another as the mill is rotated, and by means of the corrugations on the surface of the liner have imparted to them a shaking up or displacement motion so that the material between the rods is ground by attrition.

The ends of the cylinder are provided with removable heads with a series of radial grooves or channels with inclined surfaces on the interior. The effect of this inclined surface is to force the grinding rods into a slight longitudinal motion in addition to their ordinary motion.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 632/11. Turbo Amalgamator and Extraction Co., Ltd. (1), Alfred Andrew Lockwood (2). Improvements in the apparatus for the extraction of the precious metals from pulps and solutions. 20.11.11.

(C.) 634/11. Carl Axel Wilhelm Quiding. A mine candle holder. 21.11.11.

(P.) 635/11. Wilbur Alson Hendryx. Improvements in ore classifiers. 21.11.11.

(C.) 637/11. James Priestnall Naylor. Electric day and night sign. 24.11.11.

(C.) 638/11. Emil Berneaud (1), Vereinigte Fabriken Engl. Sicherheitszunder Draht und Kabel Werke (2). Improvements in fuses. 24.11.11.

(P.) 639/11. William George. Improvements in pigsties or chocks. 24.11.11.

(C.) 640/11. Charles Boyd Varty. Improvements in piston and bucket pumps. 25.11.11.

(C.) 641/11. Francois Phillipus Ferreira (1), Arie Hendrik van Buvin Schele (2). Dust arrestor for the use of rock drill machines. 27/11/11.

(P.) 642/11. Hugh McMenamin (1), Gordon Traver Easton (2). Improved lining for tube mills. 27.11.11.

(P.) 643/11. James Howard Vivian. Improvements in rock drill chucks. 27.11.11.

(P.) 644/11. Robert Wilkinson Walton (1), Karl Paul Otto Koch (2). Improvements relating to rope grips for trucks and the like. 28.11.11.

(P.) 645/11. James Harris. Improvements in rock drills, pneumatic tools and the like. 29.11.11.

(C.) 647/11. Thomas Ignatius Carney. Improved gate operating appliance. 30.11.11.

(P.) 648/11. John Sacks. A new and improved chemical preparation for sweetening purposes, and mode of manufacturing same. 30.11.11.

(P.) 649/11. Cecil Murray. Barbed guard wires for overhead electricity transmission lines. 5.12.11.

(C.) 650/11. Wilbur Alson Hendryx. Improvements in dewatering devices or apparatus for separating solids and liquids. 7.12.11.

(P.) 651/11. Thomas Pearson Smith. Ignition device for furnace burning pulverulent fuel. 7.12.11.

(P.) 652/11. Thomas Pearson Smith (1), Robert Reid (2). Clutch and brake locking mechanism for winding engines and the like. 7.12.11.

(C.) 654/11. John Alston Hart. Apparatus for producing interaction between liquids and gases or vapours. 8.12.11.

(P.) 655/11. Nochem Freed (1), Pieter August Smorenburg (2). Candle holder. 8.12.11.

(C.) 658/11. James Hart Robertson. Art of extracting metals electrolytically. 11.12.11.

(P.) 659/11. James Esson (1), Denis Francis Breslin (2). Improved means of replacing electrical fuses by an automatic method. 12.12.11.

(P.) 660/11. Martin Wagner. An improved apparatus for ejecting and spraying liquid solutions and the like. 13.13.11.

(C.) 661/11. Jean Bally. Improvements in electric furnaces. 15.12.11.

(C.) 662/11. Paul Theuerkorn (1), Mann and Willkomm (2). Improved method of covering and lining tubes and the like with lead, tin, or the like and apparatus therefor. 15.12.11.

(C.) 663/11. Nathaniel Greening. Improvements in wire fabrics for screens and sieves. 15.12.11

(P.) 664/11. John Mitchell Lafayette Henry. Improvements in supporting pneumatic hammer drills. 15.12.11.

(P.) 665/11. John Mitchell Lafayette Henry. Improvements in stope drilling. 15.12.11.

(C.) 667/11. Alfred Henry McNeil. Improvements in automatic fire alarm systems. 18.12.11.

(C.) 668/11. Josef Carl Steinebach. Device for automatically stopping electric hoists. 18.12.11.

(C.) 669/11. Franz Dade. An improved process for setting bort in diamond drills. 18.12.11.

(P.) 671/11. James Kyle. Improvements in apparatus for breaking ground. 21.12.11.

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(P.) 674/11. Samuel McCheaney. Improvements in locking devices. 23.12.11.

(P.) 675/11. Wilbur Alson Hendryx. Improvements in dewatering devices or means for separating solids and liquids. 28.12.11.

(C.) 677/11. Hans Edgar. Improvements in rock drills. 29.12.11.

(C.) 680/11. Albert Alenzo Pauly. Improvements in moulding apparatus. 29.12.11.

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(P.) 682/11. Ernest Wright. Improvements in machines for decorticating the fibrous leaves and stems of plants. 29.12 11.

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(C.) 37/12. George Ridgway. An improved filter for slimes pulp and the like. 17.1.12.

(P.) 38/12. James Alexander Leckie. An improved brick press. 17.1.12.

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Vol. XII. FEBRUARY, 1912. No. 8.

Proceedings
AT
Ordinary General Meeting, February 17, 1912.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, February 17th, Mr. C. B. Saner (President) in the chair. There were also present :—

63 Members : Messrs. W. R. Dowling, E. J. Laschinger, H. A. White, K. L. Graham, J. Gray, T. Johnson, F. W. Watson, A. Whitby, A. F. Crosse, J. R. Williams, Dr. W. A. Caldecott, R. G Bevington, A. McA. Johnston, Dr. J. Moir, J. Littlejohn (Members of Council), D. J. Arkell, L. Aubourg, A. Avent, R. A. Barry, S. Beaton, W. Beaver, A. J. Bowness, J. M. Campbell, B. P. Carter, J. Chilton, M. H. Coombe, C. Crichton, J. M. Dixon, P. L. Edwards, R. C. Evans, M. Fergusson, B. C. Gullachsen, J. M. L. Henry, A. Heymann, J. Higham, R. N. Kotzé, A. M. Kratz, G. A. Lawson, H. J. Lea, J. Lea, W. Mackesy, T. G. Martyn, E. L. Matthews, G. Melvill, H. H. Morrell, P. N. Nissen, A. C. L. Olsen, E. Pam, F. S. Pearce, F. D. Phillips, W. S. V. Price, J. F. Pyles, E. Roberts, C. B. Simpson, G. H. Smith, A. L. Spoor, H. Taylor, C. Toombs, J. T. Triggs, H. Warren, F. Wartenweiler and J. Watson.

23 Associates and Students : Messrs. E. G. Baskett, H. G. Brickhill, W. E. Burrows, J. Cranin, V. R. Dickson, B. W. Holman, G. F. Mathews, D. Nicholas, G. S. Paterson, W. H. Pickburn, F. J. Pooler, K. Richardson, H. Rusden, P. Scatterty, J. H. Smith, H. Stadler, D. J. Thomas, T. L. Thorne, C. D. Twynam, G. West, F. A. Westfield, A. Willcox, and S. A. Woolf.

31 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, owing to the late issue of the January *Journal*, were left until the next meeting for confirmation.

NEW MEMBERS.

Messrs. A. F. Crosse and R. G. Bevington were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected as follows :—

HARLEY, J. P., Wassau West Amalgamated Mines, Ltd., Tarquah, Gold Coast.

IRONSIDE, PERCY HUMPHREY, P. O. Vaalhoek, Pilgrims Rest. Mine Manager.

LIBBIS, CECIL FIELD, *c/o* W. G. Adamson, New Kleinfontein Co., Ltd., P. O. Box 2, Benoni. Mining Engineer.

POWELL, WILLIAM JAMES, P. O. Box 315, Salisbury. Surveyor and Assayer.

WALLACE, OLIVER MILLETT, Bantjes Consolidated Mines, Ltd., P. O. Box 2, Florida. Mining Engineer.

The Secretary notified that since the last meeting of the Society the following Associates had been admitted by the Council:

As Associates—

BEECH, SOLOMON JAMES, Sons of Gwalia, Ltd., Gwalia, West Australia. Metallurgist.

GERBER, OTTO ADOLPH, Crown Mines, Ltd., P. O. Box 123, Fordsburg. Reduction Worker.

GOWAR, FRANK HUMPHREY, P. O. Box 13, Langlaagte. Cyanider.

MACKAY, GEORGE JAMES, East Rand Proprietary Mines, Ltd., P. O. Box 19, East Rand. Metallurgical Assistant.

SIMMS, EDWARD GEORGE, P. O. Box 27, Barberton. Amalgamator.

GENERAL BUSINESS.

POISONING OF COWS BY CYANIDE.

Mr. A. Whitby *(Member of Council)*: Considering the number of cows which graze around the Witwatersrand district, it has often struck me as curious, and a tribute to the care exercised, that so few cases of cyanide poisoning occur. Possibly there are some which do not come to our notice. There are several points on which I am desirous of receiving information. What is the lethal dose, and what are the most likely organs of ruminants from which to obtain evidence of cyanide poisoning? This, however, appears certain, that the stomachs are not always the most likely places, except when a considerable overdose has been taken. The conveyance of the poison to the tissues appears also to be most rapid and quite independent of the strength of the solution. A cow will drink 10 to 15 gallons of water, and if we assume the water to be carrying only ·005% KCy, she will absorb from 35 gr. to 50 gr. The lethal dose for man is only 5 gr. Cyanide has been recovered in certain cases from the brain, the saliva, the intestines and the tissues generally, but no one appears to have attempted, in the case of ruminants, to search elsewhere than the stomach, which, to my mind, is almost the last place to look. I have one recorded instance where the calculated stomach content amounted to something like 90 gr., but with the small quantity of material supplied the greatest care had to be exercised in the examination. In two other cases, where the veterinary surgeon clearly diagnosed cyanide, no evidence could be obtained by analysis. Here, again, the quantity supplied was quite insufficient but I doubt, unless by working on a gallon or two, whether clear evidence would have been gathered. I think some sort of enquiry might be instituted and subjects obtained by those in a position to do so, for investigation of the points I have raised, otherwise some of the dairy farmers, private owners and others, who venture to graze their cattle in the vicinity of cyanide works may not obtain justice.

MINING EXHIBITION.

The President: I should like to refer to the Mining Exhibition. I think the Society should be particularly proud of this Exhibition. It reflects great credit on all those who participated in getting it together. It might be only a beginning, but judging from the crowds of people going through daily, it is a good beginning, and I think that the Society can congratulate itself on what it is doing—it is doing a good work for the mining industry and for the health of all employed on these fields.

WELCOME TO PRESIDENT.

Mr. E. J. Laschinger (*Vice-President*): I should like to express the pleasure of the Society at seeing its President here to-night.

The President: It gives me much pleasure to be here, and I thank you for your cordial reception. I regret that I cannot preside more often, but it takes me nearly 24 hours to come here by road and rail. I will now ask Mr. Dowling to take the chair while I read my Presidential Address.

PRESIDENTIAL ADDRESS.

By Chas. B. Saner, M.I.M.M., M.I.M.E.
(President).

I must thank you heartily for my election to the Presidential Chair. It is a position I most highly prize, especially when I look over the roll of your Past-Presidents, with whom I am now soon to be proudly classed.

I have been an active working member of the Society and in the Council for several years, and its interests have been mine, but to my great sorrow, through enforced absence, I have not been able to devote the time and energy I intended this year to the Society's welfare. In accepting the honour you have seen fit to bestow upon me, I at once regretted that custom had decreed that each incoming President should give an address which is supposed to enlarge on his particular line of the many branches embraced by the activities of the Society. However, I will endeavour to follow in my predecessors' footsteps, but must warn you that as the standard set is high I may fail; in each succeeding year the compilation of this annual address becomes increasingly difficult.

The varied character of a mining engineer's life is aptly exemplified in mine, for, when I had hoped to have been able to make up for any personal shortcoming by zeal and hard work in furthering the interests and prosperity of this Society, fate decreed that I should be located for a time far from your fascinating monthly meetings. A mining engineer in South Africa, as in every other part of the world, belongs truly

> "To the legions of the lost ones,
> To the cohort of the damned."

yet I flatter myself that we are, especially in this country, doing our little best as pioneers to discover, open up, and exploit the boundless wealth that is locked up in the bowels of these tractless acres. Agriculture may be the backbone of a country, but the exploitation of its mineral wealth initiates its prosperity and is its brain and muscle.

Australia laid the foundation of its present pastoral greatness when the discovery of gold brought to its shores population and capital. California was a comparatively unknown land until the finding of the precious yellow metal attracted millions of men and money, and here in the very near past—well within the lives of many of us—our own loved South Africa was thought to be a wind-swept sandy desert, fit for only a few outcasts and intrepid wild-game hunters, when the discovery of diamonds and gold changed all into a land, if not flowing with milk and honey, at any rate as rich and healthy and good to live in as any in the known universe.

Thanks to my predecessor (Dr. Moir), I am able to briefly refer to the past year's work, which may be regarded with some considerable degree of satisfaction. We have had good regular attendances at our meetings The papers have been varied and interesting, though not as well discussed as the labour of preparing them merits. A full and free discussion ofttimes greatly enhances the value of the original contribution, and when a man has spent time and trouble in preparing a paper the least that should be done is for others who have experience, ideas and knowledge, to come forward and add to and criticise that effort. We, on the Rand, have hitherto been proud of the fact that we keep no secrets from each other, and this should stimulate all to give of their experience. It is for the general advancement of the industry we have at heart. As you know, the Council, on representation from some prominent members, inaugurated the system of sectional meetings—that is, devoting a particular night to a special subject, hoping that by this means the busy expert would attend and take part in discussions and give of his knowledge. Unfortunately he has not materialised, especially in mining; and though we had great expectations! However, on the whole the experiment can be voted a success. It has often been stated that free opinion and open discussion is not wanted here. Weak indeed is the engineer, and poor the industry, and decadent the country that is afraid of honest and fair criticism! I trust that this Society will uphold its reputation as a straight-talking and lively body, and will not only continue its usefulness to the industry from which it has immediately sprung, but also devote its active energies to all Chemical, Metallurgical, and Mining problems throughout the Union.

Although not wishing to weary you with details, I feel I must touch on a few of the more important recent problems arising in the progress of Rand mining. From these it will be noted that we are by no means standing still and that anything tending to better mining and improved conditions is tried, and, if successful, adopted. But at the same time it must be realised that even if a special appliance or method is a marked success elsewhere, it may be a failure here—not for want of sympathetic trials, but because the local conditions may differ and render it totally unsuitable. While we have every reason to be proud of the advance made in mining methods even during the last ten years, and just as we often smile with a superior air at the crude work of our predecessors (not realising their great difficulties), so our present-day methods will in turn be belittled by our successors—in fact that is often done to-day by the newer arrivals. Nevertheless many new and serious problems will ever crop up, and be solved, and many economies effected in the efficient working of the deep level mines, which will be but the gradual improvement of the crude ideas evolved by the old pioneer and miner. Therefore, do not let us despise the achievements of the past. When comparing the practice of twenty years ago with to-day, one realises that this wonderful Rand industry, and, in lesser degrees, the other mining centres, were started with the slow and steady ox-wagon as the sole means of transport whether for machinery, foodstuffs or coal, one is filled with wonder at what was accomplished. Just as I saw at the recent magnificent Coronation Naval Review the last Super-Dreadnought as the latest perfection of the wooden cockle-shells of Nelson's day, so do we now see the great complex industrial concerns in the shape of huge amalgamated properties as the outcome of the small "one-horse" mines of a few years ago. Thanks to the accurate system of underground surveying and the splendid detailed plans kept on all mines, and the free interchange of important information with reference to faults, dikes, and other dislocations, between mines, the deep levels and amalgamated properties can be, and are being, laid out on well thought out and comprehensive lines—so different from the practice of a few years ago. Where in the past each small area had numerous shafts and surface engines fitfully working, now there are a few large shafts being sunk for each area, designed and equipped to deal with the whole mine contents and to work continually.

Large seven-compartment rectangular shafts with their long axis towards the dip—such as those (to mention a few) at Crown Mines No. 5; Village Deep No. 3 (Turf Mines); City Deep; Brakpan; Cinderella Consolidated; and Government areas; with magnificent steel headgears over one hundred feet high; reinforced concrete collar setts, raised well above the natural ground level; magnificent hauling engines driven by steam or electricity—have been, and are being sunk with remarkable speed and reduction in costs. But while in Europe water is the great bugbear to sinking operations, and elaborate and complete arrangements are made to cope with this, here for the most part water in quantities is unknown in sinking, and, generally, when met with in any considerable volume, means temporary disaster. These shafts are being equipped both above and below ground to be worked to their fullest capacity. Here we are approaching European practice; as I noticed during my recent visit how far ahead they are of us in the handling and transport of material; and what so impressed me in both English and German collieries was the elaborate arrangements made at the shaft bottom for cheap and quick handling of coal from good large well-lighted stations, and the hauling engines were kept constantly in motion. The first cost was considerable but well spent, and it was the last cost, and where our profits are shillings per ton, theirs are but pennies. No coal is extracted until the mine is well developed and opened out. Mechanical haulage on one system or another is installed wherever possible along the main levels, and these are fed by human and horse power from the various faces. On the Rand, under the new conditions due to the increased size of the mining areas under one control and the concentration of hauling through fewer shafts, wide main haulage levels are being driven straight, and for the most part, in the footwall of the reef. Mechanical haulages are being installed and the ore from the stopes sent down through passes and incline planes, etc. In connection with these levels, some excellent monthly and average footages have been made, notably amongst others being at the Modderfontein "B"; Cinderella Consolidated; East Rand Extension and Crown Mines. In conveying the ore from the face to the haulage level, in Europe the collection rails can be placed close to the working faces and remain unharmed, as few explosives and light charges are used, but here we cannot follow that practice on account of the hardness of the rock and the quantity and strength of explosives seriously damaging rails and trucks. Hence the question of transporting our ore, when broken, from the working faces to the main levels remains a problem—in fact, series of problems—as the narrow close timbered stopes of the West Rand; the comparatively wide steep stopes of the Central Rand, and the flat measures of the far East Rand, each require a different method, and as the angles of dip vary considerably even in the same mine, one system cannot be installed throughout. By this I do not wish for a moment to infer that there are no systems in use or that numerous devices have not been experimented with—to mention only a few—the mono-rail; underground aerial gear; stope conveyors of various designs; gravity planes; and what I still consider to be by far the best and most adaptable, the shaking and ground chutes—these and numerous others have been used with varying success, and other mechanical devices are continually being tried. A most important feature of late has been the marked increase of the small rock drill in stoping operations. The dire necessity of the mines to find a substitute for native labour and the world-wide publicity given to the question by the competition organised by the Chamber of Mines and the Transvaal Government has set the inventors seriously to work, and in the near future I hope we shall see a real first-class small drilling machine evolved from the numerous trials, both private and public, that are being continually carried out of various inventions submitted. But the machine to do good work must be handled by skilled men.

To my mind one of the most important introductions of new ideas on the Rand has been that of filling the worked-out underground areas with water-borne sand. The general methods adopted have been well described in a paper read before this Society and the subsequent discussions thereon, but owing to the abrasive nature of the sand very speedily destroying the pipes, some modifications in lowering the material—such as by means of large diameter boreholes—are being tried. During a recent visit to Westphalia I had the good fortune to examine some sand filling operations, and here I was struck with the improvement in mining. All the coal was taken out, a little timber put in temporarily, and the area at once filled with sand, the timber being taken out to be used again. The dip of the seams was 45°, but the material used was very different from the Rand sand, it not having nearly such a destructive effect on the pipes, and experience proving that porcelain-lined pipes gave by far the longest service. I feel sure that when sand filling is extended and the filling in of areas is concurrent with the excavation, a modification of the mining method in the deeper levels will be made. There will be fewer falls of hanging rock, and ventilation will be greatly assisted by confining the fresh air to the travelling ways and working faces.

This important question of ventilation has been energetically tackled. In the outcrop mines, owing to the numerous outlets, there was no need for artificial ventilation, but with the deep mining it has become a necessity, and most of the mines are installing fans of various capacities to suit their respective requirements, with the necessary doors to direct the air currents. But ventilation has accentuated the dust problem, so that all mines are now installing a complete clean water system—not only to damp the rock during development, but also while shovelling the broken ore in the stopes. This brings me to that grave question—the prevention of dust—a problem that I am quite certain each and everyone of us wishes to see speedily solved, as it is proving such a terrible scourge to all miners and underground workers. There is no need for me to describe any of the various devices being tried; they are known to you all, and it only behoves each one of us to do his best by every effort to enforce any regulations and eliminate this fell disease. The Transvaal Miners' Phthisis Sanatorium at Sprinkell, opened by the Administrator on the 18th November, 1911, is, I understand, an excellent institution, and the thanks of all are due to the Mining Industry for erecting it. It is filling a much needed want. During my recent visit to England I had the privilege of going over the Brompton Hospital Phthisis Sanatorium at Frimley under the able direction of Dr. Marcus Patterson. His system of cure so impressed me that I feel, even at the risk of wearying you, that I must shortly describe it :—

On entering the Hospital the patient undergoes

1st. Complete rest in the open air and under careful observation until the temperature is normal.

2nd. A regular and progressive system of exercise to promote auto-inoculation.

3rd. The exercise is gradually increased while temperature is kept regular. If any change in temperature, complete rest at once. Then the exercise is commenced again, not from the beginning, but from the stage where the change took place.

4th. The patient then works a full day for some time (three weeks at least) while the temperature remains normal at the regular occupation of the patient, or harder.

5th. After leaving the Sanatorium the diet and method of living of the patient to be regular to prevent the return of the disease.

Thus the patient does not loaf and get fat and lazy through ease and idleness, as happens so often when a miner goes off for a holiday "touched with phthisis" from here; but the system "hardens the body, invigorates the mind and keeps alive the spirit of independence." This particularly struck me when in conversation with several of the patients during my visit, and I feel sure a similar system here would work wonders for our miners when in the early stages of the disease. That is a hospital for cure. To my mind it is most necessary and would be most beneficial if a large Sanatorium on similar lines were erected in some healthy country place in the Union—I care not where—for miners and others to go to be cured, at reasonable fees; and this should be done at Government expense. As a suggestion in this way, start a huge experimental farm with white labour; thus the convalescent miner will be learning the elements of farming, and on retiring would "go back to the land" as is so eloquently preached by all Union politicians.

Another question—in fact the most important question that affects everyone in this sub-continent, whether he be associated with mining or not, is that of labour, both white and black, and although this subject has been, and is being thrashed to death by all and sundry, I crave your indulgence while I too presume to add some remarks thereon:—We find that many of the mines have erected comfortable and well-built cottages, electric lighted and water supplied, and have built up-to-date change houses with hot and cold water baths close to the shaft head. The housing of the mine employees has greatly improved of late years both for married and single. These compare very favourably with English practice, though not nearly so well fitted up as those on German and Holland mines I visited. In England, while neat rows of cottages in streets are being erected on some of the newer mines, change houses and bath-rooms are conspicuous by their absence, due, I was informed, to the refusal of the independent British workmen to be "bathed by order." While in Westphalia and the State mines of Holland I consider the other extreme has been reached. The change houses are almost too luxurious. Lofty well-aired and lighted buildings paved with concrete, and the walls faced with white enamelled bricks. No lockers are supplied, but the clothes are hung by chains high above in the lofty building where the heating pipes dry them, and a continuous supply of air thoroughly ventilates them. Hot and cold showers and plunge baths built all of the same beautiful brick; and a closed passage from the change house to the shaft head is erected. Immediately outside the change house there is a tuck-shop where soup, coffee, cocoa, milk, etc., are sold at a nominal figure. All is spotlessly clean. Everyone is made to change completely before entering the mine. This I consider is an excellent rule and should be enforced on the Rand mines.

A great step forward here has been the establishment of the Mines Training School at the Wolhuter Mine; by this means youths have an opportunity of properly learning their trade. I cannot understand how any young man, however brilliant, can expect to be a miner in a few months when learning the elements of the science by the aid of a dim light underground, while the apprentice fitter takes years in bright sunshine to master his trade. By making the mines healthy, living on them pleasant, and employment in them permanent, not only will a settled contented supply of skilled labour be ensured to every mine, but the sons of these men will naturally, as in every other trade and calling, follow in their fathers' footsteps. The first part has been, and is being done, and it behoves all the active workers on the mines to see to it that the latter follows. We must breed, not import, our future miners. So much for the skilled man!

But the advancement and prosperity of the whole Union depends upon what is known as the

"Unskilled labour supply. This question, which is mentioned at least in every mine report, and was debated and discussed at the recent Agricultural Congress, is a national one and, therefore, should be tackled seriously by the nation's Parliament. To-day the farmer is calling; the mines are searching everywhere for the precious nigger. He is being begged to come to work, instead of seeking employment as the white man has to. Never since the repatriation of the Chinese has the labour been plentiful in South Africa, and if there is not sufficient unskilled labour in this country, then it must be imported under proper restrictions. I care not whether black, or yellow.

A great advance has been made in the better accommodation, feeding and general care of the native. The compounds are substantially built, warmed, and ventilated. Excellent well-appointed hospitals with trained white attendants and native assistants are erected on most mines. Native change houses with hot and cold showers are built, or being built close to the shafts with covered inways from the compound to the shaft head. Every boy coming on and going off shift is given a cup of coffee or cocoa and a loaf of bread, while the daily ration consists of meat, rice, mealie meal and vegetables; all well cooked in central kitchens under proper supervision. A weekly quota of Kafir beer brewed not too strong is also supplied. And what is being done on some mines, and which might be more widely extended with profit, is the erection of married quarters. Thus the idle native becomes a regular worker. His family is reared in the atmosphere of labour and soon acquires civilised habits, such as the desire for work, wearing clothes, eating better food, etc. He spends more money and indirectly contributes to the State Revenue, while his children, the rising generation, because of their environment would soon become useful long-service workers.

We, the men actually handling the mines, realise how difficult it is to do good work with crude material. We must have labour, and that labour must be with us long enough to repay for proper training. Regular labour means expansion for the mining industry and consequent prosperity to farming. The two go hand in hand, and therefore the problem must be faced by all squarely.

I should have perhaps devoted more of my remarks to mining proper, and enlarged on such subjects of general interest as :—

"The new mining regulations.

The installation of large central electrical and compressed air power stations.

Some recent excellent underground survey connections.

Latest deep level pumping arrangements.

Single shift work.

Contract versus day's pay for whites.

Piece work for natives, and the recent various improvements in the reduction works"; but I realise that already I have almost worn out your patience, and can only hope and trust that the subjects touched upon and mentioned will be made the titles of good papers during the coming months.

The old days of rule of thumb have gone. The present technical institutions and colleges, both here and in Europe, are well preparing the mining engineer for the continual struggle with nature. Mining, as has been aptly said, has become a science or aggregation of sciences with numerous daily and complex problems; and there is a duty that every member owes to this Society, and I would especially appeal to the mining members to improve its usefulness by giving us the benefit of their experiences—good and bad—and for fuller information on the numerous mining problems, by writing short, concise papers of their actual experience.

If the old proverb "Silence is Golden" be anyway true, most of our mining members must by now be wealthy indeed, and I cannot do better than quote an extract from the address of the President of the Institution of Mining and Metallurgy :

> "Life is short, individual experience and endeavour so limited, while the problems of our science are so unending, that he who is unwilling to give as freely as he takes is a little better than a drone in the hive."

I have not made any mention of base metal or coal mining, since owing to my not being intimately associated with these industries, my facts and figures would only come from sources accessible to all. Our excellent Mines Department is responsible for, and issues, too, good reports; but at the same time I should like to impress on all not to trust in gold alone. "Auri Sacra Fames"—the cursed thirst for gold—should not be the only end and aim of the mining engineer in South Africa. A great future has been predicted for base metals, and on these the wealth of the richest country in the world was started. Why not here also?

In conclusion, I must thank you for so patiently listening to my rambling remarks, and I assure you that it is a great sorrow to me that I am unable to devote more time to your affairs. Though I may not be present often in person, my heart is always with you.

Mr. W. R. Dowling (*Vice-President*): It has become a practice in the past not to submit the Presidential Address for discussion. It seems to me that it is rather a pity that that has become a rule. Mr. Saner's address is so full of interesting matter gathered not only here but in England and Europe, during his holiday, that it does seem a pity that we cannot discuss it. However, our President seems to have used his holiday very well, and has given us the benefit of what he saw. I think we all agree with what Mr. Laschinger said that we regret that Mr. Saner is not now resident here and cannot give us the attention we look for from our President. It is unfortunate that he has been moved away, and no one regrets it more than he does. I move a hearty vote of thanks to the President for his Presidential Address.

Mr. Andrew F. Crosse (*Past-President*): I have great pleasure in seconding that vote of thanks. I may be conservative, but I must say that I think it would be a pity to discuss the President's Address.

The President: I must express my thanks to the Vice-Presidents, the Council and the Secretary of the Society, for carrying on the business of the Society during my absence. I think it is a case which can be brought forward as typical. We make very few changes in the administration of the Society—the majority of the old hands keep on pegging away, and the result is *esprit de corps* and tip-top efficiency—all the men put their backs into it knowing that they are going to stay. I should like also to mention that when I went to the Continent I noticed the spirit of brotherhood among the mining engineers. Being an Englishman I could not talk any language but my own. In one case I could not get on at all. I tried gesticulation—and I am fairly good at that—but I could not get my friend to understand what I wanted. I wanted to get at the capacity of a sinking pump. I did every thing possible to find out but could not make the manager understand what I was driving at. Finally, we went, as usual, to a neighbouring "tea room," and then I found out. I picked up a bottle and again queried, and found out that so many bottles of beer passed through the pump. The way I was received in my private capacity made me tremble to think what would have happened had I mentioned that I was connected with this Society. I think this shows that while we give every visitor who comes here a good time, they reciprocate it to the fullest possible extent. I should like especially to mention the name of one gentleman—Mr. Bunge—who was here before the war, but who is now Superintendent of the State Mines in Holland. He gave me an excellent time and told me everything possible, and even now if I want any information he is always ready to supply it.

ACCIDENTS IN TRANSVAAL MINES.

(With Special Reference to Falls of Ground.)

(*Read at November Meeting, 1911.*)

By James Chilton (Member).

DISCUSSION

Mr. Tom Johnson (*Member of Council*): There has been much talk about the increase in the number of deaths and accidents, but I am pleased to note that according to Table I. of the author's paper the death rate is lower for the last four years than for the preceding five years, although 1909 was an abnormal year, owing to the Witwatersrand Mine disaster.

There is, however, an apparent increase of accidents owing to the more conscientious carrying out of the rule requiring non-fatal accidents to be reported. Due allowance must also be made for the enormous number of raw natives that have passed through the mines during the years under review, besides the large permanent increase. In June, 1903, there were in the mines of the Rand 7,382 whites with 34,896 natives; in June, 1910, there were 23,674 whites with 189,345 coloured labourers. The industry has built up its coloured force at the rate of 22,000 per year besides the many thousands that have returned to their homes. There is no wonder that the accident rate is high, especially with the class of supervision that we have on the Rand.

The author is wrong in inferring that the death rate has gone up in Great Britain through the introduction of the eight hours' system. As I mentioned in my previous contribution, that increase was caused by the large explosions, and had nothing to do with the eight hours' system. H. M. Chief Inspector of Mines says that, deducting the colliery explosions by which ten or more persons were killed, the death rate per 1,000 is as follows[*]:—

1898	1·37	1905	1·26
1899	1·34	1906	1·38
1900	1·41	1907	1·46
1901	1·34	1908	1·35
1902	1·34	1909	1·38
1903	1·35	1910	1·35
1904	1·34		

The average for the thirteen years is 1·36. He also says "Now it must be admitted that the increase of accidents, if any, due to reducing the working shift in British collieries would take place in hoisting and haulage and from falls of ground."

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[*] Mines and Quarries Part II. General Report with Statistics for 1910.

Fatal accident rates per 1,000 for these classes of work for the last five years are :—

Nature of Accident.	1906.	1907.	1908.	1909.	1910.
Shaft ...	0·10	0·13	0·10	0·10	0·10
Haulage ...	0·33	0·32	0·28	0 33	0·34
Falls of ground...	0·78	0·76	0·74	0·74	0·75*

(*On deducting the case of the Water Haigh Colliery where six persons were killed by the collapse of the sides of a sinking shaft, the figure is 0·71 per 1,000 persons employed.)

These figures tend to show that the eight hours system has not caused the death rate to go up.

Quoting again from the same source :—The minor accidents in Britain for the three years that "The Notice of Accidents Act" has been in operation, were as follows :—

Year.	Per 1,000
1908 ...	144
1909 ...	151
1910 ...	152

These figures certainly show an increase, but as the Chief Inspector remarks, there are other causes besides the eight hours system that could be advanced with equal force to account for it. The author says the death rate for the Transvaal is 31·51 per million tons raised. In the year 1907-8 it was 34·6, in the year 1903-4 it was 38·9, so it does not seem on the tonnage basis that we are getting worse.

Referring to the author's Table II. it seems there is a decrease in the percentage of accidents due to ignorance and carelessness, and an increase due to the inherent danger of the work. This should be noted ; if the danger of the work is increasing, and I contend it is, and will increase so long as our present system of working is in vogue, then we must be doing some little good towards holding the accident rate down. The author and several contributors to the discussion seem to blame the speeding-up in the deep levels, but like Mr. D. Rees I do not see where the hustle comes in. I do not think big mills have any effect on accidents, for I do not think more rock per person is handled on the largest mines than on the smallest mines. There are several who think that giving certificates too easily leads to accidents. This is a matter of opinion. It is my experience that it is the miner of some years standing that gets himself or his boys hurt. The Mines Department could give us some information on this point if they have the date of issue of the certificates of the men meeting with accidents. The author's Table II. bears out my experience, which I have mentioned several times at this Society's meetings in the past. I beg to differ from the Inspector of Mines, Pretoria, about the danger in the two classes of mines, *i.e.* the Rand mines and outside mines. I have worked in several outside mines and also in several of the Rand mines and consider that there is no comparison possible. The Rand Deeps are far more dangerous than outside mines and to compare the mines of the Pretoria district with those of the Rand is like comparing a thimble with a bucket. The "Mines and Works Regulations" show this difference in no uncertain manner. Then Mr. Trevor's table, if it proves anything, proves that for the same class of mine, either gold or coal, taking gold mines as outcrops, the Rand mines show up best.

Several think that the frequent changes have a lot to do with the increase of accidents, but we have always had these changes taking place. There is no doubt that a mine which has an unsettled labour force is liable to have a large accident rate, but there is nothing new in this. The daily change per mine as shown by Dr. Macaulay's figures is 0·5% or 2·1 men, and this is certainly too high. Mr. Vaughan, Inspector of Mines, says he does not agree with the appointment of a small army of inspectors. I have not noticed during this discussion that anyone proposed the appointment of a small army of inspectors. It must not be forgotten that the Government has made an increase in their "Rules and Regulations for Mines and Works," and it is up to the Mines Department to see they are carried out as far as possible. As to asking the controllers of the mines to put in inspectors of their own, they would probably reply, and with truth, that they have increased the number of officials underground by 50% during the last few years and that there are sufficient officials to see that the work is carried on in a safe and proper manner. In my opinion there are too many officials underground causing far too much overlapping, but with all this extra officialdom we do not get the accident rate down. The Government must take their share of the blame, as it is primarily their duty to look to the life of the workers. After the war when they were building up the Mines Department, to save money they watered the Mines Inspectorate with men who had little or no knowledge of underground work, such as an Inspector should have. I am one of those who think that Inspectors of mines should be of equal standing at least with mine managers, also that they should be such that both managers and men can look up to them, with confidence for help and advice in safety matters. This has not been so in some cases in the past. The mines have also gone in for a lot of officials who at the best are only half trained. It is very amusing to hear many of these young fellows, who think that, because their parents have spent money in putting them through some mining school, they are competent to teach practical men the practical work of a mine.

Prevention of accidents cannot be carried out by those who do not understand where and what the danger is, no matter how willing they may be. I frequently see work being done for safety which actually increases the danger instead of decreasing it. Then we have the recklessness of the miners which is materially increased by the employment of inefficient bosses. Miners take many chances that they should not—as we all know, familiarity breeds contempt; that is why old hands meet trouble whilst newer hands keep clear. A good miner working on what everyone knows is a bad and dangerous job will keep himself and helpers safe, simply because it is too large to run risks with, but in a good place he gets tripped up by the small thing, because it is a small thing and he takes the risk. It is the officials' duty to see these small things and see that the men concerned make them safe. Managers should also get into their mines a little oftener and help to see that safety measures are being taken. Directors should also absorb a little more mining knowledge so that they will not be so easily taken in by a smooth tongue and bluff. There are many officials who go around a mine who see danger in a ragged roof where there is no danger, but cannot see danger in a smooth roof that is ready to fall on them.

Personally, I think the mining people are rather blind in bothering their heads so much about native labour; the use of more white labour would tend to make the working of the mines less dangerous, more stable and cheaper. Another factor in regard to accidents is the non-recognition of the need of changing our system of working in the deeps. The conditions in the deeps are very different from those in the outcrops. Everyone has to be conveyed into the mines, the roof is of a different nature, there is more machine work and the roof is more broken by the heavy charges. At greater depths the excavated area is becoming greater, many of the outcrops are finished or about finished and being robbed of their pillars are throwing extra weight on to the deeps.

Several of the contributors have laid stress on the leaving of more pillars and putting in more waste packs, but if we must leave pillars enough to make sure of carrying the roof we should have to shut down a few of our deep levels, as the expenses would become too great; fortunately, it is not necessary to do so, for both on the score of safety and economy it is better to work without pillars and pack. Although I am an advocate of waste packs and such like aids to mining, the way a lot of this work is done, is like the packs—waste—and of small use. Gold miners on the Rand here have not yet recognised the important part time plays in regard to safety when extracting a block of ground.

Summing up, we want better mining knowledge from top to bottom, stricter discipline and a strong public opinion to back things. It is to be hoped that the author's paper will help to form this public opinion.

Mr. R. N. Kotze, Government Mining Engineer (*Member*): Mr. Chilton's paper is to be welcomed as an expression of opinion from one who is not a Government official and whose pronouncement on the subject of mining accidents may possibly attract more attention and be more fruitful of results than that of such an official. Year after year the Report issued by the Mines Department has voluminous references to mining accidents, but I am afraid the mining public is not particularly interested in them and satisfies itself with perusing the passing references made to the Report in the columns of the newspapers. An inspector of mines is supposed to be strongly biassed in favour of the suppression of accidents as against the enhancement of profits, and apparently his views are not taken very seriously. I shall therefore try to curtail my remarks on the author's paper, in order to inflict as little of my presumably biassed views on you as possible.

The author's statement that our accident rate goes on increasing is not correct. This will be apparent if his table of death rates is examined. If to it the death rate for 1911 be added, which is 3·76, a steady increase is not perceptible. On the contrary the death rate is improving, even if only slightly and irregularly. The author's further statement that we hold the ignoble distinction of possessing the highest accident rate of any mining district in the world is also incorrect. The following table shows the death rate in a number of countries for the years 1907-1909, and it will be observed that the Transvaal is not the greatest sinner. The death rate for 1911 is the lowest on record with the exception of the abnormal years 1901-2.

In this table also will be found mining districts, and not unimportant ones, which have a higher death rate than the Transvaal. And again the author makes a comparison of the Transvaal with England on the tonnage basis and finds it humiliating. He states that in England the death rate per million tons raised was 1·116, while in the Transvaal it was 31·51. My own calculation gives me a death rate per million tons in England for 1910 of 6·4 for coal mines and 19·7 for metalliferous mines, as against the Witwatersrand metalliferous rate for 1909-10 of 31·5. For coal mines the Transvaal

Name of Country.	Death Rate per 1,000 Employees.		
	1907.	1908.	1909.
Australia	1·31	1·36	0·93
Austria	1·23	1·01	1·10
Belgium	1·03	1·06	0·95
British Columbia	5·10	3·99	7·62
Canada	3·16	3·29	3·73
Cape Colony	2·29	2·35	1·25
France	1·24	1·08	1·38
Germany	2·36	2·56	2·13
Gold Coast	2·75	2·66	3·71
Hungary	1·07	1·73	2·47
India	0·90	1·50	1·42
Italy	1·90	1·29	1·31
Japan	2·71	1·55	2·35
Mexico	5·18	4·64	5·78
Natal	2·73	10·38	2·85
New South Wales	0·94	1·36	0·93
New Zealand	1·43	1·45	1·64
Orange Free State	2·00	1·56	1·12
Prussia	1·44	2·42	1·88
Queensland	0·75	2·02	1·99
Rhodesia	3·96	4·28	4·92
Saxony	1·60	1·30	1·81
Spain	2·30	2·28	2·31
Transvaal	4·58	4·03	5·32
United States America	4·66	3·42	3·35
Victoria	1·16	0·93	0·77
West Australia	2·41	2·32	1·85

The individual States of the United States employing more than 10,000 miners have the following rates :—

State.	No of Miners 1909.	Death Rates.		
		1907.	1908.	1909.
Alabama	17,760	7·20	5·86	7·26
Colorado	11,472	1·97	2·62	2·28
Illinois	69,425	2·62	2·58	2·93
Indiana	20,937	2·52	2·36	2·64
Iowa	17,286	2·63	1·93	1·56
Kansas	12,359	2·57	2·74	2·83
Kentucky	16,903	1·89	2·15	2·01
Ohio	38,114	3·27	2·23	2·45
Pennsylvania	326,122	4·30	3·50	3·10
Tennessee	10,031	2·72	2·99	5·36
W. Virginia	55,433	12·35	5·50	5·85

rate for that year was 6·2. There is moreover nothing to be gained by making comparisons on a basis which allows no true comparison, and the tonnage basis for England and the Transvaal respectively is such a basis. Coal mines predominate in England, and the tonnage of coal broken per man is much greater than that of quartz in a gold mine. The position in the Transvaal is bad enough without seeking to make it appear worse by inaccurate statements and unfair comparisons.

I agree with the author's statements regarding hustle and the resulting accident rate, with this difference that I would divide the mines into large and small rather than deep level and outcrop. I may, however, point out that hustle in itself is not necessarily conducive to accidents. In support of this I may point out that efficiency necessarily means a certain degree of hustle, and that some of the best run mines on the Rand have a low accident rate. It is obvious that this good result is secured by hustle of the right sort without sacrificing the attention and time necessary for looking after safety. This again is the result of perfection of organisation and of executive control, in a word, of good management, which pays due regard both to the interests of shareholders and to the welfare of the employees. That is to say, given good management, you will have a lower accident rate. I will not, however, venture to state the converse as also true.

The author quotes Dr. Maynard's statement that the accident death-rate amongst natives shows little tendency to decrease with increased period of work. Dr. Maynard's tables, however, contradict his statement, for they show that the death-rate fell from 4·5 amongst natives working under 3 months to 3·5 amongst natives working 9 months, so that the author was correct in his own conclusion that the more experienced native worker is less liable to accidents. The analysis is however a matter of purely academic interest since there is no hope of getting natives to work for longer periods than they are already doing.

The author suggests educating the miner by lectures on safety measures. When one views the comparative failure of the South African School of Mines in its efforts to promote mining classes along the reef, one may be pardoned for doubting the success of such a scheme. If employers and employed take practically no interest in a matter so closely affecting profits and pay as the improvement of the efficiency of the miner, what chance is there for the success of a similar method for merely reducing accidents? I wish it were otherwise, but human nature cannot be changed.

The author's next suggestion is that roof inspectors be appointed. Mr. J. E. Vaughan has already critised this suggestion, and pointed out that in so far as it would remove responsibility for the security of his working place from the miner, the system would be faulty. But there is no necessity to go as far as this. The superimposition of the roof inspector over and above the miner should lead to good results. It should be his duty to affect a comprehensive examination of the roof and to supervise and perfect the work already done by the miner. If, however, the duties of the author's roof inspector be compared with those laid down in the new regulations for a shift boss, they will be found identical except that the shift boss has also other duties to perform. The suggestions made for the examination of the roof and for the reporting of such examinations are precisely those contemplated in Regulation No. 161. Obviously the number of shift bosses must be sufficiently large, and their knowledge and experience sufficiently extensive to allow of effective supervision as contemplated.

The author's final suggestion is that of systematic timbering. His remarks on this point would lead one to believe that this should be the rule on the Rand. I agree with this if the author will extend his systematic timbering to mean also systematic pillaring and packing. In most mines on the Rand systematic timbering in the form of props would be useless, owing to the condition of weight and pressure met with, but the fact that disastrous falls of hanging would be diminished in many mines by the systematic use of pillars, packs and cribs, cannot admit of any doubt.

At the last meeting of the Society, Dr. Macaulay quoted some figures showing the bearing which frequent changes in personnel had on the death-rate. Since the compilation of these statistics, for suggesting which I have to thank Dr. Macaulay, a couple of belated mines have sent in their returns and I am thus enabled to present the complete data for the producing mines of the Rand. These show that during the period January to June, 1911, the Rand employed on an average 22,180 white men and that during that period there were 17,745 changes in personnel, or an average of 13·3% per month. The relation of changes to accidents is shown in the following table:—

Monthly percentages of change in personnel.	Death Rate per 1,000 Employees.	
	July-Dec., 1910.	Jan.-June, 1911.
0 — 5 per cent....	0·93	3·47
6 — 10 ,, ...	3·36	3·48
11 — 15 ,, ...	4·03	3·73
16 — 20 ,, ...	3·80	4·34
21 — 25 ,, ...	5·90	4·96
26 per cent. and over	5·08	5·10

The simultaneous rise of the death-rate and the change rate is very striking, but after the remarks of Dr. Macaulay I need not labour the point any further.

A complete analysis of the reasons why the local accident rate is high as compared with that of England would involve the production of a small volume. There can, however, be no doubt that the principal reason for the difference lies in the ignorance of our white and coloured miners and in the labour system that has grown up here. In English coal mines the miners do not work in large gangs, as is the case with our native labour, so that on the average an accident affects a smaller number of men. Further, each man is a trained miner of long experience, well qualified to look after his own safety, in addition to which his shift boss has before the commencement of the shift also examined the working place. In the Transvaal the white miner himself is generally a man who, instead of the lengthy and careful instruction in the art of mining by his father or some other relative or friend, has picked up mining in a more or less haphazard way. If he had to do only the work the native does now, he would possibly look after his own individual safety pretty nearly as well as the English miner, but instead of that he has to provide for the security of a large gang. He ought to have the same sense of responsibility and the same mature experience and to devote the same attention to the work of safety as the shift boss or overman in an English coal mine. Can it be maintained that he has these qualifications or that he is always permitted by the exigencies of his work to spend sufficient time on safety measures? And again compare the working miner in the Transvaal and in England. In the latter country he is, as already pointed out, a man who is the superior in mining knowledge of the local ganger, while in the Transvaal the working miner is an untutored savage who rarely stays long enough in a mine to learn more than the most elementary precautions. If then our high accident rate is due principally to the ignorance of our white and native miners, one would naturally conclude that the best means to improve matters would be to dispel that ignorance. In respect of the native this is a well-nigh hopeless task although something may be done even by having the native instructed, through the "boss-boys" and the interpreters, of the dangers of such things as handling explosives, drilling into misfires, entering the stope before the ganger, entering skipways and putting his head out of a skip. In respect of the white man I do not expect much will be gained by attempting to improve his knowledge by lectures or similar methods, since he will not attend them, but a great deal would be effected if a more thorough system of practical training were insisted upon than is the case now. The suggestion has been thrown out that the Mines Department might assist in this direction by making the mining qualifications for a blasting certificate more severe, but for various reasons I do not think the time has yet arrived for such a step.

In the absence of an early improvement in the knowledge and experience of our white and native miners, one has to cast about for other remedial measures. I have already indicated one of these, which is that of placing more responsibility for safety upon men who have or should have a better knowledge of mining than the average miner. Other methods have been suggested by the author and those who have joined in the discussion on his paper, and the body of the Mining Regulations may be looked upon as a compendium, in condensed and legal phraseology, of what experience has dictated in this respect.

In conclusion, I wish to enlist some of your sympathy for the much abused miner, who frequently finds himself between the devil and the deep sea when the means for complying with regulations are denied him by the shift boss or other official. He has to run the risk either of being found out by the Inspector of Mines, with or without an accident, or of losing his job if he remonstrates too strenuously with the mine official or if in an enquiry he gives evidence against the official. Unnecessary risks are run in such cases, and when anything goes wrong the miner is pretty sure to incur some penalty. Managers and mine captains can do a great deal of good, not only in reducing the accident rate, but also in improving the contentment amongst their men, by impressing on shift bosses that in the matter of complying with regulations, the miner should have no cause for complaining that the means for doing so have been denied him. The same rule applies to safety in this respect as to efficiency, namely, that the miner must not have to wait for or be deprived of his tools. As I have already indicated, the Rand can furnish numerous examples of the maxim that efficiency and safety on a mine march together.

Mr. M. H. Coombe (*Member of Council*): The mining members of this Society are often twitted with not coming forward and taking their share in the production and discussion of the papers brought before the meeting.

There are doubtless various reasons for this attitude, the chief being I think the very conservative profession or calling of which they are members. From time immemorial this reluctance to discuss the particulars of their calling in public has been so. There are so many phases of their work, so many interests radiating from a common centre, that one may easily say too much or too little. The new comer into the mining ranks is apt to speak rather freely, but as time goes on he becomes imbued with the conservative atmosphere that surrounds all ranks of underground men, and gradually draws into his shell so far as publicity is concerned.

But on the subject matter of the paper under discussion, a matter concerning the life and limb of the miner, it is really incumbent on the mining man to give his impressions and to give of his knowledge to the general fund of information with the hope strong within him, that from the general discussian circling around this paper, some practical lines of procedure may be evolved, which will lessen appreciably our present high mining accident mortality list. I may venture to state there is not an underground official on these fields, who is not deeply grieved and upset when a fatal accident takes place in his mine, be the victim black or white. I have had sleepless nights for a week together when a succession of accidents has unfortunately occurred on a mine of which I have had supervision, and underground men will bear me out in this. These accidents seldom occur singly; if there is one, three may be expected within a short period. One's nerves are unstrung, mind and body at a tension, and sleep is impossible. Doze one may, but the least sound, or a footstep, makes one sit up and wonder if another accident has occurred. That has been my experience, and I have heard many underground "bosses" remark the same. We have not, despite the many accidents, arrived at a callous or indifferent stage, but are keenly alive to the seriousness and gravity of the position and are most anxious to prevent their recurrence as much as possible. Hence I think we should contribute to this discussion.

"Speeding up" in some of our mines is an admitted practice, and certainly tends to accidents. The fault here does not lie so much with the management as with the policy that erects big mills and drops the stamps before sufficient development has been achieved to keep the mine ahead of the mill. Every ounce of rock is wanted, waste as well as reef. Pillars can ill be spared. Timber is expensive, and the underground boss at his wit's end to do forty things at once, chances himself and his mine to luck, and hopes to get ahead of the mill or die.

This, of course, is not mining: in fact mining is becoming a lost art on the Rand. It is simply "gouging" out everything to feed a monster that is not, and cannot be, satisfied. Smaller mills, more clean reef mining, less waste hoisting, and more left in the stopes, with more pillars and filling provided will conduce to more economical work, better dividends and less accidents.

Mr. Trevor touches a vital point in his contribution when he refers to the impossibility of the white miner forcing discipline on his natives. We have been much over-regulated in this matter, The white boss at one period of our mining history was held in respect and fear by his natives, and his orders were obeyed with alacrity and despatch. If they were not, the offending native knew what to expect. Because of their fear they respected him; he was a white man, superior to them, with power to punish, and he therefore had to be obeyed. To-day the white man's voice is of little account to them. They have ceased to respect or fear their underground boss. A real or imaginary grievance instead of being taken to the mine captain or shift boss for investigation, is trotted off to the compound manager, who in turn takes it to the manager, who, much worried, probably gives orders to "sack" the man. This action is of course a triumph for the native and does not make it any easier for the next man taking the gang over. In some cases the native will go to the police direct, or write to the Native Affairs Department, or his recruiter, and these people will, of course, gaily institute proceedings, not caring twopence for native discipline or the prestige of the white man underground. These are real evils, and result undoubtedly in many accidents for which the unfortunate white miner is held responsible, and over which, because of his lack of disciplinary powers, he has little or no control. Moral suasion has no effect on the native mind; underground, physical fear is the only lever that can move him.

Wherever mining men gather one will seldom fail to hear remarks passed about the scarcity of miners. I will venture the opinion, based on close observation, that not more than 15 % of the underground men in our mines to-day are miners. Why is this? Why does not the big cheque draw the miner? Well, there are other things that count to the self-respecting miner besides a big cheque. Say what we may, put whatever construction we may upon it; the conditions of work and service in our mines are not attractive, nay, they are most repulsive to the man who knows his work as a miner. He much prefers other countries where there is less chance of getting a big cheque, but where he is at least treated as a white man, and is not hounded from pillar to post at the behest and will of a stripling boss whose experience of mining dates back for perhaps 18 months or two years since he passed through his course at a mining school. Now in this connection I want to say something that may probably offend and hurt. I hope it will not, and I trust it will be taken in the spirit in which it is said, the spirit which wishes to arrive at conclusions to enable us to decrease our accident rate. In my opinion there are too many technically trained men, and not enough practically experienced men, in charge of underground operations. All the school training in the world cannot make a miner of a man. It cannot instil that instinctive knowledge of safe and unsafe ground which is inherited in some cases and acquired in others, as the result of long and arduous experience. The technically trained man enters the field of his work armed with advantages not held by the other, and in course of time, as he gains experience, he should be the more valuable man of the two, but without that experience, he is useless, a drone, and a source of irritation to men who are doing the work. For some time now on our mines it has become the practice to take the young technically trained man after passing through his course at the school, and give him right away a position on the mines, perhaps as sampler, assistant surveyor, and in some cases as shift boss. The legitimate aspirations of these men are to positions of command; they have been taught to expect that, they look upon it as their natural right, and in fact they are trained as officers of the industry. Thence from the outset, class distinction becomes impressed on their mind, and they enter on their work with an artificial division created between themselves and the miner. A man has to have a vast amount of natural commonsense to overcome the handicap thus placed upon him, for handicap it is. The student has still to learn mining after passing through his school: he cannot learn it but from the miner, and he cannot learn it as a sampler or surveyor or shift boss. He must learn it as a miner with the miners. Here is where the authorities have gone astray in this matter, and they are doing as much harm to their students as is being done to the miner. Before a technically trained man is put into a position of command underground, he should have from three to five years practical work with the men whom he one day aspires to control. If he is level-headed he will learn a lot of things to his eternal good. He will learn to know the men, to look at things from a miner's point of view, to enter into their feelings and get their aspect of the mine and the men who are controlling it. It will knock class prejudice out of him and teach him to judge and handle men when his time comes. It will also bridge over the cleavage that has arisen, and in most of our mines between the men and their officers there is a cleavage which is widening daily, which is more and more evident, which tends to accidents, and which will, if it continues, one day lead to great and serious trouble. This placing of young men in underground positions is also unfair and unjust to aspiring and experienced men, as the latter feel that the door of promotion has been closed to them, and also that as long as they remain on the Rand, they cannot become anything but machine men or runners of hammer boys. They see their shift bosses and mine captains in charge of big mines with numbers of lives under their control, possessing little or no mining experience, and they come to the conclusion it is better to "get out" to a country where conditions are better and the pay less. Thus one loses the very flower of our miners; the incapable man takes his place, and thus the percentage of accidents must go on increasing.

I do not agree with the author that "resueing" is likely to become a lost art. Neither will hand-stoping. No manager will replace hand-drilling with machines on economical grounds. When he puts machines in his stopes, he is driven to it by circumstances. Ore for the mill being the chief reason.

Our mines could be much more safely stoped than they are to-day. The hanging should be left entirely unbroken. I know a mine where it is the usual practice to cut the hanging above the stope track for room for the trucks, in place of cutting the foot wall. The hanging, and a good hanging too, is in some places cut into as much as three feet deep to make room for the trucks. Of course this is theoretical mining, and when falls of hanging occur and natives and men are crushed, the management will wonder how it happened. More filling should be done, and a stope face should not be allowed to advance more than 50 ft. without the roof being supported by filling in with timber or with pillars. A modified form of "resueing' could be followed, worked with machines equally as well with hand labour. Sand filling does not appear to be as successful as was promised. Of course one cannot fill sand into a working face, and it is at, and in proximity to, the faces where the men are at work, that the accidents take place.

To sum up my conclusions: better mining practice is needed underground. To get this more miners must be put in command of miners. To get the miners their conditions of work must be made better, the door of promotion must be opened again, their tenure of work made more secure, and more disciplinary powers over their "boys" must be given to them.

The experiment of making miners here has been tried and failed. Miners cannot be made in a school, they can only be secured by their doing the actual work and working with men who have been doing it for a living for years. To decrease the accident rate, to work our mines safer and as mines should be worked, experienced men must be had. The industry demands them, economy demands them and the safety of life and limb demands them.

Mr. Ralph Stokes (*Member*) (*Contributed from London*). At the close of Mr. Chilton's interesting paper on "Accidents in Transvaal Mines," and of Mr. Tom Johnson's informative comments in the November *Journal*, there appears a contribution to the discussion by Mr. W. S. V. Price, to the publication of which I must take strong exception. Mr. Price reverts to the old question of the training and experience required by shift bosses, and disposes of it in the usual dogmatic manner of those whose range of vision is so narrow and whose technical self-sufficiency so complete that it is only necessary for them to gauge a fellow-worker by the standards of their own perfection to commend him or condemn.

The animus inspiring Mr. Price's criticism is clear from the following quotation:—"It would almost seem nowadays as if anyone except a miner, is good enough to be a shift boss; we see samplers, timbermen, *surveyors*, *pipe-fitters*, *etc.*, in that position." The "pipe-fitter" is traditionally a man of poor status amongst skilled workers underground. Whatever his deserts, he is certainly one of the last to hope for direct elevation to a position of control. But he admirably serves Mr. Price's present purpose, which is simply that the mine surveyor may, by means of invidious association, be referred to with disparagement and contempt. Speaking broadly, the question whether any particular surveyors or pipe-fitters merit a shift boss's position depends upon their other mining experience and training, their energy, "level-headedness," and strength of character, which could no more be discussed in general terms than the ability of Rand miners to understand the influence of complex faulting.

To emphasise this comparison, I will again quote from Mr. Price's remarks. When trying to throw contempt on some particular mine overseer and shift boss, he says "In one instance, a converged fault was supported by bits of lagging-poles." How, pray, could anything support a fault? And what is *a converged fault*, supported or unsupported? No wonder Mr. Price was disgusted to see "bits of lagging-poles"—whatever that may really mean—put beneath this interesting freak of nature, for it should certainly have been sent to the University for the enlightenment of students—and professors. However, it is still no doubt possible for Mr. Price to give us sketches of the converged fault before it collapsed and of the system that should have been adopted to support it. Only by such definite illustrations of expert knowledge can our "pipe-fitters and surveyors" be made to realise their miserable ignorance and unworthiness.

In conclusion, I may say I am at a loss to understand how Mr. Price's comments passed censorship. It is regrettable to find the Society's technical journal being used as an inexpensive medium for the expression of petty personal antipathies. I am at least comforted by the belief that if any member had discussed the technical merits of "bricklayers, metallurgists, etc.," for positions of control in the reduction works, his impertinence would have received a more timely check.

Mr. B. C. Gullachsen (*Member*): In listening to the discussion on the author's paper, I notice there is one matter in connection with the appalling accident rate on the Rand mines which has not yet been brought forward, and which, in my opinion, is closely connected with it. I refer to first-aid to the injured and ambulance work. There can be no gainsaying that in the past far too little attention has been paid to this work, and that in many cases the lack of a knowledge of first-aid has been the cause of many minor accidents developing grave symptoms, in some cases even leading to fatal results. Had the miners on the Rand in the past years been efficiently provided with ambulances and first-aid appliances, and men trained in the correct use of them, there is no doubt that the death-rate which has prevailed on these fields would have been reduced in no little degree.

How often must it have happened that a simple fracture of a bone has been converted into a serious compound fracture by rough handling in getting an injured man out of a stope, or for the want of a stretcher underground; or again, how many lives may have been lost for the want of someone with the knowledge of how to stop arterial bleeding.

In the Mines and Works Regulations of Dec. 1, 1911, an effort has been made to rectify this matter of first aid and ambulance work in mines within the Union of South Africa; but are these new regulations sufficient or efficient, and are they enforced as rigorously as they certainly should be? Surely if by doing so the enormous death-rate can be reduced even a small percentage, then not only mine inspectors but everyone connected with Rand mining should do their utmost to have the new ambulance regulations carried out. Expense in this matter surely cannot be advanced as an excuse for a mine not being thoroughly equipped as required by the regulations.

In Regulation 277 (2b) of the Mines and Works Act, 1911, it is required that at every level on which ordinary mining operations are being conducted there shall be kept at least one plank stretcher of a pattern approved by the Government Mining Engineer. Why, may I ask, should a plank stretcher be all that is required underground, whilst on the surface, where the conditions for carrying the injured person are vastly better, a Royal Navy stretcher must be provided? To carry an injured man on a plank stretcher down or up a narrow stope would be nearly impossible, unless provided with straps and footrest in order to prevent the patient slipping off. Moreover, the plank stretcher is an instrument of torture to a badly injured man. For underground work canvas stretchers should always be used, either with iron or wooden sides, into which the patient can be securely strapped. A good example of such a stretcher is being exhibited at the Mining Exhibition now being held in this building. A stretcher of this pattern is also indispensable for bringing up injured persons in the skip in vertical shafts.

Regulation No. 277 (2b) also requires that three or more bandages shall be kept at every level on which ordinary mining operations are being conducted; but what about splints? Surely these are equally if not more necessary, especially underground, where they may be used as soon as possible and before the injured person has been moved, or rather moved only so little as may be necessary to get him into a place of safety; besides, why enforce the training of all the underground staff in first aid if you do not provide them with the articles necessary for first aid. The argument may be put forward that all the necessary articles are provided for on the surface, but anyone working in a mine, especially a deep level mine, must admit that quite a long period must elapse before the skip can be obtained and the articles brought underground.

There are other points in the new ambulance regulations which might be criticised, but the object of these remarks is intended to draw attention more particularly to ambulance work in connection with accidents in mines. It cannot be disputed that the Rand is far behind other mining centres as far as ambulance work is concerned. Surely this should not be so. Ambulance and first aid classes should be started on all the mines, and I put forward the suggestion that this Society should offer a trophy for competition between the mines ambulance teams. This would arouse interest in the work and be the means of providing a number of efficient first aid men at each mine Competitions in ambulance work are now held in nearly all mining centres the world over, and great interest taken in them, as witnessed at the National First Aid meeting held in Pittsburg, U.S.A., last October, when no less than 10,000 spectators were present, including President Taft, who presented the prizes to the winning teams. During his speech he remarked that "with all our mines having numbers of men trained in first aid to the injured and ambulance work, there is no doubt that the number of deaths resulting from accidents will be considerably diminished;" words that surely should also apply the mines of the Witwatersrand.

The President: On your behalf I wecome heartily Dr. Lindsay Johnson who wishes to address you.

Dr. Lindsay Johnson, M.D., F.R.C.S. (*Visitor*): I think it is perhaps somewhat out of place for me to address this meeting because I am not in any sense a miner, but as I have had considerable experience for the last thirty years at four of our London hospitals, I have naturally seen a large number of accidents, so that it may not be altogether out of place to give a few words of advice to the gentlemen here with regard to the wounded man. There are four causes of death when a man meets with an accident in a mine. (I am speaking of accidents pure and simple, not of asphyxia from CO or other poisonous gases). He may die of shock, of hæmorrhage, of exhaustion, or through some injury to the internal parts, such as the spinal cord, brain, or viscera. Now, one of the most important things that I can advise you to do when you come across any person who has met with an accident—a severe accident—in the mine, either from a punctured wound or the falling of a mass of rock on his body, is to leave him severely alone—in other words to do nothing. There is perhaps nothing so difficult as to do nothing. By doing nothing you may be doing a very great deal from a surgical point of view. As a rule the less an untrained layman does for a patient the better for that patient. An eminent surgeon who was in Ladysmith at the time of the Boer War, told me that after the battle of Elandslaagte he persistently operated on all patients who were wounded by a bayonet or bullet in the abdomen or chest. He said that in every case in which he operated the man died, so eventually, as he could not do worse than he was doing, he decided to leave the men on the field, attended by a nurse, with a blanket under them, and he said that the vast majority of men got well without any operation being done. I think his example may be followed with a certain amount of success when you meet with an accident underground. I will illustrate my statement further, by giving one or two cases which have occured to me during the last two or three weeks in South Africa. I was suddenly called while on duty for another surgeon, to a native boy who had been caught in one of the agricultural machines. The whole of the skin was stripped off the leg and thigh leaving the muscles unprotected, and the boy was in a horrible mess. A number of people had gathered round before I could get there, and they were about to lift him up and put him into a cart Fortunately, I stopped them and got a blanket fixed to two poles, laid him gently on the blanket, covered him over with a light sheet to keep away the flies, and had him carried, out of step, at almost a funeral pace, to a dwelling house. Here he was laid down and made an excellent recovery. Had he been put into the Cape cart and driven three miles to the hospital, as it was proposed to do, I am perfectly certain the man would have been dead before he could have got there. I did nothing but keep him perfectly quiet, and he got well in a fortnight, for these coloured gentlemen get well before we do—their nervous system not being sensitive as ours. You will remember the case of Stonewall Jackson. He was badly wounded by accident by his own men on July 4, 1863. They placed him on to a blanket attached to two poles and the men were carrying him to the field hospital. One of the men slipped on a stone and fell. The result was that uncontrollable hæmorrhage set in and General Jackson died. Yet the surgeons all declared that had they left him alone covered up on the field he would have recovered. One more instance. I was called the other day at Heilbron, where I was staying with Dr. Pienaar, by a policeman who came rushing in to say that a native had got crushed in a mill and had his arm torn off just below the shoulder. He was bleeding dangerously. Dr. Pienaar and I got into a motor car and did a record run. He amputated the limb, and made a good stump. He then got the man very carefully on to a stretcher made with a blanket, and had him carefully carried to the local prison and put him in the medical prisoners' ward. The following morning he was well on the road to recovery. Unfortunately, the magistrate was very officious, and he decided that it was a contravention of the regulations to allow the man to stay there and ordered him to be removed from the prison the next day. He was taken away in a common cart and before Dr. Pienaar was able to get to the place to which he had been taken, an hour and a half away, he was in a dying condition, owing to exhaustion. I am perfectly confident that if that native had been allowed to remain at the prison he would have made a good recovery with merely the loss of a limb. This shows you how fussiness may sometimes kill a patient. When you meet with an accident case you must remember two things. You must know how to deal with the cases. Shock is one of the worst complications. When you see a man in a clammy sweat breathing painfully, irregularly and in a shallow gasping manner, and when his pulse feels very feeble, rapid and fluttering, his pupils greatly dilated or contracted, or his sphincters relaxed, you may be certain that the man is suffering either from internal hæmorrhage or from severe shock. *Under these circumstances do not move him or rush for the usual remedy—the brandy bottle.* The best thing to do is to stop any visible bleeding by pressure, and then send to the top of the shaft for some ice for the man to suck. If you have a thermometer take his temperature. It will not go up as is generally assumed. On the contrary, you will probably find that his temperature will sink two or three degrees, and the lower it goes, the worse is the condition of the patient. If you find the man cold, yet bathed in sweat, his skin cold and clammy, lay him flat down, put some hot bricks to his feet and cover him with a blanket. *Do not move him, but ring up for the doctor and let him come down to see him.* If you are obliged to move him, sling him in a hammock or blanket with the head lower than the feet, and carry him very slowly to the nearest mine hospital, then the doctor will have no cause to blame you. I have learnt one lesson, which, if I had followed earlier in my life, would have saved me a lot of trouble and many solicitor's bills, and that is "when in doubt, do nothing." If you are obliged to remove the patient, never use wooden stretchers; they are cruel to the patient. Get a canvas stretcher or, if none be handy, a blanket and attach it to a couple of bars. If the hand or arm is injured sling it by a handkerchief from the neck. Do not let it hang down, for the blood will run to the hand and cause it to swell and throb painfully. When a man is suffering from shock or hæmorrhage do not give him anything to drink—just ice to suck. Keep him warm, and use the greatest care in moving him, lest you start the bleeding afresh. Do not forget that the bleeding may be inside the body and a man may bleed to death without a drop of blood being spilt.

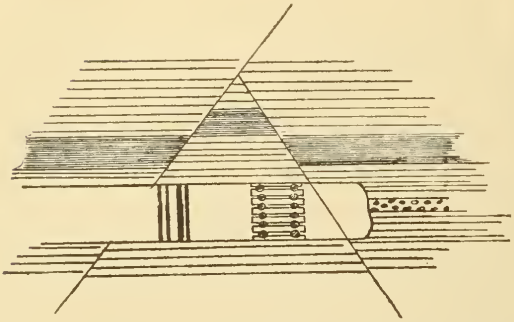
In conclusion, I most heartily concur with what Mr. Gullachsen has said in his admirable paper relative to the value of a short course of training on first aid to the wounded. I think that every miner should attend a course of ambulance lectures combined with practical instruction, and it would be well if such a course were made obligatory, as it would be the means of saving many lives, let alone money to the company.

Mr. M. H. Coombe (*Member of Council*): I wish to propose a vote of thanks to Dr. Johnson for his interesting and most useful remarks.

Mr. W. S. V. Price (*Member*): I wish most heartily to second the vote of thanks to Dr. Johnston for his helpful and timely remarks. For the benefit of Mr. Gullachsen I should like to say that there is plenty of material for splints underground so long as there are plenty of dynamite boxes.

Our President in his address this evening remarked that the Society was noted for plain speaking and hard hitting, and we have certainly lived up to our reputation this evening. Mr. Coombe thought that mining members are, or rather have been, diffident in expressing their views; but the discussion of this most important paper has shown that to-night, at any rate, they are holding their own. I had not intended speaking this evening, desiring to wait a little longer to hear the views of others, yet I must say that Mr. Stokes must have been put to considerable trouble to find personal animus in my remarks in November. I certainly did not desire to set off one class against another underground. It has remained for Mr. Stokes to sneer at a pipe fitter as the lowest class of man in a mine. I have known a pipe fitter who attended the lectures in the college here for two years, and came out very well in his examination; but he realised his deficiency in *practical* knowledge. I said nothing against the surveyor who is a surveyor, or the timberman who is a timberman; those I alluded to were those who were shift bosses without the actual experience necessary for such a responsible position, and I have no need to take back a single word that I said that night. On such a vital topic as this, where so many men's lives are at stake, it would be most wrong to talk platitudes, or mince words about the undoubted gross incompetence of some who hold this position on some of the mines, and who are undoubtedly a source of danger to themselves and all about them.

Mr. Stokes criticised my reference to a "converged fault." I do not claim to know everything about a mine, nor do I pose as the last word in mining and I may have misnamed it. (Mr. Tom Johnson: No.) I will show you what I mean. I am not acquainted with Mr. Stokes, and I am sorry he is not here to-night or he could see what I mean, and learn what lagging poles are, viz., short lengths of small timber up to, say, 4 in. or even 5 in. in diameter, which are generally used to support the stuff used in stulls, etc., in stopes, as well as in a variety of other such work. I said that the largest stick under this fault was 7 in., and the only likely way to support the fault was by building pigstyes under it. I give here a rough sketch of one showing how ground should have been supported. Any *miner* will know what they are and how to build them. They



This line should be shown coming down to the right of the broken ground.

are filled with waste rock—which in the case I referred to, was being broken up and sent to the surface, presumably to help to keep the mill going. In fact in this, as in almost every other measure that makes for safety underground, the thing would pay for itself, certainly it would do so ultimately.

In conclusion, I must repeat that I have never said a word against the surveyor who has learned mining, or the miner that has learned surveying, and I honour any man who has the grit and ambition to get on, but not at the risk of other people's lives.

Mr. E. R. Bawden (*Member*) (*contributed from Nyasaland*): I am not surprised to learn from Mr. Chilton's paper that accidents are so common in the Transvaal mines.

In American, Australian and British mining districts, the miner learns his duties from youth by practical acquaintance, being taught by skilled miners of usually 40 years' experience. On the Rand this is not the case. Unfortunately, hundreds of men who have never been underground, come to the Rand, spend a few weeks in shifting dirt or going down with a friend; somehow he obtains a blasting certificate and blossoms forth as a "miner." We know that a man can be a miner only after long and varied experience. This mushroom type of miner then takes charge of hammer boys, and then one wonders why accidents happen. Another example commonly found on the Rand is the young miner of 20 or 22 years of age. Hitherto he has been engaged in an European mine with older men, but upon his arrival in South Africa, he finds himself in charge of 30 or 40 natives and a working place with the same responsibilities as an older man; it is through these men that accidents occur.

I believe that in addition to a blasting certificate, experience should be necessary before a man takes charge of a stope. Men coming to the Rand should bring references as to their ability and experience. Miners certificates of 1st and 2nd grades should be given according to experience, and a miner, who through neglect or incompetence, causes an accident should have his certificate endorsed, and should not be given the preference of better places in the mine. A register of the age and experience of miners should be kept, so that one could see to whom the accidents are attributed, either the trained miner or the "Jack-of-all-trades."

I certainly think the hanging wall inspector a superfluous appointment which would only tend to increase carelessness on the miner's part. In regard to Table IV, I think that the average man to-day is not up to the mark of the old and past school of miners. As the mines increase yearly in size underground, the pressure is greater and consequently there are more falls of hanging walls. In stopes in which the hanging is hard and smooth, it is often a good practice to put in a few poles and regularly examine them to see if they are "taking any weight," as such poles will often serve to give a warning that the hanging is on the move. Stulls or pigsties should be more frequently adopted than at present. Timbering in the stopes is often left to natives with bad results.

Generally, I attribute the large number of accidents to the general inefficiency of a certain type of stoper who holds the usual blasting certificate and who never was, and probably never will be, a miner. Insist on getting miners, not stopers, and accidents will be less.

Mr. W. Cullen (*Past-President*): Members of the Society must have been astounded by the figures which Dr. Macaulay placed before them at the last meeting, and as I am particularly gratified that they prove my main contention, which is, that the more changes there are on a mine the higher is the accident rate. It would have been interesting if the changes had been further analysed under the respective heading of:—(a) Managers, (b) Staff, (c) white employees, (d) black employees. The last, D, is the one element which, under existing circumstances, cannot be controlled.

I think I would be safe, however, in asserting that taking the mines as a whole, and putting the blacks with the whites, the average period of employment must be under one year I do not know of any other industry which could stand this. What a field for economies!

I thought it would be interesting to carry Dr. Macaulay's analysis a little further, and I therefore asked the Government Mining Engineer to give me the figures for the changes in management on the Witwatersrand mines over a number of years. I have to thank him for courteously acceding to my request. He informs me that there are approximately 100 mine managers on the Witwatersrand, and among them there have been the following changes during the past 3 years, viz. :

		1909.	1910.	1911.
January	...	5	5	3
February	..	8	5	4
March	...	2	10	3
April	...	14	9	11
May	...	8	5	7
June	...	10	8	8
July	...	10	8	7
August	...	5	6	7
September	...	6	8	5
October	...	12	11	9
November	...	10	8	8
December	...	7	8	8
		—	—	—
Total for year		97	91	80

These figures speak for themselves. Apropos of this discussion I have received the following communication from a miner who has been working on these fields for the past 10 years. I know him well, but naturally cannot hold myself responsible for his statements. He is a well educated man, thoroughly competent in every way and is still employed at his avocation notwithstanding his having contracted phthisis :

"There is altogether far too much hustle and shuffling of the men on the mines at present. During the last few years it has been quite a common thing to see two and even three new mine captains in the space of six months. All this is naturally most detrimental to the safety of those employed below the surface. In nine cases out of ten a new mine captain means a new lot of men, and it takes weeks, if not months, for him to understand his surroundings thoroughly, because the 'hanging,' 'faulting' and 'dykes' vary from mine to mine, as do other conditions. In the majority of cases he uses the same system of overseeing in the new mine as he did in the old. For instance, not very long ago I went through a stope with a new mine captain. I drew his attention to the 'hanging,' telling him that it required some heavy timber at once. He picked up a hammer, sounded the 'hanging,' and said that my statement was nonsense as it sounded like a bell and was well pillared. I had worked on the mine for many years and knew it thoroughly. That 'hanging' caved in in two days, but, fortunately, there was no one in the stope at the time.

"Then again there have been wholesale changes and dismissals for years, and it is getting worse instead of better with the result that there is very general discontent amongst the miners. Take the case of a married man, a first class miner who has brought out his family at great expense from the old country to live on the mine premises. He has been many years on the property, has an excellent reputation, and, so far as he knows, has given every satisfaction. Quite unexpectedly the mine captain is fired. A new one is appointed, and he is not on the job for many days before he commences to discharge—the old hands as a rule having to go first. He wants to get in his own class of miner, and, most likely, he has promised jobs to a certain number of his friends before he came, so dismissals have to take place whether the men are old servants or not. What then is the use of us bringing out our wives and families ?—a man is never certain of his job no matter how competent he may be. If he does by chance get them out he very often sends them back as soon as he has enough money scraped together. He feels more free after he has got rid of them, but if work is unsteady he gets careless, he degenerates and becomes in the end a 'has been,' wandering from mine to mine not caring whether he gets a job or not. Of course miners of this class become a real source of danger.

"We have very few of the good all-round miners left now—the sort of men who knew everything from the sinking of a shaft to the laying of a track, and the pity is that nine out of every ten of them have phthisis. I discussed this accident question with one of the oldest and most capable miners on these fields the other day —one who knows them since not long after the start—and his opinion was, that considering the number of incompetent miners and learners, and the constant changes in the mines, it was a wonder that there were not more accidents than there are. I told him that I was surprised that he was not an overseer yet with all his experience and education, but he said that the 'Big Bugs' now prefer the School of Mines' men for shift-bosses, as they are good at figures and have got bags of wind to climb up and down ladders and steep stopes ! He himself had none left.

"Although I am a fair scholar myself and have been a shift-boss for many years, I have no ambition now because I am done for—got phthisis. Such are the conditions to-day. Old friends pass away on the mines and we are afraid to ask off in case we get fired. Men frequently die and are buried without half the employees knowing anything about it. If men, as Mr. J. Vaughan says, were given a chance of attending funerals, the terrible state of affairs would soon be realised. At present it is not. It is all hustle and bustle and the filling of battery-bins is the first and most important consideration. What a difference from the coal-mining districts at home. There you see old men of over 60 plodding along gaily to the pit. If you enquire they will tell you, probably, that they have been mining about the place since they were lads of 13, content, on the whole, knowing they are certain of their job till old age incapacitates them. This is what we want on the Rand. Once a boss or a manager gets a good class of steady men let him stick to them and they will stick to him. Men should not be sacked for trivial causes, but should be given a warning and a chance. Some miners live from 4 to 10 miles from their work, many of them because they cannot help it. To say the least of it, it is unfair to send a man home who has been cycling or walking for an hour before he reaches his work, because he turns up for once five minutes late at the shaft. In short, the whole thing comes to this: if there were less discontent there would be better and more efficient work, and, certainly, fewer accidents."

ZINC DUST TESTS.

By W. J. Sharwood, Ph. D., A.R.S.M., M.I.M.M.
(Corresponding Member of Council).

The increasing interest among members of this Society in zinc dust as a precipitant has suggested the publication of details of a few tests which have proved of some use in examining and comparing samples of it. Unfortunately no very definite criteria have as yet been established for valuing zinc dust for precipitating purposes, but the following will serve to give some idea as to the availability of a sample for the cyanide plant. It should be dry and fine; nearly all—say 95%—should pass a sieve of 200 meshes to the linear inch (aperture ·003) while very little—say not over 1% or 2%—should remain on 100 mesh (0·006 in.), and practically none on 50 or 60 mesh. It should not show any signs of caking, or contain lumps which do not break up at once when shaken on a sieve. Generally speaking an ash grey colour is a more favourable indication than a decidedly blue or whitish cast; but colour cannot be taken as a positive criterion. The presence of a little lead—say 2% to 3%—is an advantage. Laboratory tests made with pure zinc, and with equal weights of alloys containing up to 5% lead, in the same state of fine division (filings between 0·006 and 0·003 in.), have invariably shown more rapid and more nearly complete precipitation of gold and of silver in the case of the lead alloys—although the weight of actual zinc present was less. The presence of a little zinc oxide is apparently without detrimental effect, except as diminishing the percentage of active metal. Attempts to remove the oxide by solvents inevitably lead to the dissolving of more or less metallic zinc, and possibly to the destruction of the finest particles. No injurious effect has been traced to the small amount of cadmium usually present, and other impurities are generally negligible in amount. A number of samples which have proved satisfactory in actual use have shown from 85% to over 95% metallic zinc (*i.e.* zinc in the metallic state), most of them over 90%.

Various published analyses have shown metallic zinc ranging from less than 30% up to 92%, zinc oxide from traces to over 50%, and cadmium from traces up to 1·3% in Belgian, and 1 to 2·5% in Silesian samples. A number of analyses from various sources are to be found in Schnabel's "Metallurgy" under Zinc, Ingalls' "Metallurgy of Zinc and Cadmium," and in "Mineral Industry" Vol. II. p. 258. Some of these show carbon up to 3% or 4%, and silica or insoluble up to 9% or 10%, but in general very little of either will be found, and only traces of iron or arsenic. A few of these analyses are summarised in the annexed table, together with partial determinations of some samples used in gold precipitation.

Typical Analyses of Zinc Dust.[*]

	Metallic Zinc.	Zinc Oxide.	Lead.	Iron, Cadmium, Carbon, and Insoluble.
1 Belgian ...	91·5	Little	0·5	—
2 ,, ...	88·74	6·60	2·5	2·1
3 ,, ...	79·16	11·26	1·9	7·2
4 average Belgian	84·5	9·3	1·6	4·0
5 Silesian ...	84·5	4·9	4·3	6·2
6 ,, ...	88·5	7·4	2·0	—
7 ,, ...	—	88·2	4·0	3·0(SO_3 4·1)
8 American ...	29·6	57·5	Trace	(Insol. 9·6)
9 American, good quality ...	96·0	—	—	—
10 European, imported into U. S. good quality .	90·9	4·0	2·7	—
11 European, similar, fair quality ...	85·0	—	2·0	—
12 European, similar, fair quality ...	94·6	3·7	0·2	(Iron 0·3)
13 used at Deloro, Canada ...	—	3 to 5	1·74	—

Various authorities state the specific gravity of zinc as from 6·85 to 7·21, of zinc oxide from 5·6 to 5·78, and zinc carbonate 4·3 to 4·5. Samples of zinc dust have shown results from 6·82 to about 7·0, while the material itself weighs from about 2 to 2·5 gm. per cc., or about 120 to 155 lb. per cubic foot, depending largely upon the extent to which it has been shaken down. No connection has been proved between the apparent density and the quality, but at least one "heavy" sample was found to give unsatisfactory precipitation.

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[*] Nos. 1—4 by Firkel, 5 to 7 by Steger, 1—7 quoted by Ingalls; 7 and 8 from Schnabel, intermediate products in smelting; 10, analysis supplied by importers, quality excellent; 13, Harland, *Journal Soc. Ch. Ind.* 1897 p. 968; 9, 11, 12 original; Nos. 9 to 13 used in cyanide precipitation at various plants.

The following serves to illustrate the desirability of avoiding contact with moisture or moist air in the transportation and storage of this material. On a carload of imported zinc dust used some years ago in Montana the railroad charges were found to be based on a considerably greater weight than was indicated by the weights of the several casks invoiced as originally shipped from Europe. The total weight found by the daily use of the material bore out the railroad figures, and confirmed the suspicion that the discrepancy was due to oxidation from contact with moist air. This was further proved by analyses for oxide and metallic zinc, a sample taken near the staves of one cask showing about 10% more oxide than another from the middle of the mass. In another case, where very bad precipitation was noted, the material from the bottom of one barrel was lumpy and yielded 39% zinc oxide, while a sample from near the top of the next barrel opened contained 13·5%.

Estimation of Lead in Zinc Dust.—Weigh 10 gm. into a large beaker (400 to 600 cc.) moisten with water, then add 200 cc. water and stir well, then 10 cc.[*] strong sulphuric acid, stir well and cover. Stand beaker on a hot plate and warm to 60° or 70° C ; if action is very slow at the start add a single drop of very dilute platinic chloride solution ; stir at intervals. After action becomes slow add 10 cc. more acid and continue warming until nearly all the zinc has dissolved. Decant the solution upon a filter, wash the residue in beaker two or three times with water, passing washings over the filter.[†] Rinse residue back from filter into beaker. Dissolve residue with a little nitric acid,[‡] transfer to a casserole and rinse beaker into casserole. Add 5 or 6 cc strong sulphuric acid and evaporate to dense fumes. Cool, dilute with water, stir well, cool thoroughly, filter, wash well with water containing 5% sulphuric acid.[*] Dissolve the lead sulphate in hot ammonium acetate solution, and wash filter thoroughly with the same solution. Dilute this solution with hot water (about 80° C) and titrate the lead with standard ammonium molybdate, using 1% tannin or ferrocyanide solution as outside indicator.

The molybdate is standardized on pure lead foil, treated in the same way as the lead residue, from the zinc dust.

If the ferrocyanide titration is substituted for the molybdate method, the same standard solution will serve for both lead and zinc.

Estimation of Zinc Oxide in Zinc Dust.—Approximate method based on Solubility of Zinc Oxide in Ammonium Chloride and Ammonia.—Solution Required.—250 cc. water ; 70 gm. ammonium chloride ; 150 cc. strong ammonia water, sp. gr. 0·90.

Method.—Weigh out 1·00 gm. zinc dust, put into a stout test-tube holding 35 to 50 cc. Use a disc of soft rubber packing $\frac{1}{16}$ in. thick as a cover for the tube, or use thumb as cover while shaking. Add 25 cc. of the prepared solution, cover tube and shake well for exactly five minutes. Agitation must be violent enough to ensure all lumps being broken up. Throw mixture promptly on a 9 cm. or 11 cm. filter, rinse tube and wash filter with hot water, receiving in a 200 or 250 cc. beaker. Dilute filtrate with hot water to about 100 cc. Add a drop of phenolphthalein indicator, then HCl till about neutral, then 5 cc. strong hydrochloric acid. Heat to 80° C and titrate zinc with standard ferrocyanide, using uranium acetate or nitrate as indicator.

Zinc found in solution × 1·245 = zinc oxide in 1·0 gm..

This method is rapid and gives fairly approximate results, which are closely comparable if constant conditions are maintained in all tests. The results are slightly high, for the reason that metallic zinc is itself slightly soluble in the solvent employed—but there is no known solvent for the oxide which will not also attack the metal. Tests with freshly prepared zinc filings, passed through a 200 mesh sieve, show that the error from this source is negligible if only five minutes contact is allowed with the above solution. Carbonate of zinc dissolves in the same way as the oxide.

Estimation of Total Zinc.—Dissolve 1·000 gm. of dust in dilute hydrochloric acid, heating moderately. Make up to 200 or 250 cc. in an accurate flask, remove 50 cc. with an accurate pipette. Add 5 cc. strong hydrochloric acid, and about 2 gm. ammonium chloride, dilute to about 200 cc. with hot water, heat to 80° C and titrate with ferrocyanide.

The ferrocyanide solution is standardized on pure metallic zinc, dissolved in hydrochloric acid and treated as above.

Determination of Metallic Zinc in Zinc Dust.—Many methods have been worked out, most of which are not altogether satisfactory. Those which depend on measuring the volume of hydrogen evolved by the action of acid are hampered by the slow rate at which zinc dust dissolves in dilute acid, and by the readiness with which hydrogen diffuses through rubber tubing and leaky connections. A number of the methods proposed are given in Sutton's Volumetric Analysis. (10th. ed., p. 382-383).

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[*] The use of 25 cc. of strong hydrochloric acid, in place of 10 cc. sulphuric acid, gives equally satisfactory results.

[†] The two filtrates from the lead and lead sulphate contain the iron, cadmium, and zinc ; they may be mixed, made up to a definite volume, and these metals determined in suitable aliquot portions.

[‡] If any considerable residue (silica, carbon, etc.) remains at this stage after treatment with nitric acid, it may be filtered off, washed thoroughly and determined. If in this case a white residue remains after washing it should be treated with hot ammonium acetate solution, and this should be added to the acetate solution obtained later.

I. Very good results are obtained by determining (1) *total zinc* and (2) *zinc in the form of oxide*, by the methods above given. The difference is assumed to be *metallic zinc*. This has the advantage of requiring only one standard solution—potassium ferrocyanide—for both titrations.

II. Reduction of chromic acid, by subjecting the zinc dust to the action of potassium bichromate acidified with sulphuric acid. The bichromate should be present in considerable excess of the calculated amount necessary for the zinc used. The residual chromic acid is determined by titrating an amount equal to that originally taken, and the solution after the zinc has acted, or aliquot parts of each, either :

(1) by means of KI, titrating the liberated iodine with thiosulphate. (Sutton, p. 383) or

(2) by means of a standardized solution of ferrous sulphate.

(3) by adding an excess of a standardized solution of ferrous sulphate and completing titration with standard bichromate.

III. Reduction of ferric chloride or sulphate and titration of the ferrous salt obtained by standard permanganate or bichromate. Sutton (p. 383) recommends using 0·5 gm. zinc dust with 7 gm. of ferric sulphate and titrating one-fifth of the solution with permanganate. Presumably 14 or 15 gm. of ferric alum would answer equally well.

IV. The reduction by the zinc dust of a solution of iodine in potassium iodide has given fair results, but it is only safe to use rather small quantities, say 0·1 or 0·2 gm., titrating the residual iodine with thiosulphate.

The following are results obtained with a sample of zinc dust :—

Method I.	Total zinc	91·6%
	Zinc as oxide (and carbonate)	12·9%
	Metallic zinc by difference ...	78·7%
Method II.	Metallic zinc by reduction of bichromate (3)	79·6%
Method III.	Metallic zinc by reduction of ferric alum	77·1%

Whatever *reduction* method is employed for direct determination of the metal, the metallic zinc can be calculated thus :—

1 cc. $N/10$ standard solution = 0·00327 gm. zinc.

Determination of Precipitating Efficiency of Zinc Dust.—Solution required.—A solution of potassium silver cyanide is prepared by dissolving 10 gm. silver cyanide (AgCN) and 5 gm. of "99%" potassium cyanide in a little water and diluting up to 1,000 cc. It is then adjusted by addition of a little more KCN or AgCN until the solution indicates from 0·12% to 0·15% *free* KCN by titration with standard silver nitrate. The titration is best made by using a 10 cc. or 20 cc. sample, adding 1 cc. 2% potassium iodide and a slight excess of ammonia, the end-point is then sharper. Or 15 gm. of pure crystallized $KAg(CN)_2$ may be dissolved in a litre of water and 1·5 gm. KCN added

Method.—Weigh out 0·500 gm. zinc dust into a 300 cc. beaker. Add a few cc. water and stir till zinc is well mixed, then pour in 250 cc. of the prepared solution, stirring vigorously. See that all lumps are broken up, and continue stirring for fully five minutes. Stir occasionally (at least every 10 minutes) until the end of two hours from the addition of the solution. Then filter upon an 11 cm. filter, wash precipitate thoroughly, sprinkle with test lead, wrap it carefully in the paper, place in a scorifier with about 20 gm. test lead, burn paper cautiously in muffle, scorify 5 minutes, cupel at low temperature and weigh silver.

Milligrams silver obtained from 0·5 gm. zinc × 0·0606 percentage precipitating efficiency.

By taking 0·303 gm. zinc dust instead of 0·5, each milligram of silver is equivalent to 0.1% efficiency. Instead of cupelling, the silver may be dissolved in dilute nitric acid and titrated with standard thiocyanate with ferric indicator, using the whole or one half of the solution for titration.

Cc. $N/10$ KCNS used for 0·5 gm. × 0·654 = percentage efficiency.

This test was devised by Mr. A. J. Clark in 1904, with the idea of utilizing a reaction similar, and taking place under similar conditions, to those actually prevailing in the every day use of the zinc dust in the cyanide plant. While the results obtained by this method are not entirely satisfactory, they have afforded, during seven years' experience, a better indication of the results to be expected in practical work on the large scale, than those got by any other system of examination so far tried. Satisfactory samples of dust usually show efficiencies between 40 and 60%, and sometimes as low as 35%,—100% being the standard calculated for the complete replacement of pure zinc by silver. A better method,—one which completely eliminates the personal factor in manipulation—is desirable, but a simple determination of the metallic zinc is not sufficient; the test must automatically take into account the physical condition as well. Attempts to carry out the test in a mechanical shaker, and determining the silver left in solution after filtering after a specified time of contact, have always given much lower results than are obtained as above.

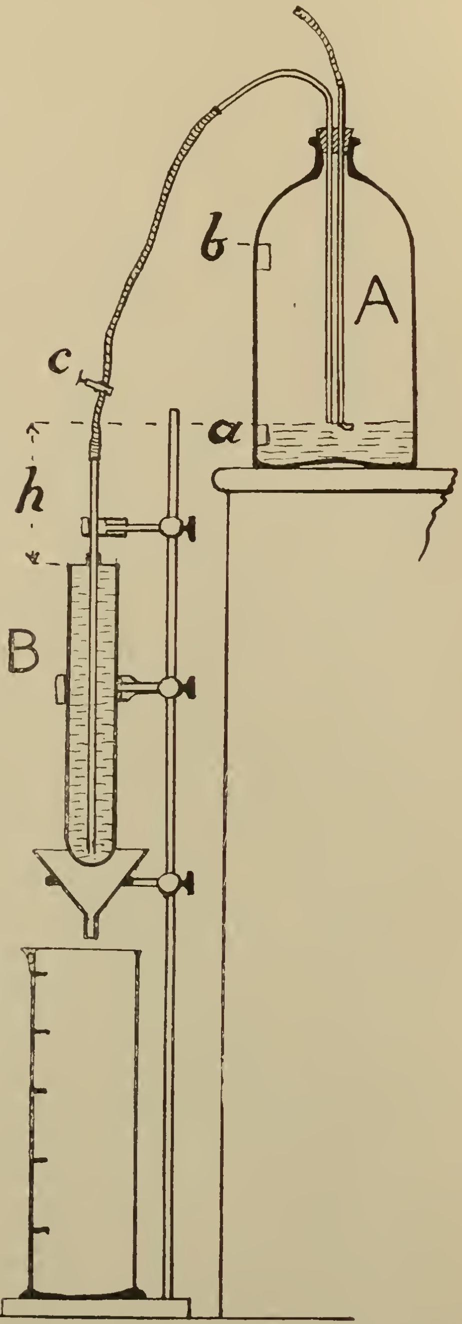
Mechanical Tests of Zinc Dust.—Sizing Test.—When sized with sieves of 100 and 200 meshes to the linear inch, using a sample of 20 to 50 gm., good samples of zinc dust usually show at least 95 and often 98 or more per cent. passing the 200 mesh, and not more than 1% remaining on the 100 mesh. This sizing is irrespective of caked lumps, which should never be found in freshly opened barrels.

Elutriation Test.—This is based on the separation of the finest particles by a rising stream of water in a vertical tube, the velocity being about 1 cm. per second. It promises to throw some light on the fineness of division of the portion passing 200 mesh, which, as above indicated, ought to include nearly all the material.

A convenient arrangement is that shown in the figure. A 2500 cc. acid bottle A is fitted with a 2-hole rubber cork, with a Mariotte tube reaching nearly to the bottom and turned at a right angle at the lower end, and a siphon outlet tube reaching to about the same level thus giving a constant level water supply. A mark is placed at *a*, the level to which the siphon empties the bottle, and another at *b* to indicate the surface when 2000 cc. additional water have been put in. The working cylinder B may be a tall 250 cc. measuring-glass, but it is better to use a large tube with rounded end, say 1·25 in. diameter by 12 or 15 in. long, with a $\frac{1}{4}$ in. glass tube placed centrally in it and connected with the siphon by a rubber tube with a spring clip *c*, both being fastened to a stand by clamps. A large graduated cylinder and funnel are placed below. The net area of the cross-section of the working cylinder is best got by sticking two labels or making two marks on it 20 cm. apart, then filling with water to the lower mark and measuring the number of ccs required to fill it to the upper, the central tube being clamped in place. Then the area of cross-section (= volume in cc. ÷ height in cm.) is obtained once for all. To adjust the velocity fill the bottle to the upper mark, fill working cylinder, open spring clip and take time with a stop watch while 1000 cc. flows out.

$$\text{Then } \frac{1000 \text{ cc.}}{\text{Area} \times \text{No. sec.}} = \text{velocity in cm. per sec.}$$

By raising or lowering the working cylinder, the velocity can be adjusted, the effective head *h* being the vertical distance of the top of the cylinder below the level of the Mariotte tube inlet, the velocity varying approximately as the square root of *h*. The clip on the siphon should be removed or kept wide open during the test, and care should be taken that the rubber connections are not sharply bent or allowed to obstruct the flow. When the point is found at



which an upward velocity of exactly 1 cm. is obtained, or some point sufficiently near, it is well to mark the position of the cylinder and clamp or measure the height h, so that this position can be reproduced, all the other adjustments remaining the same.

Method.—Put 20 gm. of zinc dust in the cylinder, moisten with a little water, then add 50 or 100 cc. and stir till all air bubbles are removed, then fill cylinder with water. A wire stirrer is used, reaching to bottom. The bottle being filled to b, open clip and start the flow. Use the stop-watch as a check on the velocity, timing the first 1000 cc. The wire stirrer must be used at first to prevent packing; most of the fine passes off with 500 to 1000 cc. of water, but it is best to let 2 litres flow through. Then allow the remaining material to settle a few seconds, rinse into an evaporating dish or casserole, decant off water, wash twice with a little alcohol, dry and weigh the residue. This may be tested with 100 and 200 mesh sieves, if not sifted before making the test, and further examined with the microscope. By commencing the test with a velocity of say 0·8 cm. per second, and repeating with the same material at 1·0 and 1·2 cm., several distinct fractions may be obtained from the material passing 200 mesh. The comparatively coarse particles are of several kinds:

1. Impurities, usually chips or sand.
2. Round particles of zinc, or aggregate made up of round particles, looking like grape-shot.
3. Feathery or fern-like aggregates of minute crystals of zinc, often forming very thin platy masses.

A separation of the coarser particles can of course be more simply made by repeatedly stirring with water in a beaker and decanting, but it is impossible to obtain comparable results unless a constant upward velocity is maintained in some such way as indicated.

The accompanying figures illustrate some of the variations shown in the appearance of certain samples of zinc dust when examined under the microscope. The upper row, highly magnified, show the appearance of the medium portion of the "through 200" fractions of three such samples, the very finest part (under $\frac{1}{20000}$ in.) having been removed by elutriation. As may be seen, these are almost perfect sphericles, and under the highest powers appear to have clean smooth surfaces, and to consist of nearly pure metal. The coarser particles, over 0·003 in. remaining on a 200 mesh sieve, are shown below, and vary greatly in appearance. In several of the more satisfactory samples, illustrated by Nos. 1 and 3, many of these larger particles have a rough, coke-like surface, but some generally exhibit branching crystalline, fern-like aggregates, and other aggregates resembling miniature bunches of grapes. It is possibly only a coincidence that the unsatisfactory sample (No. 2) had an unusually large proportion of the fern-like forms, and also showed more relatively large sphericles in the "through 200" portion. The extreme minuteness of the fine is shown by comparison with the equally magnified filing (No. 5) which had passed a 200 mesh sieve, and the granulated zinc, also passing 200 mesh (4A), the method of preparation of which is not known, but which had a very low precipitating efficiency.

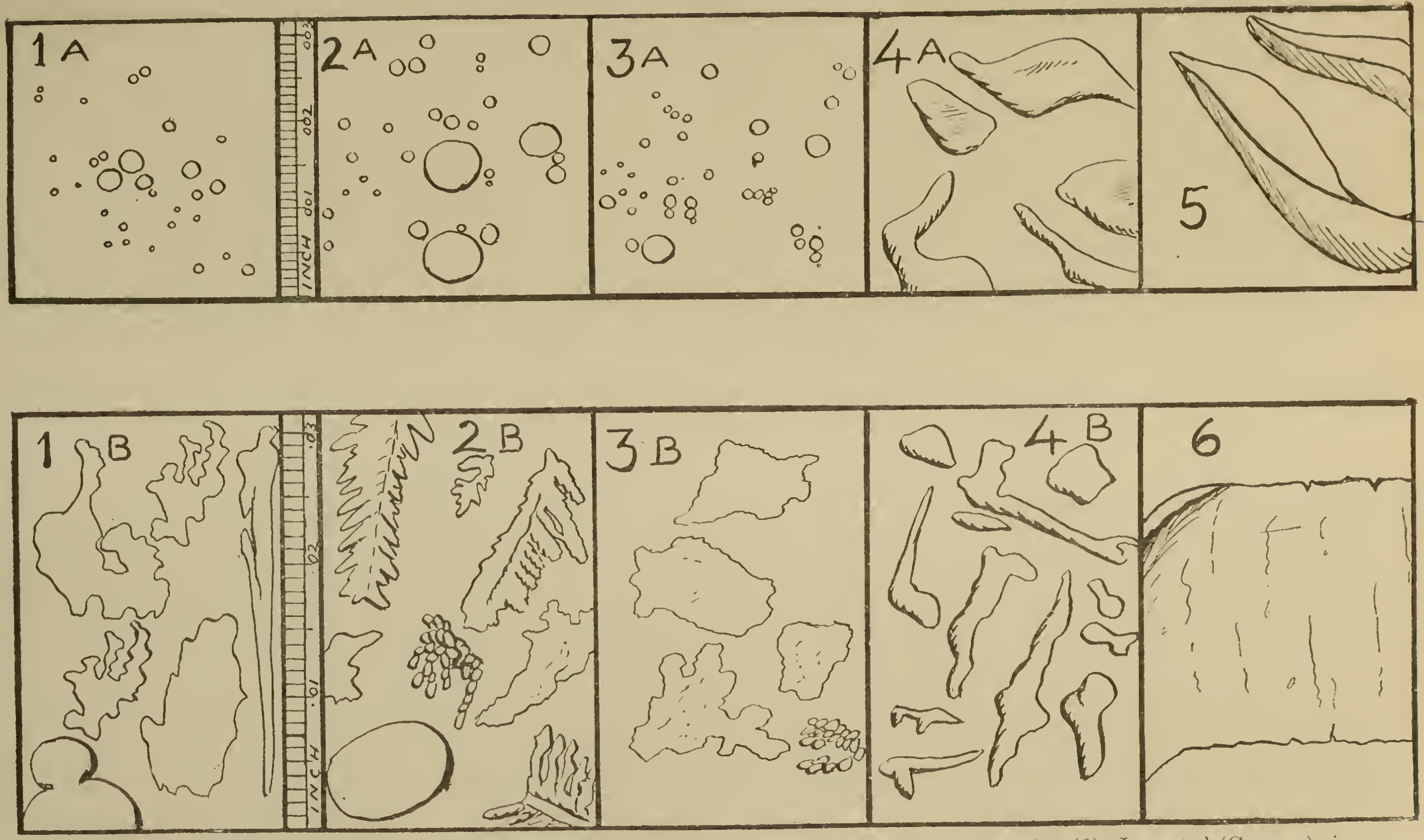
Surface Exposed by Zinc Dust.—As the efficiency of this precipitant is mainly due to its fine division or extended surface, some calculations of the latter may be of interest. To calculate the surface exposed by zinc dust we may make use of the fact that the total surface exposed by one ton (2,000 lb.) of any material in uniform grains—either cubes or spheres—is $\frac{2304}{g\,d}$ sq. ft., g being the specific gravity of the material and d being either the edge of the cube or the diameter of the sphere, expressed in inches. Hence, the density of zinc being very nearly 7, assuming it to be in the form of spheres of a uniform diameter of d inch, we have:—Square feet surface exposed by 1 lb. $= \frac{0{\cdot}165}{d}$.

Thus the portion which will *just* pass a 200 mesh ($d = 0{\cdot}003$ in.) sieve will expose only 55 sq. ft. per lb., while the material averaging only 0·0001 in. in diameter—which constitutes a considerable proportion of the whole, as may be seen by a microscopic examination of the finer portion—exposes 1,650 sq. ft. per lb. It is interesting to compare this with zinc shavings. If we assume this material to consist of a continuous rectangular strip, of width a and thickness b, we have:—

Area exposed by 1 lb. zinc shavings $= 0{\cdot}055 \left(\frac{1}{a} + \frac{1}{b}\right)$ sq. ft. If, instead of being rectangular, the cross-section of the shaving is a parallelogram of angle A, the above expression must be multiplied by cosec A; in this case the thickness b is measured at right angles to the width a.

For instance with shavings $\frac{1}{32}$ in. wide, and of various thicknesses, the surface exposed would be:—

Thickness	$\frac{1}{800}$	$\frac{1}{1000}$	$\frac{1}{1500}$ in.
Sq. ft. per pound ...	45·8	56·8	84·3



Description of Figures.—(1). Imported (German), zinc dust, satisfactory, precipitation good. (2). Imported (German) zinc dust, unsatisfactory, precipitation not good. (3). Domestic (U.S.) zinc dust, satisfactory, precipitation good. (4). Granular zinc (5). Zinc filings, passed 200 mesh sieve. (6). Zinc shaving, extra fine, about 1/50 in. wide.

The number of particles (spheres) of zinc dust in a pound, assuming the specific gravity as 7, is $\frac{7{\cdot}544}{d^3}$. Hence, if the diameter is 0·001 in., 1 lb. will contain 7544 million particles, while at an average diameter of 0·0005 in. there will be 8 times as many. This enables us to get some idea as to the intimate nature of the mixture obtained if an "emulsion" of zinc dust is prepared by distributing it with absolute uniformity through the solution to be precipitated. Suppose $\frac{1}{5}$ lb. of zinc dust is used per fluid ton (55,296 cu. in.) of solution, or 1 in 10,000 parts by weight, the diameter of the particles being taken as 0·001 inch. From the above figures it appears that each cubic inch will contain about 27,300 particles, if uniformly distributed, whence the average distance between the particles will be approximately $\frac{1}{30}$ in. If the particles are of half this diameter and equally distributed there will be 8 times as many to the cubic inch, and half the distance apart.

All samples shown in upper row highly magnified, (see scale;) those marked (A) are carried off by a stream of water rising 1 cm. per second, but finest dust has been removed. Those marked B remain on a 200 mesh sieve, and are less magnified, see scale.

Addendum.—The foregoing notes were just ready for mailing when the October *Journal* reached me, containing a communication from Mr. M. Thornton Murray on the same subject. The sample of zinc dust examined by him appears to have been somewhat coarser than the average, as it is usual to find, in the material of less than 0·003 in. diameter, a good deal of dust not over 0·0001 in. diameter, and some considerably finer, so that I should be disposed to consider 0·0005 in. more nearly an average for this fraction.

Mr. Murray's assumed gravity (6·92) would give $\frac{0{\cdot}1665}{d}$ instead of $\frac{0{\cdot}165}{d}$ which I have assumed for the sq. ft. per lb., or approximately 1% more surface than my formula would indicate. Thus, instead of 144·8 sq. ft. per lb. as calculated by him for a mean diameter of 0·00115 in., my formula gives 143·5 sq. ft. Actually I have found a range from 6·82 to 7·0 in the specific gravity of samples of zinc dust tested.

Mr. Murray has, however, seemingly obtained his "mean diameter" by multiplying the assumed diameter of each of his fractions by its percentage weight, and dividing the sum of these products by 100, which is not mathematically correct, for the reason that there are far more particles in 1% of the finer grades than in 1% of the coarse[*], if we consider a given weight of the total material. This makes his calculated mean diameter somewhat too high, and the resulting surface correspondingly too low. This may be seen by computing the surface for each fraction separately, as below, from Mr. Murray's data for percentages and diameters of the various fractions, the finest fraction alone giving a larger surface than he has computed for the whole.

Fraction between	Mean diam. of fraction (d inch)	Sq. ft. for 1 lb. of fraction 0·165/d.	Per cent. of fraction.	Sq. ft. for 1 lb. of total zinc.	Sq. ft. for 0·90 lb. metc. zinc.
·0197 & ·0100	0·149	11·07	0·16	0·018	0·0162
·0100 & ·0060	0·008	20·63	2·42	0·05	0·045
·0060 & ·0030	0·0045	36·67	2·88	0·106	0·0954
·0030 & 0·0	0·00085	194 1	94·54	183·50	165·15
Total ..	0·0009[†]	183·67	100·00	183·674	165·3

If the average diameter of the particles in the finest fraction be assumed at 0·0005 in. instead of 0·00085 the surface becomes approximately 330 sq. ft. for each pound of that grade, or 312 ft. for a pound of the total dust, or about 288 sq. ft. of zinc reckoning 90% of metallic zinc, the surfaces exposed by the coarser fractions being negligible in comparison.

As others have pointed out in a previous discussion on fine grinding,[‡] the results of the calculation depend almost entirely upon the average diameter assumed for the material passing the finest sieve used, which we have no means of precisely determining.

The President: In closing this very successful and interesting meeting, I wish to thank all for attending and especially those who contributed to the evening's interest, I regret exceedingly it will be some time before I can preside again.

The meeting then closed.

A Review.

MINERALOGY, by Dr. F. H. Hatch, published by Whittaker & Co., London and New York. Fourth Edition, entirely re-written, with 124 illustrations: pp. ix. and 253. Price 4s.

It is now 20 years since the original edition of this valuable little work was published, and it certainly can be said that the author has done a service in enlarging it and bringing it up to date. We think that it will probably have no rival in regard to convenience of reference, a point in which the larger general treatises on this subject fail, usually from being overloaded with detail and with information on those rarer substances which have not as yet received useful application. The present work, however, despite its small bulk, omits nothing of industrial importance however remote. The book is well got up, and apparently quite free from the usual typographical errors. There are a few curious chemical mistakes in the book: thus on p. 174 goethite $HFeO_2$ is stated to contain 81% of iron instead of 63%, and on p. 219 both KNO_3 and $NaNO_3$ are stated to contain nitrogen peroxide, a belief which was exploded some 60 years ago. Again, the author still prefers the old-fashioned discrete formulae in many cases where the modern constitutional or structural formulae are better, *e.g*, calling Fe_3O_4 the epitrioxide of iron, instead of $Fe(FeO_2)_2$, the ferrous salt-derivative of goethite considered as ferrous acid $HFeO_2$, just as spinel is magnesium meta-aluminate $Mg(AlO_2)_2$. Similarly vanadinite should be written $Pb_2ClVO_4 + Pb_3(VO_4)_2$, and dechenite $Pb(VO_3)_2$. Also the chemical names given to many of the silicates on p. 77 are unsatisfactory: thus andalusite is not an orthosilicate, because Si : O : 1 : 5. It is really a basic metasilicate $(AlO)_2SiO_3$. The English of the book is sometimes rather slipshod, *e.g.*, the end of par. 3, p. 66: but these criticisms are of a trifling character and do not affect the value of the book. (J. M.)

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[*] This is obvious from a consideration of an extreme case. Suppose a mixture of equal weights of bullets of two sizes, half the weight being balls of 1·0 in diameter, and half of 0·10 in. The mean diameter is not 0·55 in. (as would be found by a calculation based simply on distribution by weight) since, for every 1 in. ball there are 1,000 balls of a diameter of 0·10 in.

[†] Approximate true mean diameter calculated back from total surface by formula (0·165/d = sq. ft. per lb.)

[‡] See this *Journal*, Vol. VII., pp. 120, 207, 265, 289.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DISCOVERY OF A NEW ELEMENT, PROBABLY OF THE PLATINUM GROUP. — "Well authenticated reports from Western Canada state that Mr. Andrew Gordon French, a native of Glasgow, and well-known as a metallurgist, has discovered in the Nelson district of British Columbia an absolutely new element, which is expected to prove of high commercial value. The new element, it is added, has been named 'Canadium' in honour of the Dominion. It is, Mr. French states, of the platinum group, and will probably fill one or other of the two vacant spaces among the noble metals in what chemists know as 'the periodic table.' These spaces are between osmium and tungsten, and between ruthenium and molybdenum, and it is considered likely that 'Canadium' will take its place between the second two.

'Canadium' is of a beautiful white colour, and of brilliant lustre, apparently suitable for gem settings and similar purposes. It was found in the dyke rocks in the Nelson district, running in quantities from a few pennyweights up to three ounces to the ton. It occurs pure in semi-crystalline grains, and in short rods about half a millimetre in length by one-tenth of a millimetre in thickness. Mr. French also found it in metallic particles in the form of scales in platinum bearing ores. These particles, which have a bluish white colour, contain the metal alloyed with a volatile substance which may be osmium, as it was dispelled by the blowpipe flame, leaving a brilliant bead of 'Canadium.' It is not platinum, ruthenium, palladium, nor osmium. It is much softer than these, and is quite easily melted by the blowpipe. The new metal does not become tarnished by lengthened exposure to damp, and is not oxidised by continued heating in the blowpipe oxidising flame. It is soluble in HNO_3 and HCl, and in aqua-regia without residue, and its solution in HNO_3 yields no precipitate with chloride of sodium solution. This differentiates it from silver. It is not blackened by lengthened exposure to moist sulphuretted hydrogen or to alkaline sulphides, tests which also prove that it is not silver. It is not blackened by tincture of iodine, and its nitrate solution is not precipitated by iodide of potassium. These negative qualities, Mr. French states, differentiate it from palladium.

Its melting point is somewhat lower than fine gold and silver, and very much lower than that of palladium. It is electro-negative to silver in dilute acid solution. These characteristics definitely show it to be a new metal of the palladium group.

The new element was found in a platinum mine previously discovered by Mr. French, and from which platinum, iridum, palladium, rhodium, and osmium have been obtained."—ANDREW GORDON FRENCH.—*Chemical News*, Vol 104, No. 2716, p. 283. (J. G.)

ROUGH DETERMINATION OF CALCIUM OXIDE IN LIME USED IN THE CYANIDE PROCESS.—"Calcium carbonate and magnesium oxide are useless as protective alkalis, so that a quick method of determining calcium oxide is of importance. The author has obtained results of sufficient accuracy for technical purposes by titrating a 'milk' of the sample and water with oxalic acid and phenolphthalein until the pink colour returns only very slowly. The slow return (if any) of the pink colour is due to magesium oxide, but the end point for the calcium oxide is easily recognised."—L. W. BAHNEY, *American Institute of Mining Engineers*, Oct., 1911: *Mining and Engineering World*, Nov. 4, 1911.—*Journal of the Society of Chemical Industry*, 1911, p. 1389. (J. G.)

CARBON MONOXIDE DETECTOR.—"The apparatus comprises a glass vessel through which the air to be treated is forced by a rubber bulb, a strip of paper moistened with palladium chloride solution being suspended in the vessel. When the vessel has been filled with the air under test, the cocks are closed and the time is observed at which the test paper begins to darken and is completely blackened. The apparatus very quickly (in one minute or less) indicates the presence of quantities of carbon monoxide exceeding 0·1% and will detect (after about eleven minutes) the presence of as little as 0·01%."*—R. NOWICKI, *Oesterr. Zeits. Berg-und Hüttenw-*, 1911, *59*, 587-8.—*Journal of the Society of Chemical Industry*, 1911, p. 1336. (J. G.)

OCCURRENCE AND DETERMINATION OF NICKEL IN FERRO-VANADIUM.— "Samples of ferrovanadium of different origin were found to contain nickel, two for example containing 2·85% and 0·91% respectively,

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[* This appears to be the same as the Simonis instrument.—ED. COMM.]

whilst three others were free from nickel. Brunck's rapid method of determination (*Journal of the Society of Chemical Industry*, 1907, 643) gave accurate results: 1 gm. of the finely pulverised material is dissolved as far as possible in 30 cc. of HCl (sp. gr. 1·12), the solution oxidised by means of nitric acid (sp. gr. 1·4, added drop by drop) and evaporated, and the residue heated to destroy nitrates. After cooling, it is re-dissolved in HCl (sp. gr. 1·12), and the solution filtered from the silica, which is washed with hot dilute HCl. The filtrate is titrated with 20 cc. of tartaric acid solution (1 : 5), made faintly alkaline with ammonia, and the nickel precipitated with dimethylglyoxime."—H. SCHILLING, *Chem.-Zeit.*, 1911, *35*, 1190.—*Journal of the Society of Chemical Industry*, 1911, p. 1317. (J. G.)

DETERMINATION OF NICKEL IN ALLOYS.—"Referring to the dimethylglyoxime process (*Journal of the Society of Chemical Industry*, 1911, 549) the authors state that nickel may be precipitated in the presence of iron, aluminium and chromium, provided the solution contains sufficient tartaric acid to prevent the precipitation of these metals by ammonia. In a series of determinations upon a steel containing about 1% of nickel, the result afforded by direct precipitation was practically the same as that obtained by precipitation in the iron-free solution. It was also found that, after precipitation with dimethylglyoxime, nickel may be determined volumetrically by dissolving the precipitate in an excess of standard sulphuric acid and titrating back with standard potassium hydroxide, the end point being shown by the appearance of a faint yellowish colour in the previously colourless solution. The acid is standardised against pure nickel which is precipitated with dimethylglyoxime and titrated as described."—S. W. PARR and J. M. LINDGREN, Trans. *American Brass Founders' Association*, 1911, *5*, 120-124.—*Journal of the Society of Chemical Industry*, 1911, p. 1391. (J. G.)

DETERMINATION OF NICOTINE IN TOBACCO. "The tobacco leaves are first dried at a temperature of between 60° and 70° C., or in a desiccator, to remove all moisture. The ribs are then removed and the leaves crushed in a mortar to a fine powder. Twenty grams of this fine tobacco powder are weighed out in a porcelain dish, 10 cc. of the NaOH solution added (6 gm. solid NaOH dissolved in 40 cc. of water and made up to 100 cc. with 96% alcohol), and the contents of the dish mixed, transferred to a Soxhlet apparatus, and the nicotine extracted with ether. After extraction for two hours all the nicotine may be assumed to be in solution. After removing the ether the residue is nicotine, chlorophyll, etc. To the residue add 50 cc. of NaOH solution made by dissolving 6·4 gm. NaOH in 100 cc. water. Distill in steam until the distillate measures 500 cc. The distillate contains all the nicotine. Titrate 100 cc. of distillate with $N/10$ sulphuric acid, using cochineal as indicator, until violet blue turns onion red. 1 cc. $N/10$ sulphuric acid indicates 0·0162 gm nicotine. Repeat and average number of cc. of sulphuric acid used. Let x = average number of cc. used for each titration. Then x × 0·0162 × 25 = percentage of nicotine."—MARSHALL LUNDIE.—*S.A. Journal of Science*, Vol. viii., No. 6, p. 190. (J. G.)

NOTE ON THE APPROXIMATE ESTIMATION OF STARCH BY IODINE.—"The following method has been found to give approximate results in mixtures containing starch not previously heated:—Such a quantity of substance is worked on as will contain about 0·1 gm. of starch. This is heated to 190° C. with about 5 cc. of glycerine for 5 minutes. The solution of starch in glycerine is then diluted to about 50 cc., and filtered with aid of filter pump and steam jacket. An ordinary German filter with platinum cone is better than a hardened filter. The cold solution is next precipitated with a strong solution of iodine in 10% KI solution. The precipitated iodide of starch should be granular. After standing it is collected on a fluted filter (previously moistened with water containing 1 + KI), then washed with boiling 90% alcohol. The filter is next spread on a glass plate, and the precipitate washed into a platinum dish with a jet of boiling water. The contents of platinum dish are boiled to expel iodine, and afterwards evaporated over water bath to constant weight, and corrected for ash. A sample of starch gave 80 against 83·7 theory. A mixture of starch and fat-free mustard gave 43 against 41·8 theory. A sample of flour showed 70% of starch. The highest accuracy is not claimed for this process at its present stage, and it entirely fails with bread and cocoa. In the former the precipitate cannot be filtered off, and in the latter theobromine or some other substance appears to be precipitated."—LESTER REED.—*Chemical News*, Vol. 104, No. 2,715, p. 271. (J. G.)

RAPID DETERMINATION OF ZINC (IN ORES, ETC.)—"From 1 gm. to 3 gm. of the finely powdered material are heated with 10 cc. of nitric acid and 30 cc. of hydrochloric acid; after dilution, 50-60 cc. of concentrated ammonia are added in order to precipitate the silica, iron and lead. After quickly boiling, cooling, and diluting with dilute ammonia (1 : 3) to a volume of 300 cc., the solution is filtered and 100 cc. of the filtrate are treated with bromine water to precipitate the manganese. Without filtering, about 1 gm. of granulated aluminium is added to the ammoniacal solution to precipitate the copper. 5 cc. of a solution of sodium sulphite and 1 cc. of dilute ammonia are then added, the solution is boiled, filtered, and the precipitate washed with a dilute ammoniacal solution of sodium sulphite The filtrate after neutralising with hydrochloric acid and adding 10 cc. excess of dilute hydrochloric acid (1 : 3) is titrated by Schaffner's method, using a solution containing 21·63 gm. of potassium ferrocyanide, and 14 gm. of sodium sulphite per litre, and an indicator containing 9 gm. of ammonium molybdate per litre which gives a deep red brown colouration with free hydrochloric acid and potassium ferrocyanide. The analyses of ten varieties of zinc ores and products are given, the results obtained by the above method not differing from those by the usual gravimetric method by more than 0.5%."—K. VOIGT, *Z. angew. Chem.*, 1911, *24*, 2195-8.—*Journal of the Society of Chemical Industry*, 1911, p. 1390. (J. G.)

DETERMINATION OF SULPHUR IN PIG IRON AND STEEL.—"The iodine solution used in the titration of the sulphur, after precipitating the latter as a sulphide, is well known to be far from being stable. As solutions of permanganate keep their strength much better, it is advantageous to prepare for each determination a fresh solution of iodine by adding standardised permanganate to a solution of potassium iodide, in presence of dilute sulphuric acid. A solution is prepared containing 30 gm. of potassium iodide, and 10 gm. of sodium bicarbonate per litre (the sodium bicarbonate is necessary to avoid hydrolysis of the iodide solution). 10 cc. of this solution acidified with dilute sulphuric acid, are treated with a measured quantity of the permanganate solution, and the liberated iodine is titrated with thiosulphate, the strength of the latter being then adjusted till 1 cc. equals 1 cc. of permanganate. The determination of sulphur in iron or steel is carried out as follows :—The borings are heated with hydrochloric acid in the usual apparatus, and the gases, after passing through a wash bottle, are led into 50 cc. of ammoniacal cadmium chloride solution (20 gm. of cadmium chloride, 400 cc. of water, and 600 cc. of ammonia of sp. gr. 0·960). The cadmium sulphide is filtered off, washed and placed together with the filter in a flask in which 10 cc. to 20 cc. of the iodide solution, 25 cc. of dilute sulphuric acid, and (from a burette) a sufficient quantity of permanganate, have previously been placed. After shaking, the excess of iodine is titrated with thiosulphate, using starch as an indicator. On the liquid becoming colourless, it is titrated back with a few drops of permanganate till the blue colour just reappears."—H. KINDER, *Stahl u. Eisen*, 1911, *31*, 1838-9.—*Journal of the Society of Chemical Industry*, 1911, p. 1388. (J. G.)

DETERMINATION OF PHOSPHORUS IN PIG IRON AND CAST IRON WITHOUT SEPARATING SILICA.—"Four grammes of the sample are dissolved in 60 cc. of nitric acid of sp. gr. 1·2 with gentle heating. The solution is made up to 100 cc. mixed and filtered. In dissolving, care must be taken to exclude even traces of HCl, otherwise gelatinous silica may separate. Also, the solution should not be evaporated to a volume of less than 30 cc. Of the clear filtrate 25 cc. representing 1 gm., are pipetted off for the phosphorus estimation. (Portions of the remainder may be taken for the estimation of manganese and for the determination of combined carbon by the colour method.) To complete the oxidation of the phosphorus, 2 cc. of potassium permanganate solution (40 gm. per litre) are added, and the solution boiled for 2 minutes. Following this, the solution is cooled by immersing in running water for 10 minutes. The excess of permanganate is destroyed by the addition of a little sodium peroxide. To complete the solution of manganese peroxide a second addition of sodium peroxide is generally necessary, and the action may be assisted by the addition of 1 cc. of concentrated nitric acid. No more than 1 cc. of nitric acid should be added, otherwise the phosphorus results will be too low. After the manganese peroxide has all gone into solution, the excess of hydrogen peroxide is destroyed by heating, and the phosphorus is then precipitated at a temperature of about 40° C. with ammonium molybdate solution. The time occupied in bringing the estimation to this stage is 45 minutes. The phosphomolybdate precipitate is collected on a weighed filter, dried for $\frac{1}{4}$-$\frac{1}{2}$ hour, until the point of the filter paper has become dark blue-green. Results obtained by this method are given which are in close agreement with those obtained by the long method involving the filtration of silica—Ledebur's instructions being followed in the latter method."—E. R. E. MÜLLER, *Chem.-Zeit.*, 1911, *35*, 1201-2.—*Journal of the Society of Chemical Industry*, 1911, p. 1315. (J. G.)

METALLURGY.

SLIME FILTRATION.—"Much has been written concerning an accurate definition of the term 'slime,' but no comprehensive definition seems to be generally accepted. An excellent study of the nature of slime, by R. E. Ashley, was published in the *Mining and Scientific Press* of June 12, 1909. The accepted present practice in milling work is that all material in a pulp finer than a 200-mesh screen is considered as slime. The definition that a slime is the unleachable portion of a mill-pulp is also in use. A more comprehensive definition than the foregoing is that a slime consists of a mixture of sand finer than 150 or 200-mesh screen with an amorphous clay-like material, consisting principally of hydrated aluminum silicate.

The general method of slime-treatment is to agitate the slime with a cyanide solution for a sufficient time to dissolve the gold, and then, either to filter off the surplus solution and displace the remainder with water, or to thicken the slime by settlement and decantation, and then to filter and displace the remaining solution by water. The mechanical appliances in use for filtration are grouped as follows ;

These filters are continuous in action.

I. Suction-filters, or filters in which a vacuum is used to accelerate filtration.

A. Appliances using a thin slime-cake and practically continuous in their action. (Oliver and Ridgway filters.)

B. Appliances using a thick slime-cake and intermittent in their action. (Moore and Butters filters.)

II. Pressure-filters, or filters in which hydrostatic head, compressed air, or pumps are used in order to secure greater pressures than are possible with a vacuum-pump.

These filters are intermittent in their action.

C. Ordinary filter-presses.

D. Sluicing filter-presses (Merrill filter-press).

E. Filtering-chambers or cylinders ; filters in which the filtering-basket is enclosed in a cylinder. (Burt, Kelley, and Sweetland filter-presses.)

III. Centrifugal filters, or filters in which centrifugal force is used to separate solution from slime.

The filters in section I and II, with the exception of the Ridgway, employ vertical filtering-surfaces. The Oliver makes use of a revolving cylindrical surface as a filtering surface. Centrifugal filters are in process of development and have not as yet secured any foothold in gold and silver metallurgy. It is not improbable, however, that some comparatively simple filter based on the use of centrifugal force will be perfected, and will successfully compete with the other forms. At present the suction filters are in greatest use. Of the pressure-filters, the ordinary filter-presses have gone out of use, except as clarifying presses, and filters of groups *D* and *E* only are in use.

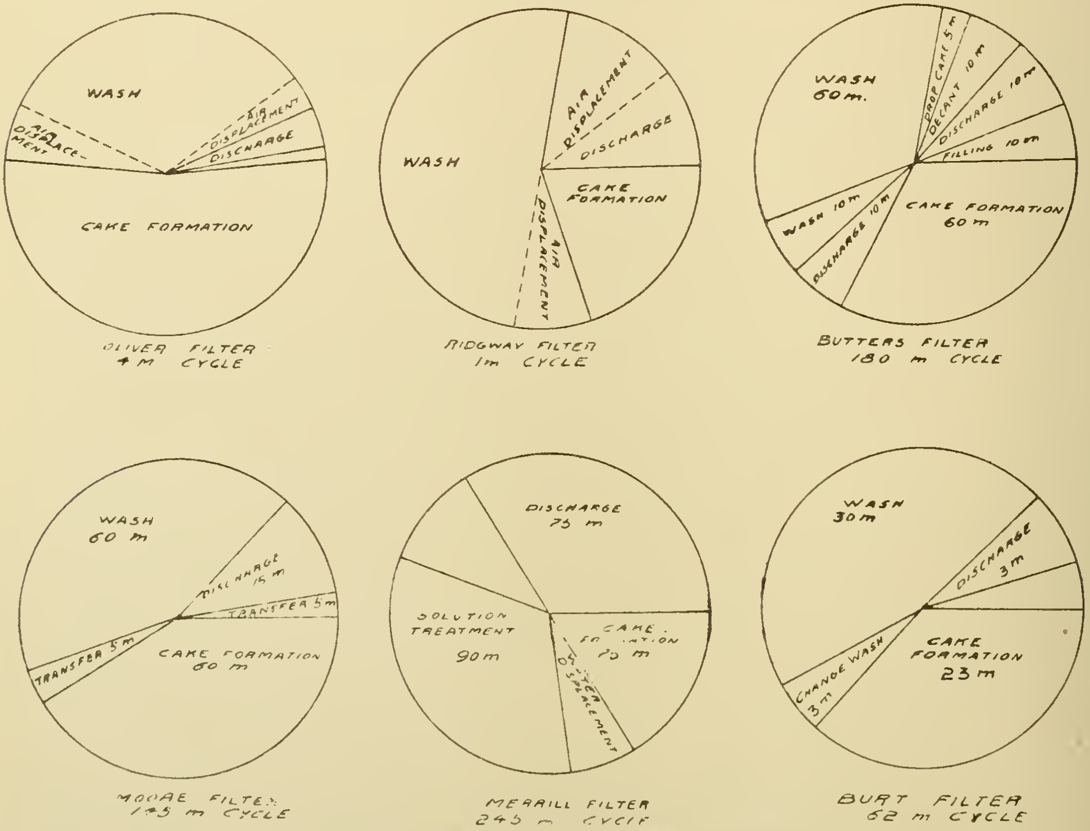
The development of slime-filtration is of interest. Filter-processes and filtering-beds in vats were first used. The filtering-beds were soon discarded and the filter-press systematically developed. The size of the press was increased, mechanical devices to facilitate discharge and decrease the proportion of labour required were invented and introduced ; but in spite of all this the cost of treatment in filter-presses remained high. In western America the filter-press never received much recognition, but in Australia filter-pressing was extensively introduced, and slime was successfully handled by this method. It remained for an American, Charles W. Merrill, to complete the last improvement in the filter-press. By the introduction of the sluicing system the slime-cakes can be washed out of the filter-cells and the press operated without opening or separating the filter-plates for each charge. This improvement reduced the labour and cost, and increased the effectiveness of the filter-press. The Merrill pres represents the culminating point in the filter-press line of development in slime filtration.

The Moore filter was the first suction-filter in the field, and introduced the idea of the canvas-covered filtering-cell immersed in the slime-pulp and utilizing suction to draw the solution through the walls of the cell and to build up a cake. The necessary transfers are made by lifting the filtering-basket out of the pulp. The Butters filter introduced the idea of a stationary filtering cell, and effected the transfers by pumping the slime-pulp and wash-water from the vat in which the filtering-cells were immersed. The relative merits of the two systems have been sufficiently discussed in the technical literature. Both the Moore and the Butters filters have reached a point where little or no further improvement seems possible. Like the Merrill, either one of these systems will satisfactorily meet the requirements of slime-filtration. The combination of the ideas involved in the filter-press and the suction filter is seen in group *E*, or the filtering-chambers. The Kelley, the Burt, and the Sweetland may be compared to a Butters filter installed in a pressure-tank.

The effort to secure a continuously-acting filter has resulted in two important types being developed, of which the Ridgway and the Oliver are the best known. Both of these filters utilize a comparatively thin slime-cake. Both operate very successfully, and compared with the thick-cake machines have decided advantages, briefly stated as simplicity of design; probably lower capitalization-charges for equal capacities; lower operating costs; and less attention required in the operation.

With the exception of the Oliver filter, the general method of operation of both suction and pressure-filters is the same. The slime-pulp is delivered to the filter in the proportion of one of dry slime to from three to one of solution. The pulp is forced into the cells of the pressure-filters and a cake formed against the canvas walls of the cells, the surplus pulp, if any, is withdrawn, and wash-water forced in until the contained solutions are displaced. The cake is then forced off from the canvas surface, either by water or air, or a combination of both, and sluiced out. In the vacuum-filter the filtering-cells are immersed in the pulp, a vacuum is formed, and a cake built up; the surplus pulp is then withdrawn either by lifting the filtering-cells out or by withdrawing the pulp by pumps, and the cakes are immersed in water for washing. In the Moore filter the cakes are discharged by forcing them off from the cell by water or air and dropping into a hopper for sluicing away; in the Butters the cake is forced off in the same way, but while still immersed in the wash-solution. The wash-solution is then withdrawn, either by decanting or pumping, and the slime-cake and the surplus wash sluiced out. The Oliver filter performs the operations of cake-formation, washing, and discharge in continuous sequence. Three steps may be designated as common to all these filters: cake formation, washing and discharge. The cycle of operations of the more common forms of filters is shown in the accompanying figure. Typical examples have been taken in each case.

The condition under which slime-cakes are forme and washed are the critical points to be considered



OLIVER FILTER
4 M CYCLE

RIDGWAY FILTER
1 m CYCLE

BUTTERS FILTER
180 m CYCLE

MOORE FILTER
145 m CYCLE

MERRILL FILTER
245 m CYCLE

BURT FILTER
62 m CYCLE

Cycle of Operation of Various Types of Filters.

Physical Analysis.

	Type.	Sand Slime 50 of Sand.	Fire-Clay Clay-Slime.	*A*	*B*	*C*	*D*	*E*	*F*	Size.
				Percent	Per cent.	Per cent.	Percent	Percent	Percent	mm.
Less than 2 minutes settlement: + 100 screen	0·10	0·43	22·3	4·5	1·70	2·3	6·2	2·3	6·6	0·17 —0·22
Less than 2 minutes settlement: − 100 + 150	1·30	3·65	14·0	6·9	9·0	12·30	11·3	4·46	6·8	0·11 —0·17
Less than 2 minutes settlement: − 150 + 200	1·20	6·60	4·6	7·3	9·0	13·0	8·35	7·0	10·7	0·05 —0·11
Less than 2 minutes settlement: − 200	25·0	43·80	3·3	39·2	43·70	34·0	38·05	49·54	68·5	
More than 2 minutes, less than 4 minutes ...	8·5	6·3	2·0	4·25	4·90	11·0	5·5	3·6	0·7	0·01 —0·05
More than 4 minutes, less than 8 minutes ...	11·6	8·4	2·2	3·6	4 60	6·7	6·2	3·8	0·65	0·005—0·02 0·005—0·01
More than 8 minutes ...	54·3	30·8	51·6	34·25	27·10	22·7	24·4	29 3	7·05	0·0 —0·005
Thickness of cake, inches	1·00	1·00	1·00	1·00	0·75—1·0	1·25—175	1·25	1·75	1·5—2	
Filtering rate: Building Washing	5 lb.	10	3	11·6 10·0	26 16	8·3	7·25	30		
167 per 100 sq. ft.										
Sp. gr. dry slime ...	2·62	2·67	2·57	2·58	2 62	2 62	—	2·684		

Chemical Composition.

	Percent		Percent	Percent	Per cent.	Per cent.		Percent
Silica	74	—	68	74	77·2	68	—	64·4
Alumina	10·9	—	22·9	12·8	13 9	15·8	—	13·5
Ferric oxide	1·1	—	1·1	1·4	1·9	2·0	—	2·9
Lime...	1·0	—	0 0	0·8	1·0	1·0	—	5·2
Magnesia	0·6	—	tr.	0·0	0 5	0·0	—	1·2
Aluminum silicate (Al_2O_3 $2SiO_2$), calculated ...	23·6	—	49·9	27·9	30·3	34·4	—	29·43

Details of Filter Plants.

	A	*B*	*C*	*D*	*E*	*F*	*G*
Type of filter	Butters	Butters	Butters	Butters	Merrill	Kelley	Butters
Number of units	2	2	2	2	3	1	1
Number of leaves per unit	60	95	168	72	64	10	100
Number of leaves	120	190	336	100	192	—	100
Area of leaf, square feet	100	91	100	92·6	41·4	—	100
Total filtering area	12,000	17,290	33,600	9·262	7,980	360	10,000
Tons slime per 24 hours	175	250	1,000	150	216	100	150
Tons slime per 100 sq. ft. per 24 hours ...	1·4	1·44	2·97	1·62	2·72	27·7	1·5
Slime pulp consistency, water : slime ...	3:1	3:1	1·5:1	2:1	2:1	1·1	2:1
Filter support...	slats	slats	matting	slats	plate	wire netting	slats
Thickness of cake, inches	1	0·75-1	1·25-1·75	1·25	1·75	1·5-2	1·5
Moisture in cake, per cent.	38	35	35	33	—	12	29
Time forming hours	1	0·5	1-1 33	1	0·66	0·03	2
Time washing, hours	1	1·0	1·41-1·66	1·25	0·41	0·20	2
Time Transfer, hours	0·75	0·83	0·59-0·51	0·5	3 01	—	1·83
Total time cycle, hours	2·75	2 33	3-3·5	2·75	4·08	0·5-0·75	5·83
Filtering rate per 100 sq. ft. per minute	11·6	26	8·3	6·83	30	—	—
Filtering rate per minute for wash ...	10	16·3	—	—	—	—	—
Tons of solution and wash per 24 hours...	848	1,356	2,000	350-400	328 340	*75	484
Canvas used, ounces	12	12	12	12	18	12	10
Life of canvas, months	8	6	41 est.	18	12	0·5-1·5	—
Frequency of acid wash, days	20	21	30	21	60	15	30
Approximate cost per ton slime ...	$0.18	$0.27	$0.075	$0.25	$0.238	$0.35	$0.119

*8 to 12 hours.

the discharge and sluicing away of the cake is a comparatively simple matter and requires no special comment. My experimental work was largely confined to suction-filtration, and pressure-filtration was only briefly studied. Experiments on the variation of the filtering rate with variable thickness of slime-cake both while building up and in clear water, indicate that the filtering rate during building up a cake is greater in the pulp than in clear water for thin cakes, while for the thicker cakes the reverse is true. The general effect of increase of pressure upon the filtering rates of cakes of varying thickness is to increase the filtering rate. This is more marked with the thin cakes, while with thick cakes the effect of an increase of pressure is to increase the density of the cake and thus reduce its permeability. With high pressures this effect is more marked, and indicates that a point would soon be reached where the increased pressure would result in decreased filtering rate. This is particularly true of slime containing a small proportion of sand, and much less so with slime containing a large proportion of sand. In the use of both the Moore and Butters systems, experiments should be made with different intensities of vacuum, for it may be found that a vacuum lower than the maximum obtainable with the available apparatus will give a higher filtration rate, and thus decrease the time for both building up and washing. Many experiments were made upon the effect of temperature on filtration, the character of the slime-cake, the effect of the character of the filtering surface and the method of its support, the rate of building up the cakes, and washing the cake. For details of these experiments and the curves showing the results attained, reference should be made to the original paper.

Data were secured from a number of slime-plants in Nevada through the courtesy of the different managers. The table shows the results of a number of physical analyses of slimes obtained from these plants. For purposes of comparison the type slime, the sand-slime mixture, the clay slime, and the filtering rates are included. The lower table gives the data of the slime-plants from which the mill-slimes in the upper table were taken.

The practical conclusions which may be drawn from the experimental work may be summarized as follows:—1. The proportion of clayey material in ores which are to be subjected to 'all-sliming' and filtration should be maintained at a minimum. 2. The slime-pulp should be as free as possible from sand coarser than 150-mesh, and as large a proportion of the pulp as possible should consist of material passing 200-mesh. 3. The slime-pulp before filtration should be settled as thick as possible consistent with ready handling by pumps and in pipes. 4. The temperature of the slime-pulp should be maintained between 20° and 30°C., or higher. 5. The temperature of the wash-water and the pulp should be the same. 6. Vacuum-pressures should be varied until the proper intensity for the given slime is obtained. 7. Where the very clayey slime is to be filtered, as much fine sand should be crowded into the pulp as it will carry without undue settling and clogging. 8. No. 10 canvas supported by slats gives the best all-round service for the thick cake, and No. 12 canvas on wire netting answers the requirements for the thin-cake filtering machines. 9. With slime containing a large proportion of colloid or clayey material pressures greater than those obtainable with vacuum apparatus are of questionable advantage. 10. With slime containing a large proportion of clayey material the vacuum-filters should be used. 11. With slime containing a small proportion of clayey material and much fine sand both vacuum-filters and pressure-filters could be used with perhaps equally good results. With slime containing much coarse and fine-sand the chamber-filters with air-agitation and high pressure would perhaps give the best results. 13. Of the vacuum-filters, the thin-cake continuous filters are a decided improvement over the thick-cake filters."—GEO. J. YOUNG, *Mining and Scientific Press*, Oct. 28, 1911, p. 552. (H. A. W.)

CYANIDING CONCENTRATE AT TREADWELL.—"At the meeting of the American Institute of Mining Engineers held in October at San Francisco, W. P. Lass presented an elaborate paper on the new cyanide plant erected for the purpose of treating the pyritic concentrate recovered at the five mills of the Alaska Treadwell group of mines. These mills crush in the aggregate about 5,000 tons per day, and by means of Frue vanners about 90 tons of concentrate is obtained per day. The gold content of this product varies from 2½ to 4 oz. per ton. The average may be taken at rather over 3 oz. The gold contained in it represents 37% of the content of the original ore. Until recently this concentrate has been sold to the Tacoma smelter, and the companies were charged $11·95 for freight and treatment. In order to reduce this cost it was decided three years ago to experiment with fine-grinding and cyaniding. Accordingly a small plant was erected consisting of a tube-mill, amalgamating plates, Pachuca vat, and accessories. It was found that 75% of the gold content could be recovered by amalgamating the ore during and after reduction to −200, and that by amalgamation followed by cyaniding a total recovery of 96% could be effected. It was figured that with a plant capable of handling the whole product the cost would be $3·25 per ton, and this added to $2·48 representing the loss of 4% on a 3 oz. gold content would bring the total cost to $5·73 per ton. Thus a saving of $6·22 would be obtained by substituting local treatment for shipment. Subsequently it was found that the same extraction could be obtained by cyaniding direct without amalgamation. However, it was thought best when building the full plant to provide for both processes, but after five months run the amalgamation plates were discarded. The concentrate is sluiced direct from the bins to Dorr classifiers, which feed their coarse product into Abbé tube-mills. These mills have corrugated sectional liners, and 3 in. Danish pebbles are used. When amalgamation is employed the overflow from the classifiers and the discharge from the tube-mills is passed over two sets of plates for treating the coarse and fine pulp respectively. The pulp from the amalgamators or, if these are omitted, direct from the tube-mill circuit goes to Callow cones, the spigot discharge from which is given a short preliminary cyanide treatment in 4 Pachuca vats. The main treatment is effected in 8 Pachuca vats, and is in two stages. In the first treatment, eight hours agitation is given with 0·1% solution, and it takes 18 hours for settlement and decantation. Afterward a second treatment is given with solution drawn from the wash-solution storage tanks, the cyanide strength being 0·075%. This agitation lasts for 2 hours, and the solution is then decanted and used as solution on a new charge of pulp in another vat, cyanide being added at the same time. The pulp after the second agitation goes to Kelly filter presses, and after extraction of the solution is given a wash. The gold solution goes to Merrill precipitation presses. The cyanide solution passes from the end to the beginning of the series of operations, in the opposite direction to the flow of ore. A two months run from May 16 to July 15 of this year, after amalgamation had been discarded, gave the following results as regards cost and extraction:

Content of concentrate	$55·49
Content of tailing	$1·89
Tons treated	4,080
Extraction	96·6%
Cost per ton:	
Labour	$1·24
Power...	$0·25
Pebbles (13½ lb.)	$0·18
Cyanide (2½ lb.)	$0·53
Zinc-dust (2·09 lb.)	$0·17
Other chemicals and supplies ...	$0·35
Total	$2·72

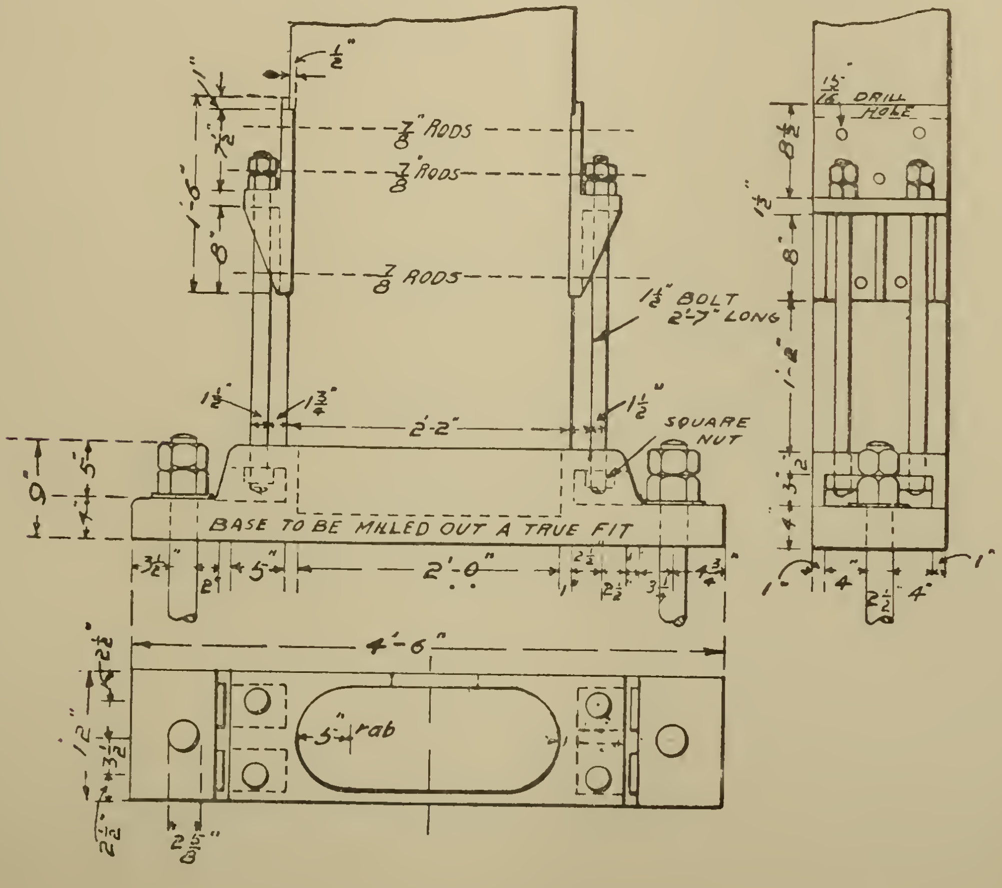
—*The Mining Magazine*, Dec. 1911, p. 467. (A. R.)

A METHOD OF SECURING BATTERY POSTS.—"The shoe and bracket for securing battery posts, shown in the accompanying illustration, and described by Francis W. Sewell, was designed for the Guanajuato Consolidated Mining and Milling Company.

The shoe is of cast iron and weighs 860 lb. It has a bearing surface on the concrete foundation 54 × 12 in. The shoe is secured to the foundation block by two 2½ in. rods, each 5 ft. long, threaded at both ends. The threaded portions are of enlarged cross-section, so that the diameter at the base of the thread is not less than that of the remainder of the rod. The lower ends of the rods pass through broad, square, cast-iron washers, embedded in concrete, and secured thereto by square nuts. Hexagonal nuts are used at the top end to hold down the shoe; lock nuts are shown in the accompanying sketch, but these are not necessary.

The shoe has a cavity cast in it, 24 × 10 × 6 in. deep, with rounded ends, into which the foot of the battery post fits. The bottom of this cavity is milled so as to give an even bearing surface to the post. At each end of this shoe are cast suitable projections, bored vertically with 1$\frac{5}{16}$ in. holes to receive the ends of 1½ in. bolts from the brackets. The two side brackets, which weigh about 100 lb. each, are recessed into the 12 in. faces of the posts, and secured in place by five ⅞ in. rods, which pass through the post from one bracket to the other. These rods are threaded at both ends. Nuts hold the brackets firmly in position. Passing vertically downward from each bracket to its respective end of the shoe are two rods, aforementioned, with square nuts to attach them to the shoe and hexagonal nuts to attach them to the bracket. The lowest part of the bracket is 14 in. above the top of the shoe, so it will be seen that once the post is set in position, with all the nuts tightly screwed down, there is little chance of any movement. With the exception of the lower nuts of the 2½ in. rods holding down the shoe, all the nuts are readily accessible and the rods connected with the side brackets are easily replaced,

In the erection of the battery the cavity is half



filled with oil and the foot of the post is cut slightly larger than the cavity, the point being cut so as just to enter. Some specially long rods are passed up through the brackets from the shoe, and fitted with nuts, which, on being screwed down, force the post into the cavity. This action is aided by blows with heavy hammers on the top of the post. The oil is forced into the end of the grain of the wood and passes upward some distance. This insures that no moisture can lodge in the cavity to rot the post. Frequent applications of oil are given to the posts, until it soaks in slowly, indicating saturation, after which the posts are painted."—*Engineering and Mining Journal*, Sept. 1911, p. 30. (A. L. E.)

Melting Metals at the Mint.—"In the yearly report of the Master of the Mint particulars are given of the new gas furnaces used for melting metals. Many experiments had been made with electric and oil furnaces when a change from coke furnaces was first contemplated, but these were not found so suitable as those using gas. In the experiments with ordinary town gas a variety of systems were tested, involving the use of gas at ordinary and high pressures, combined with variations in the pressure of the air. The burner ultimately adopted was made by S. N. Brayshaw, of Manchester. For the Brayshaw burners in use at the Mint the gas is taken at the ordinary pressure in the main (about 3 in. of water), and mixed with air from a powerful blowing-engine at about 2 lb. pressure. With care the noise of these burners can be reduced sufficiently to be unobjectionable. The furnace linings compare favourably with those hitherto used for coke in regard to endurance, and the crucibles used for gold are found to last longer, each crucible serving for an average of eighteen heats, as against twelve when coke is used. The adoption of gas as a fuel in place of coke has been accompanied by several incidental economies, and it is interesting to note that the present charge for fuel averages about 4¼d. per cwt. of standard gold bullion, as against 7d. when coke was used. It should be added that the fuel needed to warm the furnace in the morning has been eliminated; but, including this, the cost rarely exceeds 5d. per cwt. on a day's work. During 1910 the Mint installed sixteen gas-heated furnaces, each capable of taking a 400 lb. crucible, and fired with Brayshaw gas-burners. These sixteen furnaces have a maximum capacity of 2¾ tons of sterling silver, or 2¼ tons of bronze per melt, and four melts of silver, and three of bronze, can be made in an ordinary working-day of 10 hours. Five furnaces similarly gas-heated, but of a smaller size, are in use for melting gold. It will be seen that a substantial saving in cost of fuel has been effected by the use of gas. The incidental economies are still greater. The cost of carting and handling both coke and clinker is saved, and the room formerly required for the storage of them is set free. The expenditure on crucibles, which is a large item, has been reduced by 33%."—*The Mining Magazine*, Dec. 1911, p. 452. (A. R.)

MINING.

Mining on the West Coast.—"A cynic once observed of the Gold Coast that it was not 'rich enough to pay and too rich to leave alone.' On November 1st last year, in the course of his letter urging the necessity of Government assistance for recruiting, Lord Harris wrote: 'Most of the mining companies working in the Gold Coast Colony are at present obtaining gold from rich shoots of ore only; while the remaining parts cannot be worked owing to existing conditions.' By way of illustration he further stated that the costs at the Abosso to June 1910 were 33s. 3·4d., and at the Taquah 47s. 8·2d per ton, as compared with an average of 17s. 1d. on the Rand at December 1909, and that at the Abbontiakoon 281,938 tons of ore were developed, averaging 12·7 dwt. and 221,739 tons averaging 3·3 dwt. Turning to consider the statistics of the industry for last year, we find that the output was 189,444 oz., a decline on recent years, owing, it is said to development being required in several instances. The actual sum of production since the 'Jungle' boom is as follows:—

	Crude oz.		Crude oz.
1901	... 6,162	1906	.. 225,959
1902	... 26,911	1907	.. 293,218
1903	... 70,775	1908	... 297,977
1904	... 93,548	1909	... 235,512
1905	... 168,457	1910	.. 189,443

For the current year indications are that the output will be about 265,000 oz. Four mines only were producing on any scale, the output for 1909 and 1910 comparing as follows:—

	1909. Crude oz.	1910. Crude oz.
Ashanti Goldfields ...	53,390 ...	84,631
Abosso	42,305 ...	41,430
Taquah	34,668 ...	25,314
Bibiani	25,419 ...	21,278

The Taquah conglomerate mines crushed 115,794 tons for 62,350 oz., giving a return of 10·77 dwt. per short ton, while the quartz mines crushed 188,366 tons of ore with an extraction of 11·27 dwt. per short ton. Three dredging companies were at work in 1910, as in the previous year. The amount of ground treated was 2,902,863 cubic yards, from which an average recovery of 2·51 gr. was obtained, the total yield being 15,347 crude oz. against 18,514 crude oz. in the previous year."—*London Mining Journal*, Dec. 16, 1911, p. 1213. (A. R.)

Ores in Sandstone and Shale.—"In *Economic Geology* for September, Waldemar Lindgren contributes a paper on the occurrences in various parts of the world of ores of copper, lead, vanadium, and uranium in sandstone and shale far away from igneous rocks. These ores appear to have a common mode of origin, and he discusses their probable genesis. The ores are usually of low tenor, and in the majority of cases are not at present of any commercial value. The primary ores in such deposits are chalcocite with small amounts of bornite, chalcopyrite, and pyrite; galena; roscoelite (a vanadium mica); and carnotite (vanadate of uranium). Small amounts of nickel, cobalt, molybdenum, barium, and selenium compounds are often found. In some deposits all these minerals are found together, but usually they are segregrated in such a way that one metal predominates. Large gangue is found accompanying, and is usually confined to calcite, barite, and gypsum. Where the deposits have been exposed to atmospheric action, the minerals are oxidized. The deposits are notably confined to formations belonging to the Permian, Jurassic, and Trias, and characteristic occurrences are connected with fractured or brecciated beds or where fossil plants abound. More rarely the ores are found in fissures in the same formations, where they were evidently deposited by ascending or descending atmospheric water. Occasionally the ores have have been leached and carried down into the lower strata. The occurrences of copper deposits of this type are mainly on the flanks of older continental areas containing pre-

Cambrian copper deposits. The probability is that these rocks were weathered and the copper ore contained in the detritus leached by atmospheric water charged with salt and gypsum. On being carried to lower horizons the solutions would be concentrated, and the copper minerals precipitated by reactions with coaly vegetable matter or by the calcareous cement in the sandstone. Thus the deposits are distinctly epigenetic, and are not connected with ascending thermal waters. As regards the vanadium deposits, the occurrences are somewhat similar, but the association with carbonaceous matter is even more obvious. Mr. Lindgren reviews the principal deposits in various parts of the world. The Russian districts in Perm and the Southern Urals have long been well known for this class of copper ore, and more recently the deposits of the Akmolinsk district of Southern Siberia has become prominent. He also gives an outline of the known occurrences in Germany and the United States. The Katanga deposits are mentioned, but in the absence of information the author is not in a position to decide whether they come within this category. The Mansfield cupriferous shales are not included, because the majority of geologists consider the copper minerals as syngenetic with the beds themselves. Mr. Lindgren also reviews the occurrences of vanadium and uranium in various parts of the world."—*The Mining Magazine*, Dec. 1911, p. 461. (A. R.)

MINING METHODS AT TONOPAH, NEVADA.—"In general, few stulls are used in the Tonopah mines, but in the wider stopes, square sets are being employed. The practice is not to fill the sets as mining progresses, but to wait until stoping is well along. This is all right, but full advantage is not taken of the natural conditions, as more extensive use could be made of slides in getting the ore out of the square-set stopes, and little shovelling should be necessary. The posts of the square sets used at the Tonopah Mining Company's mines are too short, as there should be a clearance of at least 6 ft. 8 in. between floors in order to operate with the greatest economy. In the Tonopah-Belmont workings, peculiar sizes for the different members of the sets are used, 10 × 10 in. timbers being employed for caps, while the girts are cut from 8 × 10 in. stuff. In the sets where the tracks run, double-length caps are used at the Belmont.

In flat stopes that are under 12 ft. wide, necessitating only two stulls, most of the ore is "bull-dozed" out of the stope by main strength with shovels and in a manner that should hardly be called mining. Sheet-iron chutes are used in the flat stopes of the Tonopah company, and at first, following South African practice, these were suspended by chains hung from the roofs, the chutes being swung by hand. It was later found that the ore would run in the chutes at an angle as low as 25° without swinging, after which the chutes were laid on the floor of the stope. These chutes are made of ⅛ in. sheet iron in a semi-circular form with a diameter of 30 in., and are 8 ft. long, being reinforced by 2½ × 2½ in. strap iron, riveted near both ends. In these stopes the chutes are laid to overlap one another, and carry the ore to a small pocket at the bottom of the stope. In some of the stopes the Tonopah company uses the ore-filled method until the width exceeds 12 ft., when the square-set method is used.

At the Montana-Tonopah mining is done on waste filling. This is the most interesting stoping in the camp and caused the mine to look the most shipshape of any that I saw when at Tonopah. At this mine the ore is sorted fairly well in the stope, the rest of the waste for filling being obtained from inclined raises that are put up in the walls to prospect for nearby orebodies, as well as to furnish waste for filling the veins, which are from 3 ft. to 8 ft. wide. A series of slightly dipping step faults break up the walls, making it imperative to fill the opening as the stope progresses. These faults, taken together with another series of steeply dipping faults, so break up the vein that it is entirely out of the question to raise to the level above and obtain the excess waste required in the stope from development work carried on upon the different levels. In addition the walls must be prospected closely between levels, as well as on the level, due to the dip faults.

Where the vein is narrow the waste is blasted down first, levelled off and the ore blasted down afterward. A good method employed at the Montana-Tonopah mine is that of having the machine miners drill the holes in the roof and leaving them for the 'muckers' to blast; this is made possible at the Montana property, as the ore occurs as a number of stringers through a mass of waste. The shovellers, mainly 'single-jack' miners waiting for a place on a machine, understand the use of dynamite better than do the machine-trained miners, and therefore blast the holes more lightly, breaking the rock into larger pieces, thus enabling most of the bands of waste between the ore stringers to be rapidly sorted. Knowing that it is easier to sort coarse ore than fine, the shovellers realise that they can make a better showing with the coarse and so make better men for blasting. At the Montana-Tonopah it has been found possible with this method to mine at a profit ore that would hardly pay were it used from wall to wall. Small chutes cribbed with 2 in. planks are used. In lagging the stulls above the level for carrying the ore in these stopes, 3 in. planks are used instead of 8 in. poles, as is the practice at the ore-filled stopes of the Tonopah company. This is cheaper and the planks are cut so as to fit tightly against the wall.

Day Work at Tonopah Mining.—An interesting thing about the mining at the properties of the Tonopah Mining Company is the arranging of the work so that as much as possible can be done on day shifts. This permits most of the men to get a good night's sleep and at the same time the load on the machinery is kept well balanced, as is obligatory when operating with purchased electric power. All drilling in the stopes, as well as the 'mucking' in development workings, is done on day shift: the drilling in the development faces and the tramming and hoisting of the ore are done on night shift. The blasting is all done on night shift by a special crew of three men, each loading and blasting on an aver-

Montana-Tonopah Mining Costs.

	Per ton.
Labour:	
Ore breaking	$0·844
Timbering	0·211
Hoisting and dumping	0·254
Blacksmithing	0·038
Surveying	0·039
Foreman and shiftbosses	0·075
Sampling	0·019
Assaying	0·027
Sorting and shovelling	0·618
Tramming	0·337
Watchmen	0·014
Storekeeper	0·011
	$2·487

age about 125 holes in the 8 hour shift. This permits the drillers to run the machines up to quitting time and by the use of a trained crew of men for blasting a considerable reduction has been made in the cost of explosives. In addition, better air is secured for the miners, as the blasting is done first in the poorly ventilated places.

Supplies :	
Ore breaking	$0·392
Compressed air	0·108
Timbering	0·201
Hoisting and dumping	0·084
Hoisting—electrical power ...	0·125
	$0·910
	$3·397

Method of Drilling.—Waugh air-hammer drills are used almost universally throughout the district for stoping, although a few Shaw drills are employed in some of the mines. With the air-hammer drill a miner in a stope can put up from 12 to 18 holes, averaging 5 ft. in depth. Drifting is done with one-man piston machines and with one drill in a face. A clean face is always left for the machine men at the Tonopah company's mines, thus permitting better progress to be made, and the work of drifting is made easier by blasting on iron sheets and using square-point shovels in the drifts. Since the company has been using these sheets a shoveller can clear out a round, 10 to 15 tons of broken rock, in from 4½ to 5 hours, depending on the length of the tram ; he can then use the remainder of the shift to shovel ore in the stopes that has not been blasted into the chutes.

Montana-Tonopah Development Costs.

	Per ton.
Labour :	
Breaking	$0·650
Timbering	0·087
Hoisting and dumping	0·118
Foreman and shiftbosses	0·047
Tool sharpening, etc.	0·026
Tramming and 'mucking' ...	0·300
Surveying	0·019
Watchmen	0·008
Storekeeper	0·005
	$1·260
Supplies :	
Breaking	$0·348
Compressed air	0·084
Timbering	0·045
Hoisting and dumping	0·035
Hoisting—power...	0·064
	$0·576
	$1·836

A noteworthy feature of the mining practice at Tonopah is the large amount of sorting that is done on the surface. This is made possible by the great quantity of waste that occurs and owing to the wide difference in appearance between the waste and the ore, rapid and efficient sorting can be done.

The development costs at the Montana-Tonopah are typical of the district. The raises are two compartments, the drifts 6 × 7 ft., and the company uses a 1½-compartment shaft, characteristic of the mines that have been found by prospecting under the andesite capping. The cost of these three classes of work, exclusive of compressed air and general charges, was $6·24 per ft. for drifting, $5·83 for crosscutting, $5·10 for raising and $14·43 for sinking. Approximately one-fifth foot of drifting and crosscutting was done per ton of ore mined in the period under discussion, and 42% of this work found ore. The cost of mining was $3·40, development $1·83, making a total cost of $5·23 per ton. An accompanying table shows the itemized costs at Montana-Tonopah. The cost of mining at the mines of the Tonopah Mining Company is $3·95 per ton, while that at the Tonopah Extension is $3·11."—CLAUDE T. RICE, *Engineering and Mining Journal*, Dec. 16, 1911, p. 1184. (A. R.)

TIN IN GERMAN SOUTHWEST AFRICA.—"The *South African Mining Journal* for October 21 contains extracts from the report of E. Jorissen on the geology of a new tin district recently discovered near the line of railway between Swakopmund and Windhoek, German Southwest Africa, and 94 miles from the coast. The deposits have been traced over a tract of country measuring six square miles, in an undulating flat round the southern base of Erongo mountain. This mountain is a boss of granite intruded through schist and quartzite. The schist is a quartz-mica schist containing biotite with some muscovite, and the quartzite only differs by the large amount of quartz grains ; in some places the two merge into each other, and at other places are strongly contrasted. The strike of the beds averages N 20° E and the dip 40° to 60° S. Numerous dikes of felsite, fine-grained granite, and granite-porphyry cut through the strata, and they can be generally traced to the main granite intrusion. Nearly all the granite dikes are greisenized. There are several phases of intrusion, the principal ones being (1) the main intrusion with which are associated the numerous detached intrusions of pegmatite and many parallel greisen dikes ; (2) a granite-porphyry stage in which the later dikes cut through the previous group ; (3) greisen dikes cutting through the phorphyry ; and (4) red granite dikes not greisenized. Each stage appears to be connected with tin, but the lodes in the neighbourhood of the porphyry dikes seem to be the most promising. The best samples have come from coarse white pegmatite. The lodes are notable for the unusually large crystals of cassiterite. One mass of cassiterite is said to have weighed 500 lb. Dr. Jorissen does not commit himself to a definite opinion as to the eventual average ore, as the workings are too shallow as yet, nor have the veins been followed sufficiently to prove continuity, but in view of the interest caused by recent discoveries in Cape Colony near the Damaraland border, and the development work done by the De Beers Company not far away, his report is of importance."—*The Mining Magazine*, Dec. 1911, p. 461. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Askling, C. W., and Roesler, B. Internal Combustion Engines and Gas Producers. *C. Griffin.* Net 12s. 6d.

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Chalkley, A. P. Diesel Engines for Land and Marine Work. 8vo., pp. 238. *Constable.* Net 8s. 6d.

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Leask, A. Ritchie. Refrigerating Machinery, its Principles and Management. 5th Ed. Cr 8vo, pp. 310. *Simpkin.* Net 5s.

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Searle, Alfred B. The Natural History of Clay. (Cambridge Manuals on Science and Literature). Roy. 16mo, pp. 184. *Camb. Univ. Press.* Net 1s.

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Ostwald, Wilhelm. Outlines of General Chemistry 3rd Edition. 8vo., pp. 614. *Macmillan.* Net 17s.

Pring, J. N. Laboratory Exercises in Physical Chemistry. Cr. 8vo., pp. 178. *Sherratt and Hall.* Net 4s.

Stocks, Herbert B. Water Analysis for Sanitary and Technical Purposes. *C. Griffin.*

Text-Book of Rand Metallurgical Practice (A) Vol. I. edited by W. A. Caldecott. *C. Griffin.*

Abstracts of Patent Applications.

(C.) 594/11. William Thomas Bell (1), Arthur John Wood Graham (2). Improved construction of valve for air compressors, blowing engines and other fluid pressure engines or apparatus. 28.10.11.

This invention relates to a valve for air compressors, blowing engines, etc., so constructed that it has no wearing parts and is not suspended on pins or links.

(C.) 510/11. John Broad Roberts. A system and mixture for lubricating air compressors and purifying compressed air. 16.9.11.

This application is in respect of :—

a. A mixture of soft soap (made from fish oil and caustic potash) and water.

b. An arrangement of mixing cylinders, storage cylinder, pipes and valves for distributing the mixture to the air cylinders of compressors.

The object of the inventor is to obviate the risk of explosions in air compressors, and to give a supply of air free from the contamination produced by the use of some lubricating oils.

(C.) 537/11. Charles Dickie (1), Fred Lambert (2), Richard Stable (3). A safety apparatus for suspending mine cages and the like. 30.9.11.

This application is in respect of a proposed safety catch arrangement for preventing the fall of a skip in a vertical shaft in the event of the rope breaking.

The arrangement consists of two horizontal spindles placed on opposite sides of the shaft guides, and having at each end a toothed or suitably roughened wheel to be rotated by contact with the guides.

The spindles are carried in bearings capable of sliding towards or away from the guides and are threaded right and left hand between the bearings, on these threads are fitted nuts, having an inclined surface on the side furthest from the centre of the cage, sliding on a fixed inclined surface carried in the framing and so arranged that the revolution of the spindles, when the wheels are in contact with the guides on the downward travel of the skip, moves the nuts along the inclines and forces the centres of the spindles closer together, thus increasing the grip of the wheels on the guides and ultimately bringing the skip to a standstill.

The wheels on the end of spindles are kept out of contact with the guides in the normal working by a rope connection to the shackle which slacks down, on the rope breaking, and allows the wheels to bear on the guides and commence rotating the screwed spindles.

(C.) 89/11. Stephen William Smith (1), John Duff Innes (2). Improvements applicable to the actuating fluid supply connections for rock drilling machines, pump connections and the like. 11.2.11.

This application is for improvements to the air supply connection to rock drilling machines, pumps, winches, and the like. The connection is made with an automatic valve in the spud or coupling piece which, when the connecting hose is coupled up, opens the valve, and when the hose is uncoupled frees the valve which is automatically closed by the pressure of the air or other fluid.

(C.) 77/11. George Anderson Robertson. Improvements in driving means of automatic feeders for feeding ore and the like to crushing and grinding machines. 7.2.11.

This invention contemplates using an ordinary feeder such as the "Challenge" type which is illustrated in connection with the application, by means of projecting studs attached to some rotating part of the grinding machine. The studs engage the ordinary griping arm which operates the friction disc of the feeder.

(C.) 118/11. G. H. Rotherham (1), Vernon D. J. Hoar (2). Segmental disc filter. 23/2/11.

This application relates to apparatus for slime filtering and comprises a filtering element in the form of an inclined segmental disc with filtering surface on the underside, revolving in a tank with inclined bottom. In the elevated portion of the tank bottom an opening is provided through which the dewatered portion is discharged. A valve arrangement similar to that of the "Oliver" filter, is provided to allow the cake to be formed while the disc passes through the pulp lying in the lower portion of the tank, and discharged by back pressure when it arrives opposite the discharge opening.

The feature of this apparatus which distinguishes it from other well known filters is the inclined plane of the filter disc for the purpose of facilitating discharge of the solids.

(C.) 628/11. Sidney Williams. Process for separating precious metals from their ores. 18.11.11.

This refers to a process of extracting precious metals by means of a solution containing sulphuric acid and potassium bromide or any other acid and salt in such proportions that when mixed no reaction takes place.

(C.) 266/11. Michael Ferenczy. The nitre-oxidation process for extraction of gold from refractory ores or sands. 10.5.11.

This application relates to a process for treating refractory auriferous ores, which consists in applying a definite solution of potassium nitrate and sulphuric acid as a preparatory or oxidising wash, prior to treatment in the usual way by either amalgamation or cyanidation.

(C.) 142/11. Edmond Ernest Johnson. Improvements in extracting metals from their ores, and apparatus therefor. 1.3.11.

This application refers to improvements in the well known zinc dust precipitation process, the chief objects being to make a more perfect admixture of the dust with the gold-bearing solution and to lengthen the time of contact while preventing undue access of air. Cone-bottomed vessels, with inflow pipes reaching nearly to the bottom and a special type of zinc dust feeder, which regulates the feed in accordance with the volume of solution flowing, are employed.

(C.) 109/11. John Denyer. Improvements in furnaces for heating drills and the like. 20.2.11.

This invention relates to a type of furnace of simple construction, in which paraffin oil, petroleum or other liquid hydrocarbon is used as fuel for heating drills or bits used for machine or hand drilling.

The furnace consists essentially of two channel irons or their equivalent, one above the other, which contain bricks of a refractory material suitably grooved. One end of the bed is closed off with a firebrick and the other by a conical or similarly shaped combustion chamber with lateral projections. The whole is carried on a suitable trestle or bearer for the shanks of the drills to rest on while being heated.

The inventor claims to have provided an efficient furnace of simple construction, and further, that the use of his furnace results in a material saving in the quantity of fuel consumed and also in the wear and tear on the bricks and other parts of the furnace.

(C.) 632/11. Turbo Amalgamator and Extraction Co., Ltd. (1), Alfred Andrew Lockwood (2). Improvements in the apparatus for the extraction of the precious metals from pulps and solutions. 20.11.11.

This application refers to an apparatus for use in the extraction of precious metals from pulp or solution, and consisting in the combination of a vessel, revolving amalgamated copper plates therein, the surfaces of which pass through or are inclined at a small angle to the axis about which they revolve, a reservoir and a pump for causing the pulp or solution to circulate from the reservoir through the vessel and back to the reservoir.

Solutions of a solvent, such as potassium cyanide may be used in connection therewith, and means provided for precipitating the dissolved metal in the amalgamating vessel. For this purpose either electrolysis or chemical precipitants may be used.

(C.) 160/11. John van Nostrand Dorr. Improvements in settling apparatus. 17.3.11.

This application refers to an appliance for attachment to the driving mechanism of revolving stirrers or scrapers in settling tanks for slime or similar finely divided material suspended in water.

The object of the appliance is to give warning when the resistance encountered by the revolving blades is such as to endanger the gearing.

An arm carrying a spring is attached to the vertical spindle; and driven by a projecting lug on the driving wheel which is loose on the spindle, should the pressure increase to a dangerous extent, the spring is compressed and actuates a point, or indicating such pressure increase, a bell contact is also brought into operation, thus giving audible warning as well.

(C.) 650/11. Wilbur Alson Hendryx. Improvements in dewatering devices or apparatus for separating solids and liquids. 7.12.11.

This application relates to improvements in dewatering apparatus, and comprises a tank with vertical delivery tube at bottom, a vertical shaft with screw thereon within said delivery tube, a second shaft surrounding the first shaft—also provided with propeller blades which rotate in lower portion of tank and above the delivery tube, and means for driving the latter more slowly than the first mentioned shaft.

(C.) 658/11. James Hart Robertson. Art of extracting metals electrolytically. 11.12.11.

The application deals with the method of extracting metals which consists of simultaneously subjecting the powdered ore while in suspension in an electrolyte to the combined action of heated vapour or gas while in motion, and an electric current.

(C.) 148/11. Geo. H. Rotherham (1), Vernon D. J. Hoar (2). Segmental revolving disc filter. 8.3.11.

This specification describes a slime filter consisting of segmental filter discs, slowly rotating in a vat of pulp upon a hollow horizontal shaft, through which an intermittent vacuum or pressure is applied.

C.) 654/11. John Alston Hart. Apparatus for producing interaction between liquids and gases or vapours. 8.12.11.

This application refers to apparatus for producing interaction between liquids and gases. Relative movement between the two phases by means of a wall, formed with a large number of specially shaped ports, through which the gas may be passed, is promoted by the whirling of the liquid caused by the shape and disposal of these ports.

(C.) 115/11. Charles William Dowsett. Device for maintaining an even flow of pulp from a centrifugal pump. 22.2.11.

This application relates to the regulation of the output of centrifugal pumps and prevention of surging when not working at full capacity, and consists in the provision of throttling means in the suction in the shape of a plate loosely mounted to slide over the orifice, together with an air supply to give an equivalent amount of air, in place of the pulp withheld.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 39/12. John Berry. Anti-phthisis air-cut-out device. 18.1.12.

(P.) 40/12. James Herbert Dinwoodie. Improvements in respirators. 18.1.12.

(P.) 41/12. Lewis Christie. Improved cattle guard.

(P.) 42/12. Alfred Wright (1), Fenning Pratt Kidson (2). Hammer with wedgeless, jarless and unbreakable handle. 19.1.12.

(P.) 43/12. Rowland Owen Pickin. Improvements in dust allaying attachments for rock drills. 19.1.12.

(P.) 44/12. James Hodgson (1), Samuel Wyndham Hodgson (2), James Hodgson (3). An improved water service for mine workings. 20.1.12.

(P.) 45/12. Fred Steel (1), Robert Blane (2). Improvements in rollers for supporting the hauling ropes in mechanical haulages, and for other analogous purposes. 20.1.12.

(P.) 46/12. William Arthur Caldecott (1), Walford Robert Dowling (2). Improvements in and relating to tube mills. 20.1.12.

(P.) 47/12. William Alderson. Improved method for preventing dust particles escaping from drill holes. 20.1.12.

(P.) 48/12. William Alderson. Improved method for destroying dynamite and powder fumes when blasting. 20.1.12.

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(P.) 53/12. James Henry Reid (1), George Goodwin (2). Improvements in electrodes for electric furnaces. 24.1.12.

(P.) 54/12. James Henry Reid (1), George Goodwin (2). Improvements in electric refining crucibles. 24.1.12.

(P.) 55/12. Hugh Denehy (1), Frederick William Bergner (2). Improvements in automatic direct acting overwind trip gears for electric and other hoists. 25.1.12.

(P.) 56/12. Fridge Alexander Gillan (1), Hugh Denehy (2). Improvements in electrical overwind safety devices for electrical hoists. 25/1/12.

(C.) 57/12. James Harris. Improvements in straightway valves and the like. 26.1.12.

(P.) 58/12. James Conning. A new or improved system of ventilation. 26.1.12.

(P.) 59/12. Andre Leon Felicien Levy Strauss (1), Henry Cohn (2). Improved torch suitable for lighting fuses. 26.1.12.

(C.) 60/12. Wilheim Hinselmann. Improvements in conveyors of the jigging type. 26.1.12.

(C.) 61/12. Milton Dana Morrill. Improvements in or relating to moulds for concrete walls, floors, ceilings and the like. 26.1.12.

(C.) 62/12. Unbreakable Pulley and Mill Gearing Co., Ltd. (1), Frederick William Strondley (2). Improvements in means for the lubrication of shaft and like bearings. 26.1.12.

(C.) 64/12. Thomas Arthur Binks. Improvements relating to beds. 26.1.12.

(C.) 65/12. Arthur Loftus Bethell. Improvements in sand paper and abrasive materials. 27.1.12.

(P.) 66/12. Frolich and Jansen. Pole ladder. 29.1.12.

(P.) 67/12. Edmund Charles Pym Allardyce. Improved fittings for ensuring the removal of the entire contents of slop tanks, and other receptacles for storing liquid matter. 29.1.12.

(P.) 68/12. Valentin Janced. Improvements in apparatus for lashing or removing broken rock, ore, mineral or the like from the stopes or other underground workings of mines or the like, applicable also for the construction of dams and other similar purposes. 31.1.12.

(P.) 69/12. Joe Abrahams. Improved patent acetylene gas hand lamp. 31.1.12.

(P.) 70/12. Thomas Pearson Smith. Improvements in interlocking means for the brake and clutch gears and winding and like engines. 31.1.12.

(P.) 71/12. Gustavus Adolphus Sheeley. Improvements in shaft and track rollers for supporting hauling ropes and the like. 2.2.12.

(P.) 72/12. Gustavus Adolphus Sheeley. Improvements in elastic or resilient wheels. 2.2.12.

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(C.) 74/12. Henry James Rees (1), Henry Livingstone Sulman (2). Improvements in or relating to the deposition of metallic salts from solutions containing them. 2.2.12.

(C.) 75/12. William Bullock. Improvements in stamp mills. 2.2.12.

(C.) 76/12. John Elford Porter (1), William Wallace Thomas (2). Load-carrier. 2.2.12.

(P.) 77/12. Walter Augustus Olive (1), William Harrison (2). Safety catch for skips and the like. 2.2.12.

(P.) 78/12. Frederick Walter Bawden. Improved method of cool storage. 5.2.12.

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(P.) 98/12. Frederick Kimberley Loewenthal. A new or improved material or preparation for closing punctures in the inner or inflatable tubes of pneumatic tyres. 13.2.12.

(P.) 99/12. Leslie Walter Welsford (1), William Arthur Barton (2). An improved apparatus for moulding concrete blocks for building purposes. 13.2.12.

(P.) 101/12. Francois Pierre Fernand. An automatic apparatus or machine being an improved method for spraying sand or any other coarse or fine powdered substance into sanitary buckets in water closets. 15.2.12.

(C.) 103/12. William Philpot Curran Bain. Improvements connected with wire fencing. 16.2 12.

(C.) 104/12. Frank Robert McBerty. Improvements in telephone exchange systems. 16.2.12.

(P.) 106/12. Alfred Louis Thorne. An attachment for rock drills to absorb the dust created by drilling operations. 17.2.12.

(P.) 107/12. William James Barnett (1), Oswald Victor Whittle (2), Fred Morgan Linscott (3). Improvements in means for igniting blasting fuse. 17.2.12.

(P.) 108/12. James Anderson. Automatic gravitating rolling roller crusher. 20.2.12.

(P.) 109/12. Louis Clague Partington. Improvements in or relating to automatic signalling on electric tramways. 20.2.12.

(P.) 110/12. Harry Ernest Waters (1), John Russel Cawthorne (2). Improvements in ore feeders. 20.2.12.

(P.) 111/12. Fabrian Mercer Cox. An improved caisson and air lock for the recovery of precious stones and minerals from deep waters. 21.2.12.

(P.) 112/12. Gustavus Adolphus Sheeley. Improvements in elastic or resilient wheels. 22.2.12.

(P.) 113/12. John Hart. Improvements in acetylene lamps. 22.2.12.

(P.) 114/12. Thomas Dawson. Improved process for extracting the moisture from green wood and for speedily and efficiently drying and compressing same for manufacturing purposes. 23.2.12.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Arkell, D. J., *l/o* Johannesburg; Fortuna Mine, P. O. Box 124, Barberton.

Barrett, F. C., *l/o* Bulawayo; Bushtick Post Office, Rhodesia.

Bawden, E. R., *l/o* Nyasaland; Poldice House, St. Day, Scorrier, Cornwall, England.

Brading, T., *to* P. O. Box 332, Bulawayo.

Brodigan, V. F., *l/o* East Rand; P. O. Box 45, Sabie, Transvaal.

Cardwell, R. H., *l/o* Cornwall; Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.

Clarence, J. V. G., *l/o* Randfontein; Meyer and Charlton G. M. Co., P. O. Box 27, Jeppestown.

Duncum, E. C., *l/o* Germiston; Bantjes Consolidated G. M. Co., P. O. Box 2, Florida.

Gray, J. W., *l/o* Dublin; c/o Pahang Consolidated Co., Ltd., Sungei Lembing, Kuantan, Federated Malay States.

Hamilton, E. M., *to* Dolores Mines Co. of America, 111, Broadway, New York, U.S.A.

Hartog, V., *l/o* Kimberley; P. O. Box 5512, Johannesburg.

Hunter, C., *l/o* Pilgrims Rest; Piggs Peak Development Co., Swaziland.

Mackesy, W., *to* P. O. Box 5428, Johannesburg.

McCormick, A. R., *l/o* Pilgrims Rest; Elandsdrift, P. O. Ross Hill.

Mills, L. D., *l/o* U.S.A., P. O. Box 5218, Johannesburg.

Parsea, A. E., *l/o* Rhodesia; Nourse Mines, Ltd., P. O. Box 32, Denver.

Pryce, L., *l/o* Germiston; City Deep, Ltd., P. O. Box 1411, Johannesburg.

Scatterty, P., *l/o* Roodepoort; Lancaster West G. M. Co., P. O. Box 360, Krugersdorp.

Simpson, J. R., *to* c/o Mrs. Cook, Main Road, Roodepoort.

Simpson, F. L., *l/o* Australia; P. O. Box 391, Bulawayo, Rhodesia.

Sims, E. G., *l/o* Barberton; c/o D. McWilliams, Esq., Gatooma, Rhodesia.

Tennant, A., *l/o* Randfontein; Lancaster West G. M. Co., P. O. Box 347, Krugerdorp.

Wilson, L. N., *l/o* Nigel; c/o H. Hay, Esq., West Coast Amalgamated Mines, Adijah Bippo, Tarquah, via Sekondi, Gold Coast Colony.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

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Vol. XII. MARCH, 1912. No. 9.

Proceedings
AT
Ordinary General Meeting, March 16, 1912.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, March 16th, Mr. W. R. Dowling (Vice-President) in the chair. There were also present :—

53 Members : Messrs. H. A. White, M. H. Coombe, J. Gray, J. Littlejohn, A. Richardson, Prof. G. H. Stanley, F. W. Watson, A. F. Crosse, E. H. Johnson (Members of Council), R. A. Barry, W. Beaver, A. J. Bowness, J. Chilton, F. W. Cindel, T. H. Clesham, W. M. Coulter, C. Crichton, J. Daniel, G. A. Darling, T. Donaldson, A. L. Edwards, L. L. Ellis, R. C. Evans, B. C. Gullachsen, B. J. Hastings, A. M. Kratz, G. A. Lawson, H. J. Lea, L. Marks, T. G. Martyn, G. Melvill, H. H. Morrell, D. Nicholas, P. N. Nissen, A. C. L. Olsen, E. A. Osterloh, F. S. Pearce, F. D. Phillips, J. F. Pyles, M. H. Reid, G. A. Robertson, R. W. Robinson, A. Salkinson, A. M. Sim, A. L. Spoor, S. H. Steels, W. A. C. Tayler, J. A. Taylor, A. Thorne, C. Toombs, J. T. Triggs, and J. Watson.

17 Associates and Students : Messrs. E. G. Baskett, H. G. Brickhill, I. Dicks, J. Gibson, Alfred James, L. T. Leyson, G. J. Mackay, G. F. Mathews, F. J. Pooler, P. Scatterty, H. Stadler, E. H. Tamplin, A. M. Thomas, T. L. Thorne, C. D. Twynam, J. L. White and A. Willcox.

16 Visitors, and Fred. Rowland, Secretary.

The Chairman : I regret to inform you that owing to the Exhibition the Secretary was unable to get the February *Journal* out in time for this meeting. We were also late with the January *Journal*. We rather deserve a vote of censure ; but we hope to do better in future. It is necessary for you to confirm the minutes of the January meeting, as published in the January *Journal*, and I would ask you to do so.—Agreed.

NEW MEMBERS.

Messrs. J. Littlejohn and A. F. Crosse were elected scrutineers, and after their scrutiny of the ballot papers, the Vice-President announced that all the candidates for membership had been duly elected as follows :—

Gardiner, Beauchamp Lennox, London and Rhodesian M. and L. Co., Ltd., P. O. Box 80, Salisbury, Rhodesia. Metallurgist.

Mills, Louis David, P. O. Box 5218, Johannesburg. Metallurgist. *(Transfer from Associate Roll.)*

Preez, Douw du, Tweefontein Colliery, P. O. Minaar. Surveyor.

The Secretary notified that since the last meeting of the Society the following Associates and Student had been admitted by the Council :

As Associates—

Collins, Edgar Archibald, c/o Mr. C. E. Knox, 300, First National Bank Building, San Francisco, Cal., U.S.A. Mining Engineer.

Hall, Thomas Dennison. Consolidated Goldfields Laboratory, P. O. Box 108, Germiston. Laboratory Assistant.

Law, Carlos W. van, Pachuca, Hidalgo, Mexico. Metallurgist.

Leonard, Dillon Tredway, P. O. Box 89, Gatooma. Amalgamator.

Millar, Frederick, Nourse Mines, Ltd., P. O. Box 32, Denver. Reduction Works Learner.

As Student.—

Estill, Frederec (S. A. School of Mines and Technology), 27, Gordon Road, Bertrams, Johannesburg. Laboratory Assistant.

GENERAL BUSINESS.

The Chairman : Under this head will come the visit to the Coalbrook Collieries. I beg to propose a hearty vote of thanks to the directors and management of the Clydesdale Collieries, Ltd., for the most interesting information which the members received in their visit last Saturday and for the entertainment provided.

The motion was carried unanimously.

ACCIDENTS IN STAMP MILLS AND CYANIDE WORKS.

Mr. G. A. Robertson (*Member*) : A recent accident which occurred in the stamp mill of the

Aurora West mine whereby an employee, Robert Scrimgeour, lost his life, is my excuse for bringing to the notice of the members of this Society the necessity for rules and regulations to govern the operations in stamp mills. The handling of stamps is the most dangerous occupation on the surface of a gold mine. The accident referred to as reported in the daily papers occurred through the breaking of a chain resulting in a stamp falling on the deceased. Now, I think that most experienced millmen have seen a chain break while working with stamps. Chains and tackle in use there are subject to considerable wear and tear owing to the awkward work they have to do. For instance, the general method of hauling a stamp out of a mortar is to coil a chain three times round the stem and pass one end of the chain through a ring on the other end of the chain. The stamp is then raised from this chain which grips the stem. The links of a chain used in this way are liable to get twisted, and when additional strain is applied to the chain by raising a stamp (stem and head fixed) past the lower guides the stamp is liable to get seized and this not to be detected until undue strain has been applied to the chain. Another way in which chains are liable to snap is in attempting to force a broken stem up through a neat-fitting guide. Those not acquainted with the details of mill practice would take it for granted that if a stem breaks just above the head and it is immediately hung up that it will easily pull through the guide without being slackened. This is not always the case because if the broken end of the stem has been welded it may be rough, and further it may be rusty and not able to pass through the guide without being forced, which and perhaps results in the chain breaking, which I have seen occur. The right way is of course not to force a stem through a guide, but millmen are like others, they are not infallible, and as trouble does not come alone in stamp mills there are times when the millman finds it advisable to get a move on, and he may not be so cautious in these details as he might be.

Chains and tackle in a stamp mill are as a rule under the care of the mill fitter, but their inspection is liable to be forgotten unless a rule exists for this to be done periodically and recorded in a book kept for the purpose.

I think a committee ought to be formed to enquire into the subject, but in order that all may have a voice in the matter, if permitted, I am prepared to bring forward a short paper on the prevention of accidents in stamp mills.

A recent explosion which was reported as having occurred at the Crown Mines cyanide works, resulting in the death of an employee, also goes to show that rules should be posted in the cyanide works instructing employees in the precautions necessary in the execution of certain work.

Mr. Andrew F. Crosse (*Past-President*): I think it is usual at the March meeting to announce when the Annual Dinner is to be held. Perhaps the Chairman will be good enough to inform us.

The Chairman: In connection with our Annual Dinner, the usual committee was elected at the last meeting of Council, and they have the matter of the dinner in hand. Our previous practice has been to rely entirely upon that committee, because they have made such successes of our previous dinners. The committee have not yet fixed the date, because there are many considerations which weigh with them, such as other functions taking place at the same time and the availability of speakers.

WELCOME TO MR. ALFRED JAMES.

Before proceeding further, I would like, on your behalf to welcome a distinguished visitor in the person of Mr. Alfred James, Past-President of our sister society, the Institution of Mining and Metallurgy, and a very old pioneer of the Rand and of Barberton. Mr. James is one of a well known mining family, a Past-President of the Institution of Mining and Metallurgy, and member of many technical societies and needs no introduction from me.

Mr. James arrived here in 1888, and was at once identified with the mining life of the Witwatersrand at the Langlaagte Estate Mine. Subsequently he erected the first cyanide demonstration plant here early in March and April, 1890, quite a long while ago. In that connection I have recently seen an interesting photograph of the original cyanide demonstration plant on the Salisbury Mine, which comparatively few of us, I am afraid, remember. In this photograph we find Mr. John Stewart MacArthur—inventor of the cyanide process; Mr. Alfred James, our visitor, and Dr. Caldecott, one of our Past Presidents. An interesting feature of the plant was that it was an agitation plant for the treatment of sand. However, it seems to me that to that plant and to the enterprise of these gentlemen we owe the position we stand in to-day where we are able to make a profit for the companies and enabled to work on so large a scale on low grade banket ore. Mr. James is one of the most rapid travellers and acquirers of metallurgical knowledge that I know of, and is the centre of metallurgical knowledge in London, as is shown by his Christmas letters to those who are privileged to receive them. I wish, on your behalf, to welcome him to our meeting this evening.

Mr. Alfred James (*Associate*): I need scarcely assure you of the pleasure it is to me to be here this evening. I have to thank the Chairman for the extremely kind way in which he welcomed me; as I came here this evening, my mind instinctively travelled back to the good old days. I do not know why we should call them the good old days. I suppose others will call these days "good old days," but my mind goes back to the good old days when everybody came to these meetings and spoke, and spoke in vigorous terms, and enjoyed himself immensely. I remember quite an eventful evening when Mr. Crosse wanted me to speak on dry crushing, and the conversation took quite another turn; we had an extremely lively evening, which I am not likely to forget.

Whilst I am here I should like to express what you members must already know, and that is the obligation of the industry to the results of the research and painstaking investigation carried out by members of the Society. All over the world your transactions are regarded with much interest. You will find one of those associates whom you have elected to-night is a man of no mean metallurgical attainment in Mexico; but it so happened that in Mexico he had to get a little information and found that he had to get it from you, and that has resulted in your being asked to elect him a member, so that he may be assured of receiving your transactions first-hand instead of having to ask somebody else for the good things they contain.

You have identified yourselves in the last few years particularly, as it seems to me, with the subject of crushing, amalgamation and tube milling. Personally, I should have also welcomed it had you also gone into the other end and devoted more attention to the products in the material going to waste; but, having regard to the information which you have been so good as to place at the disposal of the world on the subject of how to increase the duty of the stamp and the efficiency of grinding apparatus, I confess that I should personally welcome the efforts of the members of the Society if they could induce, say, the Mines Trials Committee to undertake more of that research which has been so useful both to the industry here and elsewhere as to certain definite data connected with tube milling. For instance, if you are going to crush through a 1 in. mesh, is that an economical size of particle to feed into a tube mill, or is it not? Again, if you are going to crush to a 1 in. mesh is that more economically done by stamps or by rolls? Again, if you are going to feed a 1 in. mesh product into a tube mill, is it reasonable that the tube mill should be of the same diameter as the tube mill into which you feed a $\frac{1}{64}$ in. particle, or should the diameter be larger so that the fall of the pebbles may be greater? Again, should the length of the mill be the same in which you only break down particles and do not fine slime, and so on? There are so many points on which one is working empirically. One is working in the dark. A man alters his screens to find out how much greater output he gets per unit of horse power regardless of the limit or size which may later be found an economical feed for a tube mill. Data worked out by the Mines Trials Committee would be most useful. Take Mr. H. S. Ball's paper on tube milling, read before the Institution of Mining and Metallurgy. You will find he advances some most interesting information. If, instead of a student, some Past-President or Member of the Council had set himself to work with a full sized tube mill, I think the information obtained would not only be of the utmost value to the industry here, but also to us in other parts of the world who are busy watching your proceedings.

I have to thank you for the very kind way in which you have received me, for so long past a stranger.

The Chairman: Mr. James has given us some very useful hints for papers by the members, and I hope these hints will be adopted at an early date.

THE ASSAY OF GOLD-BEARING CYANIDE SOLUTIONS BY ELECTROLYSIS.

(Read at September Meeting, 1911.)

By C. Crichton, F.C.S.

REPLY TO DISCUSSION.

Mr. C. Crichton (*Member*): I much appreciate Dr. W. J. Sharwood's contribution to the subject matter of my paper, read at the September meeting, and would like to express my thanks to him for his information in regard to Mr. A. M. Henderson, of Victoria, whose original publication caused Mr. Erskine to experiment on, and finally adopt the electrolytic method for the assaying of solutions at the Kleinfontein Group assay office.

Dr. Sharwood's experience when using Mr. Henderson's method coincides with that of Mr. Erskine who after the completion of the first few tests decided to substitute carbon for iron anodes. Undoubtedly also a great improvement was effected by the use of larger cathodes, entailing as it did a greater distance between electrodes, and producing more solution resistance to the current, thus ensuring to a great extent that the deposited gold would be of firm and adhesive character.

Dr. Sharwood does not state what lamp resistance he used with the 110 volt D.C. lighting current, but I should imagine, from his results, that the current he employed was of a fairly high pressure, and this, taken into consideration together with the fact that his solutions contained a good percentage of thiocyanate and metallic salts, also the fairly close proximity of anode to cathode, evidently produced a too rapid deposition of metals, hence the loss of detached portions of deposited metals whilst folding the cathode. Certainly the whole character of the solutions with which he made his trials vastly differs from the ordinary cyanide works solution we are acquainted with on the Rand, and the comparative absence of dissolved metals, thiocyanate, etc., has facilitated our accomplishment of a reliable solution valuation by means of electrolysis. I note that in his first experiments Dr. Sharwood used anodes of lead, with disappointing results. Personally, after a series of trials using various metals as anodes, I have arrived at the conclusion that the only satisfactory anode is one of carbon. Very few metals, if any, remain unaffected either by the cyanide, or its accompanying protective alkali, during electrolysis. Both iron, lead, and tin speedily become coated with a slimy precipitate which, if it is not removed, greatly increases the current resistance.

Dr. Sharwood has made a very interesting suggestion in connection with the use of anodes of Acheson graphite in preference to anodes of carbon, his claim being that anodes of graphite are insoluble, non-absorbent, and do not precipitate gold ; it is well known however that anodes which are insoluble in cyanide solution are troublesome for the reason that a film of gas is formed on the surface of the electrode thereby increasing the resistance to the flow of current from the electrodes to the solution, and necessitating some means being employed whereby the E.M.F. of the current is automatically increased as the resistance increases, so as to keep the current density through the solution constant. It would appear that in the tests carried out by Dr. Sharwood the solution resistance increased during electrolysis, as in the case of tests Nos. I. and II., where he states that the current fell from 0·45 to 0·33 ampere, and assigns as the probable cause of this fall in current density, the precipitation of solids out of solution consequent on the decomposition of thiocyanate. I incline to the opinion that the loss of salts out of solution would to a certain extent increase the solution resistance, but would suggest that the increase of resistance in the specified tests was partly a *contact* resistance, due to causes as stated above. The fact of Acheson graphite being non-precipitant to gold in KCN solution is certainly an important feature, and more particularly so where the current supply is drawn direct from the lighting circuit mains and its continuity therefore beyond the control of the assayer. At Kleinfontein, however, the danger of incorrect results arising from stoppages of current has been practically obviated, as under existing arrangements the current supply is obtained from secondary batteries charged from the lighting main, through a lamp resistance, and can be entirely disconnected from the lighting circuit by means of two-way switches, so that we have an assured complete control over the current used.

In reply to Dr. J. Moir, I should like to state that the voltage we employ does not exceed seven volts, and that though for quite a long time only four volts were used, we found that the current fell off owing to contact resistance, and to overcome this difficulty we increased the E.M.F. by making an addition of another two volt accumulator cell to our storage. As regards Dr. Moir's contention that the hydrogen bubbles given off at the cathode tend to dislodge particles of gold, I need only say that this has not been our experience ; indeed the gas bubbles serve a very useful purpose in keeping the solution more or less in agitation, and have the further advantage of enabling one to see at a glance whether the apparatus connections are good and if the current is circulating in each solution. Dr. Moir's other objection to the voltage we employ is not altogether reasonable, and as I have already stated our solutions are fairly free from dissolved metallic salts, so that no particular object would be served by lowering the E.M.F. to prevent the deposition of base metals, the only result from the pursuance of such a policy would be a much greater delay in completion of the electrolysis.

Prof. G. H. Stanley suggests rotating anodes or cathodes as a means of hastening the complete deposition of gold in solution, and no doubt a series of five or six revolving anodes would be useful for experimenting purposes. We considered this question of agitation some time ago but came to the conclusion that the introduction of the mechanism necessary to accomplish agitation would involve more or less personal attention, which with our present apparatus is not required, and decided to keep the apparatus as simple as possible.

The Chairman : I would like to propose a vote of thanks to Mr. Crichton for the additional information which he has given.

NOTES ON HIGH DUTY GRAVITY STAMP MILLS.

(*Read at October Meeting.* 1911.)

By Peter N. Nissen (Member).

REPLY TO DISCUSSION.

Mr. P. N. Nissen (*Member*): The author wishes to thank those members who have contributed to the discussion of this paper, believing that the ideas expressed by way of criticism of the several advantages of the Nissen stamps over the multiple stamp, as brought out in the paper, were advanced in a fair spirit. The expressions of commendation on the success of the stamp and the general interest of the paper are hereby gratefully acknowledged.

Upon reviewing the discussion, the author can but note that the points criticised are few, and while not feeling that the excellence of the Nissen stamp is proved beyond all doubt by such lack of criticism, yet it is gratifying that the paper did more or less cover all the points worthy of consideration. So far as the claims made for the Nissen stamps are concerned, as based on the tests made at the City Deep, and in consideration of the criticisms made by the various members, the author expects to show in this reply that not only were all the claims fully proven by the tests, but that it will also be possible to bring out further points of merit for the Nissen stamp not previously referred to. The author was very guarded in his statements in the original paper, and in many cases failed to bring out all the advantages of Nissen batteries over multiple stamp batteries. Now that the discussion is closed so far as this Society is concerned, it is necessary to advance these points, and to do this the various criticisms will be dealt with seriatim and the conclusions finally summarised.

Feed.—In the paper the author made an important point of method of control of the ore feed to the stamps. Mr. Laschinger, in his first contribution fully allows this, and went so far as to say that the proper feeding of stamp batteries was probably the most important factor governing their efficiency. Although this is undoubtedly true, it has certainly not been achieved in the ordinary design of multiple stamp mortar, where a number of stamps are fed with one feeder, which is controlled by but one of these stamps, In passing, the author would say that attempts have been made to feed multiple stamp mortars with more than one feeder. In one case as many as five were tried, but the results were disappointing. The advocates of the multiple stamp mortar system have not been able so far to determine which stamp should control the feed to give the best conditions of feed to all the stamps. It is obvious that under these conditions ideal control can never be secured with a multiple stamp mortar. On the other hand the Nissen stamp operates in an individual circular mortar and each stamp is fed by its own feeder.

While this simple and obvious explanation was not deliberately given in the paper, it was undoubtedly recognised by the majority of the readers. Mr. Laschinger, however, in his second contribution returns to the point with some criticisms very wide of the mark. The author can only point out again that the competitive test runs were conducted by the Central Mining and Investment Corporation, Limited, and while every precaution was taken by the author to obtain the highest efficiency possible from the Nissen stamps it is unreasonable to suppose that like efforts were not also made with the City Deep battery. Both mills were run by the same man, and the work done in both cases speaks for his skill as a battery-man.

In finishing the discussion of control of feed, Mr. Laschinger is most unhappy in the choice of his words:—"That the author has probably accepted the conditions of maximum duty as tacitly understood, but probably also some of the members are not so trustful as the author may have assumed." There is nothing in the paper, nor in any of the author's previous writings on this subject that would indicate or lead one to suppose that he would be so easily satisfied, or that the Central Mining and Investment Corporation Limited, would in any way have the City Deep stamps operated to a disadvantage in the tests. Certainly when the cost of these tests was borne by this Corporation and the number of stamps they now have in operation represents the investment of so much capital, it is not likely they had any desire to be misled. This is especially so when they have shown such a progressive spirit on former occasions in testing other alleged improvements in stamp battery designs and constructions.

The author is quite unable to follow Mr. Mackesy in his explanation of the way the feeding of the outer stamps occurs. His objections to a low feed is quite contrary to the author's view, as will be borne out by experienced operators. It is the author's opinion that the fact of being able to run the Nissen stamp on a low feed is largely responsible for the high efficiency in this type of stamp mill, as well as for the even wear of shoes and dies.

Screen Area.—The screen arrangement of the Nissen stamp being one of the radical departures from usual practice, it is but natural that this feature should be commented upon. Mr. Laschinger's opinion that the screen for an ordinary rectangular single discharge mortar is of ample area for coarse crushing, leaves unexplained whether it is sufficient when fine crushing is practiced. He also avoids comparing the relative position of the two types of screens, which is the important point. Mr. Jane touched on this point also with the same results. Now it is universally admitted that the Nissen stamps during the competitive tests did more work, not only than the City Deep stamps, but more than any other stamp of multiple type has ever done under like conditions. Seeing that these results were obtained with a circular screen, it is safe to assume that this type of screen must have assisted the discharge, or at least not have been detrimental. If the screen area and its arrangement were not an advantage, then the improved results must have been due to other conditions, yet the critics do not allow this either. The author's conviction, based on experience, is that the circular screen contributes very greatly towards the increased duty obtained from Nissen stamps, and the following reasons will serve to bear this out:—

Figs. 11 and 12 roughly show the two types of screen in relation to the die.

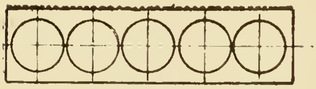

Fig. XI

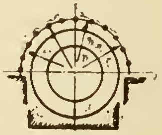

Fig. XII

The important feature of discharging the crushed ore through the screen is not so much the area as the manner in which the screen is presented to the wash of the pulp. A comparison of Figs. 11 and 12 with Fig. 13 clearly illustrates this. The ideal condition is undoubtedly to discharge each particle immediately it is reduced to the desired size, therefore to meet these conditions as nearly as possible, a screen of which all its apertures are equally effective must be used. A screen such as is permissible with a multiple stamp mortar cannot meet this demand, since by the radial deflection of the pulp, a great number of particles must strike this type of screen at such an angle that the aperture is largely reduced if not practically closed, whereas with the screen arrangement of the Nissen stamp mortar, all apertures are presented at a right angle to the line of pulp flow. Without a free discharge more work must be necessary to reduce the particles, or they must become rearranged in the mortar before they can be discharged, thus a loss in efficiency occurs. The heavier stamps, as used in Nissen batteries, when properly fed seem to reduce the ore at one blow to a greater extent than stamps of lighter weight. This fact has also been demonstrated in other tests than the ones under review. Theoretically a screen over the whole circle would give the best results, but practical considerations limit the available area to about 180°. With the heavy steam stamps having 4-sided mortars, as used in U. S. A, it is necessary to reserve one of these sides for feeding purposes.

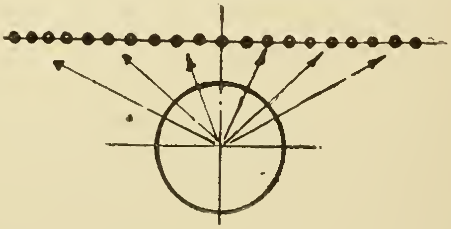

Fig. XIII

The screen area per ton of ore crushed is considerably larger for Nissen stamps than is provided in multiple stamp mortars. That there can be no disadvantage in this is obvious from the foregoing, and the theory expressed by Mr. Jane that this means greater cost of maintenance has no foundation in fact, as the wear of screens is necessarily dependent upon the amount of ore passed through the openings and, other conditions being equal, the total wear should be the same for both types of screens irrespective of size; in fact the flat screens should wear more for the same tonnage passed owing to the small percentage of effective area and the consequent greater abrasion.

Shoes and Dies.—The author was very guarded in the paper in his remarks on the proper size of shoes and dies for different conditions of ore and screen, for so far as he is aware no tests have been made to settle these points. The size that might be found proper for ordinary stamps may not necessarily be suitable for the Nissen stamp, owing to the different action of the pulp in the mortar. The size of these parts was determined by the diameter of the mortar, so that as much of the interior as possible should be available for crushing with only a small allowance made for getting out the dies. With the diameter of die used together with the return wash of the pulp towards the centre of the mortar each time the stamp is lifted, practically all the uncrushed ore comes under the falling shoe of the stamp at each blow. In multiple stamp mortars, no matter how large the die, such a condition is impossible. Under the circumstances, it would have been interesting had Mr. Torrente given the reason why the stamps he refers to were provided with 9¼″ diameter dies. The author sees no convincing evidence in Mr. Mackesy's statement favouring enlarging the diameter of shoes and dies, for he believes this question must be settled by experiment, with each type of battery. In any case, the results of the tests show that the size adopted for the Nissen stamps can not have been a disadvantage. Mr. Roberts also endorses the author's policy in enlarging the die. However, his statement without detailed data, will scarcely satisfy critics; further, as he has said, what happens in a multiple stamp mortar need not necessarily indicate similar results in the Nissen mortar.

Regarding Mr. Jane's statement that the figures given for wear of shoes and dies are too high, the author can but point out that they correspond exactly with the average of the two figures given by him. The author is loath to make too much of this point, because the experimental run should not be compared with the results from a large battery over a long period of operation, nor were any special precautions taken to assure the very best material in the shoes and dies used in the test.

Duty with Fine Screening.—So far as the capacity of heavy stamps on fine screens is concerned, the author ventures to state that reliable figures can be obtained from many of the mines on the Rand, that have accumulated such data. Nissen stamps have shown duties as follows:—

Weight of Stamp lb.	Size of Screen mesh	Capacity per Day tons.	Discharge Gradings per cent.				
1,100	1,200	9·0	+50	+80	+100	+200	−200
1,500	900	10·7	15·0	16·0	10·5	11·5	47·0
1,700	900	11·0	12·5	14·5	9·5	18·0	45·5
			+60	+90	−90	−200	
1,980	1,200	10·7	18·6	21·47	16·08	43·85	

NOTE.—Screen openings given in popular terms.

Centre of Gravity.—Mr. Kratz was good enough to prepare sketches showing the relative centres of gravity of various stamps, including the author's latest design. Until he saw these sketches the author did not fully realise how great an improvement he had made. His thanks are therefore due to Mr. Kratz for the assistance rendered. However, assuming that a low centre of gravity is an advantage, as is claimed by several writers, it is scarcely correct to treat as immaterial the difference between 4 ft. 4 in. and 6 ft. 6 in. or 7 ft. 9½ in.

Life of Cam Shafts.—Mr. Laschinger in doubting the author's statement that the life of cam shafts will be no longer with the Nissen stamps than with ordinary multiple stamp batteries, evidently discredits what is now generally accepted by all engineers, namely: that the life of a piece of metal subjected to repeated and reversed stresses is directly proportionate to the number of applications of such stresses, always provided that the material is of the same quality. Certain experiments made at the National Physical Laboratory, Teddington, as set forth in a paper recently discussed in London, bear out this statement.

Cam shaft difficulties are influenced by the nature of the material used for these shafts and by the skill used in erecting and subsequent maintenance of the battery framing. Given satisfactory conditions in these respects, the life of a cam shaft is determined by fatigue only, which means a shorter life for the 10 stamp battery shaft than for the 4-stamp shaft of the Nissen mill.

Running Time.—Mr. Jane does not state that the exceedingly fine running time he mentions for the Simmer Deep mill was obtained without the use of some of extra stamps available in the mill which a 98% running time would indicate. The average running time for the whole Rand is nothing like the percentage given, notwithstanding the fact that numbers of stamps are now permanently hung up in many of the mills due to using coarser screening; so that spare batteries are available for commission should any repairs be necessary on these regularly working. The author's assumed better running time for Nissen stamps is based on the greatly reduced risk of breakage in this type of battery and the smaller loss of capacity caused by hanging up one stamp of 30 tons daily capacity instead of 5 stamps of 70 tons capacity, so that the indicated better running time should be greater than has been claimed.

Vibration of Mortars.—Since Mr. Torrente's remarks on vibration of mortars, the author has visited one of the mills with which this gentleman is connected and satisfied himself that Mr. Torrente was correct; but he is still of the opinion that Mr Torrente will agree with him that the conditions he saw there are unusual, and not equalled throughout the Rand.

Ore Storage.—In Mr. Jane's comments on sufficiency of ore storage, he no doubt overlooks the fact that no new mills on the Rand will be permitted to run on Sundays. Moreover, the comparison made was for the same size of bin for both types of battery.

Tanks as Ore Bins.—Although the author made no important point of using circular tanks for ore bins, this is a construction now in much favour in U.S.A., especially for storage purposes, and on copper ores. No excessive corrosion takes place due to the causes mentioned, and the author would refer Mr. Jane to the steelwork removed from the Simmer Deep mill bin after two years subjection to the action of corrosive water, where the author sees nothing but paint rubbed off.

Length of Mill.—The author cannot understand why Mr. Kratz should have gone to so much trouble to show that a battery of 2,000 lb. Nissen stamps would be longer than a similar capacity battery of ordinary stamps of 1,650 lbs. The comparisons made in the paper were quite on a different basis, since only 1,400 lb. stamps were considered in the estimates, for reasons clearly set out. In the light of recent results, the author is more convinced than ever that addition of weight over 1,400 lb. to stamps of the multiple stamp mortar type must be made at the expense of greater maintenance ; this increase being out of proportion to the gain in crushing capacity. Mr. Kratz's argument is based on a design where Nissen stamps are installed in separate 4-stamp batteries, whereas the author's estimates were based on a design as shown in Fig. 14. It will be seen that this design allows for necessary ore bin capacity and shows two 4-stamp batteries connected, using two cam shafts which have their third bearings on a common post ; thus the resulting length of the mill for 48 Nissen stamps is 197 ft. as against 193 ft. for 100 stamp battery of 1,400 lb. as shown by Fig. 15. A comparison with the City Deep battery is hardly fair, for it is not likely that stamps of such great weight will be again installed, an indication of this is before us in the Modder "B" Plant, which was built after the City Deep battery had been started ; the stamps in Modder "B" being but 1,650 lb. new weight and about 1,550 lb. average running weight. Mr. Laschinger's remarks based upon Mr. Kratz's estimates are thereby rendered valueless and his deductions are therefore wrong. The figures arrived at by Mr. Kratz concerning increased power required for pumping are also incorrect. In this connection, the author feels he may point out that Mr. Kratz has been unfair in his power for pumping calculations, in that he took a ratio of water of 7 to 1 instead of 2·5 to 1 or 4 to 1, as was used in the test.

Cost of Mill Bin.—Mr. Kratz's objection to the author's claim of reduced cost of ore bin is rather inconsistent. The height of the bin for a given tonnage must be regulated by the length and breadth, and as he has shown that the Nissen battery is longer than necessary for a battery of ordinary stamps, it is evident that for the same capacity the bin can be much lower, built up of much lighter members, consequently it will be cheaper, however, as the Nissen bins need not be longer than the bins for the ordinary stamps, yet owing to more compact construction and the type of feeder and mortar used, the point of dis-

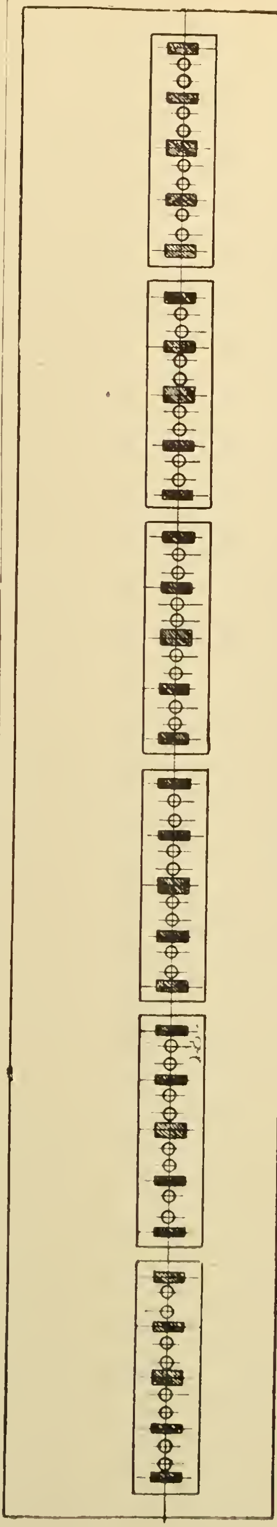

Fig XIV

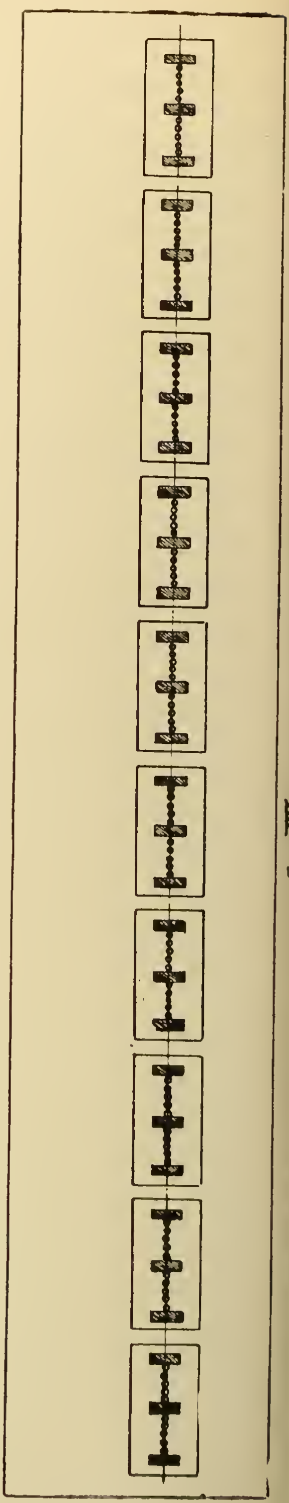

Fig. XV

charge is considerably lower, so that the total height is less, therefore it would still be cheaper.

Height of Building.—As the bins are lower and the stems so much shorter, the mill structure must also be cheaper, notwithstanding the statement made by Messrs. Kratz and Jane. By comparing the relative designs and taking measurements from the top of the battery foundations to the roof, this statement is easily verified. The very short stems and the manner of removing them sufficiently indicates this fact, less cost of building must consequently be evident. Fig. 16 also illustrates this.

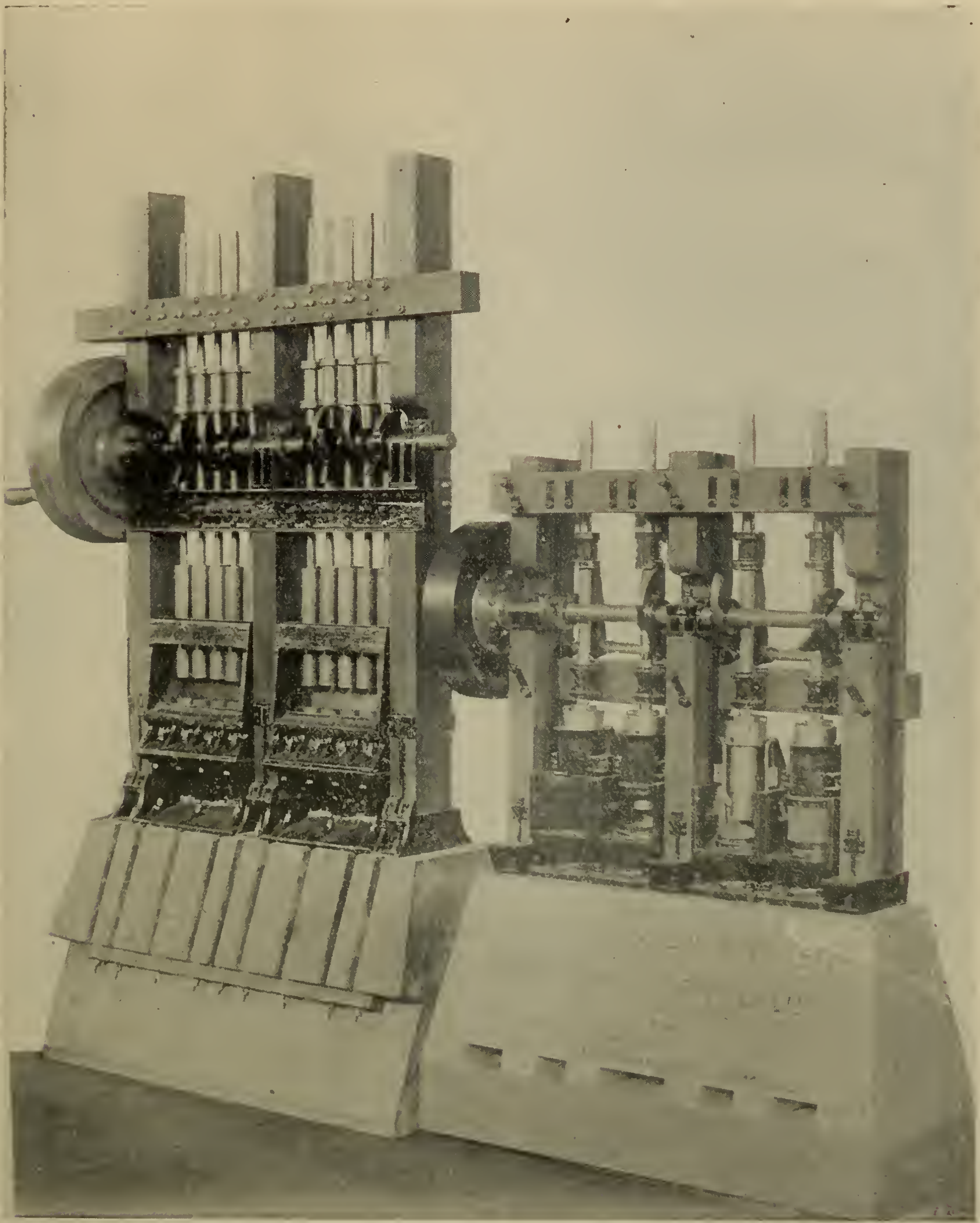

Fig. XVI.
Photograph of models to scale of a four stamp Nissen Battery and a ten stamp Battery (Modder "B" type).

Maintenance.—It seems hardly necessary to make any more direct claim for lower maintenance of Nissen stamps than is contained in almost every line of the paper. It would therefore appear that Mr. Laschinger did not give this paper that careful reading the author had reason to expect from one contributing to the discussion thereof. It must be quite evident to anyone taking this trouble that a great saving in maintenance will result. The item "cam shafts" has been referred to, also "wear of shoes and dies" and it is only natural to suppose that stems 5 in. in diameter will give a longer life, even without being better guided. Mr. Laschinger's reference to low cost of maintenance of the City Deep mill is unfortunate, when the expected economy from using the excessively heavy stamp has been quite disappointing.

Power Consumption.—It is a generally admitted fact that the one point established by the trials at the City Deep is the saving in power shown by the Nissen battery. It is fortunate therefore that it was tested against the heavy stamp having a short stem, as representing the latest practice on the Rand in multiple stamp type battery. There is nothing to indicate that the City Deep battery did not show as great a capacity as any other battery of equal weight and design could have done under the same conditions. It certainly did better than any battery of the old long stem short-head type could have done, as will be shown later. As already referred to in this reply, every precaution was taken to obtain reliable absolute results, as well as to deal fairly with both types of batteries. The power measurements were taken under the supervision of the engineers of the Central Mining and Investment Corporation, Ltd., and the tabulated statement given in the paper was prepared by them. That the total power figures were not given is explained by the fact that both the motor and transmission units were much larger than required for the 4-stamp Nissen battery, hence they were running inefficiently. It seems hardly necessary to state that the allowance made for friction was in no way guessed at, but was actually measured by reliable instruments used for the tests. By excluding these losses, the two types of batteries are put on a more equitable basis with the exception of guide friction, which is a considerable factor not always appreciated. The power required per ton of ore crushed by each type of stamp is given in Table I. of the paper, and as the results are so favourable to the Nissen stamp, Mr. Laschinger went to considerable trouble to find the explanation. Using the data given for number and height of drops, he figures the probable power requirements, which in each and every case except tests 5 and 6 of the City Deep battery, exceeds the actual power as shown. Thinking that this must be wrong, he calculated the probable drop from the actual power used, notwithstanding the fact that the actual friction is not known, and the results although remarkably similar for the Nissen stamp, according to his figures show an anomaly for the City Deep stamps in tests 5 and 6. There not being any explanation in the paper for the great difference shown, Mr. Laschinger promptly concludes there is something wrong, so considered it unfair to use the power figures of tests 5 and 6. The explanation however, is quite simple, and the author can only regret that the reasons were not given in the paper, because the results of tests 5 and 6 of the City Deep stamps instead of being useless are of greatest value, as indicating the relative efficiency of the long stem short-head stamp, to which reference has previously been made. Upon completion of tests 1, 2, 3 and 4, the Manager of the mine not being satisfied with the performance of the standard City Deep stamps decided to change the battery that was being used in the tests; accordingly this battery was completely reconstructed, the stems lengthened to 18 ft., they formerly being as illustrated in Mr. Kratz's contribution—page 185; the change being represented by Fig. 17 herewith. The weight was thus changed from 1,863 lb. to 1,775 lb., while the relative position of the guides are shown in Figs. 17 and 18. Figs. 19 and 20 represent the two types of Nissen stamps. It is hardly necessary to point out that the longer stem and consequently increased height of centre of gravity due to shortening the head, immediately increased the guide friction. If this fact is in any way doubted, it is only necessary to observe the two types of batteries in operation; the excessive vibration of the longer stems being most apparent. That Mr. Laschinger did not look for some practical explanation of this point, rather than enter upon a lengthy calculation without a foundation is to be regretted. Considering the tests 1, 2 and 3 as between the Nissen stamp and standard City Deep battery and test 5 and 6 against the long stem short-head type of battery, as now used in many thousands on the Rand, it is seen that they compare as follows with Nissen stamps:—

Tests 1, 2 and 3.

Average power consumption of Nissen stamps per ton of ore crushed ... 3·6 h.p.—hours
City Deep stamps ... 5·0 h.p.—hours
Saving in favour in Nissen stamps ... ... 1·4 h.p. equals 28% equal, at 80% efficiency to 1·75 h.p.—hours equals 1·3 electrical units equals ·8d.

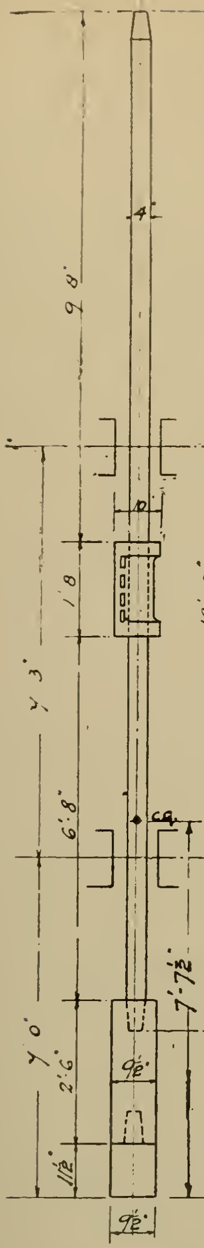

FIG. XVII.

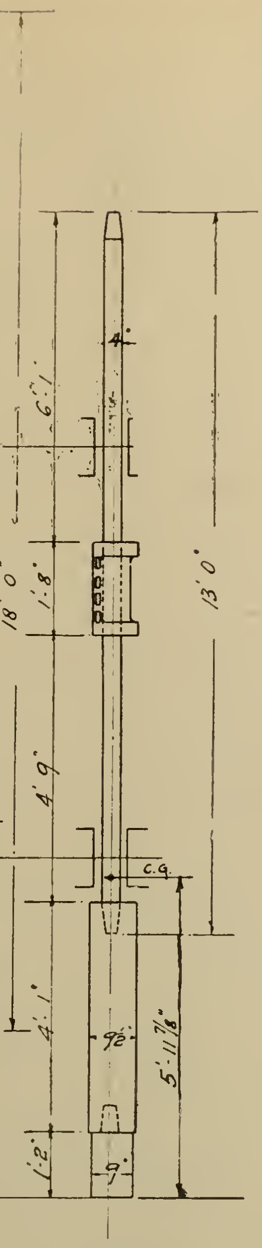

FIG. XVIII.

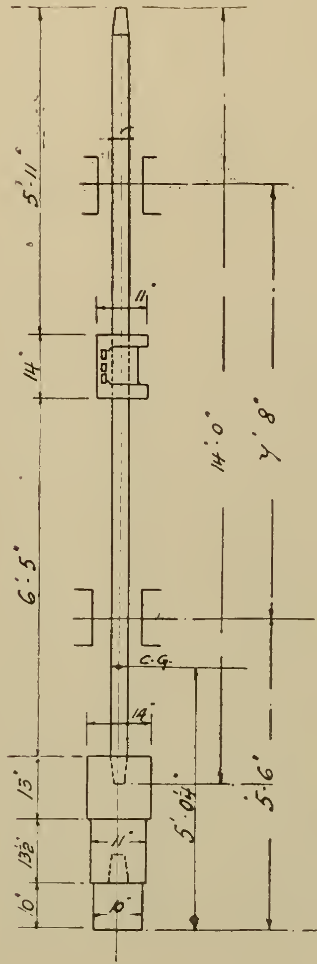

FIG. XIX.

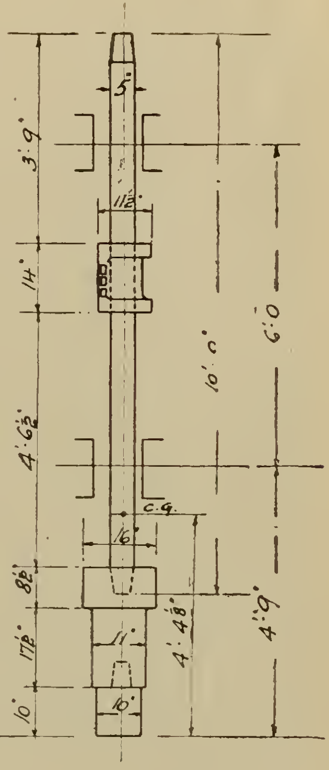

FIG. XX.

Tests 5 and 6.

Average power consumption of Nissen stamps per ton of ore crushed ...	3·06 h.p.—hours
Long-stem City Deep stamps	4·94 h p.—hours
Saving in favour of Nissen stamps	1·88 h.p.—hours
	equals 37%

Equal, at 80% efficiency to 2·30 h.p.—hours equals 1·75 electrical units equals 1·05d.

The Nissen stamp therefore when compared with the latest practice in heavy weight short stem type of stamp produces a saving of, ·8d. per ton milled, and when compared with the usual long stem type of stamp shows a saving of 1·05d. The average of the two is exactly what was claimed in the paper. As there are many thousands of long-stem short-head stamps now in use, which do not have the very rigid guide supports found in the City Deep battery, it is evident that a considerable waste of power is now going on.

Naturally there must be some guide friction with the Nissen stamp, although Mr. Laschinger's figures do not take this into account, but even if there was no such friction, the shoe in the Nissen battery must have remained approximately 1·35 ins. from the die. If guide friction is assumed in lieu of extra height of lift, the stamp would have dropped to say within 1·5 ins. of the die, the guide friction being ·15 in. or 2·15% of the power recorded. A theoretical assumption of City Deep stamp drop would show they came to rest ·7 in. from the die, and considering that it actually must have come within 1 in., the friction is represented by ·3 in. drop or 4% of the power. It was however certainly much more.

Considering the long stem short-head type of stamp (tests 5 and 6) the shoe has theoretically dropped to a point ·5 in. below the face of the die, but assuming that it dropped to within 1 in. of the die the guide friction would be represented by 1·5 in. or 20% of the total power.

The foregoing theoretical comparison is of no practical value, excepting to emphasise the waste of power through guide friction. A comparison between the standard City Deep stamps and the long stem type will bring this out, for no allowance need be made for greater screen efficiency and better feed control to which the better work of the Nissen stamp can be attributed. The author believes that the reduced guide friction of Nissen stamps is due to the short rigid stem working in well designed guides, and as the result of these tests, he has increased the diameter of the stem to 5 in. and shortened it to 10 ft. emulation of which is quite impossible in 5 stamp mortars without considerably increasing their length.

Grading.—In Mr. Laschinger's remarks on grading analysis, in which he called attention to the close resemblance of the crushed product, he failed to point out that in all cases, excepting test No. 6, the ore feed was coarser for Nissen stamps; nor is this taken into consideration by the Stadler method of figuring the relative mechanical efficiency.

Estimates.—By casting doubt on the author's comparative estimates relative to the cost of construction, Mr. Laschinger made the mistake of adopting Mr. Kratz's figures of the relative length of the two types of batteries for the same tonnage. He also disregarded the basis on which the estimate was made, since he persists in comparing 48 Nissen stamps to 70 of the ordinary City Deep type. Sufficient argument has already been advanced with reference to this battery in the foregoing portion of this reply. As the author's comparison was between 48 Nissen stamps of 2,000 lb. and of 100 stamps of the 5 stamp battery type of a reasonable weight (1,400 lb.), he can only state that the figures given in the paper have been checked by a local engineer of considerable experience, and as no detailed analysis was supplied in connection with the criticism, they must stand as they are.

The remarks made by Mr. Barry, as coming from a well known mine manager, are much appreciated by the author, as well as are those of Mr. John R. Williams, whose qualifications to pass an authoritative opinion, no one can question. Mr. Williams statement with reference to decreased sliming by Nissen stamps is borne out in the gradings given in connection with special capacities with fine screens. He will also be pleased to notice that the building is not greatly longer than is required for the ordinary stamps, although he has aptly answered this criticism.

Mr. Thomas has questioned the advisability of using 12% for interest, amelioration, depreciation and redemption of a reduction plant. Of course this is the author's personal opinion of the necessary tax, but this item is of minor importance in the estimates; it being of no consequence whether 10% or 12% is considered.

Answering Mr. Robertson's question concerning the age of the Nissen stamp, the author may say that the first patents were filed in U.S.A. in 1900, although the first battery of any importance was not started until 1903. The stamps then weighed 1,100 lb. each, yet the capacity shown was so great (9 tons per stamp through 1,200 screen) that the possibilities of such a type of stamp was at once evident. Since this date a gradual evolution has taken place until it now appears with the many mechanical advantages added to what is considered a correct principle, all of which is set forth in the paper.

The author believes that had he brought this type of stamp out to the Rand before the excessively heavy stamps had been tried, he would not have had the same opportunity to make such tests as have been made, since the advocate of the multiple stamp battery would have considered the same capacities possible from each stamp in a rectangular mortar.

Mr. Mackesy although not conclusively condemning the Nissen battery thinks the multiple stamp type can be made superior to it. Replying to this the author can only say that many skilled engineers as well as highly intelligent mill operators have been striving to do this for a great number of years, and the results of these efforts are represented by the modern batteries here on the Rand, and following the law of evolution, in which the author firmly believes, some time must yet elapse before Mr. Mackesy's idea is reached. In the meantime the Nissen stamp may be improved also.

With the installation of a large battery of Nissen stamps on the Rand and after the operators become familiar with this new type of stamp, the greater reduced maintenance expense will certainly be clearly established. The experience that will be gained by the mill men must certainly suggest further improvements that will yet increase the general worth of the Nissen stamp.

In closing this discussion the author ventures to consider that he has established the following advantages for the Nissen stamp over the best type of ordinary gravity stamp, viz :—

(1) Saving in power ranging from ·8d. to 1·0d. per ton of ore milled.
(2) Reduced maintenance expense.
(3) Reduced capital expenditure.

The Chairman: I wish to congratulate Mr Nissen on his spirited reply.

PRACTICAL APPLICATIONS OF THE SPECIFIC GRAVITY OR MOISTURE FLASK

(Read at November Meeting, 1911.)

By H. Stadler (Associate).

DISCUSSION.

Mr. Andrew F. Crosse (*Past-President*): At our late Exhibition there was a very interesting balance or apparatus shown for applying Mr. Stadler's principle of taking the specific gravity of the pulp. I do not know whether all of you noticed it, and I am sorry it is not here to-night, as I understood it would have been. I believe it is Mr. Stadler's invention; I am not sure. There are three beams. First of all, the weight of the flask is taken, then the next weight is taken to equal the thousand grammes of water which it would contain if it were full of water only, then it is filled with pulp perfectly full, and then one can read off directly the amount of water or slime, whichever is preferred, in that litre of pulp, and as long as Rand slime is being used, which has a specific gravity of 2·7 as near as possible, it is quite correct. In other parts of the Transvaal where the slime often has a higher specific gravity it would be necessary to make a correction, which is a simple matter.

The arrangement can also be used as an ordinary balance for weighing from 1 kgm. to 3 kgm. It is most convenient on a mine and it is just one of those inventions that ought to be encouraged, because everything that leads to the quick and rapid determination of anything we have to do or carry out must lead to greater accuracy and less chance of making mistakes through calculations. The result is simply read off without having to make any calculations of any kind. I hope at the next meeting Mr. Rowland will be able to have it here for you to see.

The Chairman: Mr. Rowland has undertaken to try and produce this apparatus at the next meeting.

NOTES ON ABSORPTION OF GOLD BY AMALGAMATED COPPER PLATES, CAST IRON RETORTS AND AMALGAM TRAYS.

(Read at December Meeting, 1911.)

By Prof. G. H. Stanley, A.R.S.M., F.I.C. and M. Thornton Murray, M.Sc.

DISCUSSION.

Mr. John Watson, F.I.C. (*Member*): Having read with great interest the valuable paper submitted to the December meeting, I wish at the outset to congratulate the authors on the very thorough manner in which they have carried out: firstly, the sampling by sections; secondly, the analysis of each sectional sample.

One can, I think, safely predict that a paper of this kind will often be referred to in the future by mine managers, metallurgists and students generally. The part which interested me most was that dealing with cast iron retorts. The author's samples were taken from a retort which had been in use from one to two years, or possibly longer. I have heard of one large mine on the Rand where their retorts lasted barely a year.

On the City and Suburban mine we had two cast iron retorts which had been in use for at least a year or two before the war, and which we used again about three times every month, from 1902 up to 1908. I think it was in the first half of 1908 that Mr. Reginald Payn, who was assisting me temporarily, drew my attention to two small pieces of gold adhering to the top of an old bearer bar which had been taken out. As the front of these retorts had the usual projecting flange I could not account for the presence of this gold unless it had leaked through some miniature flaw in the retort. At any rate, I never felt easy in my mind until we had the two old retorts out and two new ones in. On taking out the old retorts, they were well scraped on the inside and examined, but only in one place could I find a tiny hole (containing a kernel of retorted gold) about the size of a green pea. By way of sampling approximately these old retorts were drilled at points half way along from back to front, $\frac{3}{16}$ in. holes being drilled right through at the top, bottom and each side.

The drillings from these holes were assayed and gave the following results :—

Hole Marked.	Position.	By Pot Method Fine Gold	By Scorification (Dr. T. K. Rose) Fine Gold
		dwt.	dwt.
G	Side	14·00	170·00
X	Top or bottom	2,500·00	580·00
S	Side	91·00	15·00
P	Top or bottom	1,018·00	3,040·00
	Average	905·75 or 45·287 oz	951·25 or 47·562 oz per ton

These results show considerable variation, which is not surprising when one remembers that the samples were rough turnings. An attempt was made to pulverise each sample in a steel mortar to a finer state of division, but this was only partly successful. One point these assays do demonstrate is that the absorption of gold is much greater at the bottom than at the sides. I have marked "X" and "P" samples, "top or bottom" as the retorts had on several occasions been turned around 180°, the late top becoming the bottom. For several years we laid the bulk of the amalgam on the bottom of the white-washed retorts, with only a double sheet of newspaper between the retort and amalgam, with one full amalgam tray in front. The object of this was to get more amalgam in at a time. The first retort weighed 736 lb., the other weighed 635 lb. Although I had the second retort drilled in a similar manner, owing to pressure of other work the drillings were not assayed. About that time we had some changes in management and nothing was done with the two old retorts until the latter part of 1909 when our present manager, Mr. Glyn, had them both turned on the inside. The ends were cut off and a thickness of $\frac{1}{4}$ in. turnings was taken from the inside, all around, also from the inside of the ends. The shop foreman told me that the numerous heatings and coolings had made the metal almost as hard as steel.

Smelting of Iron Turnings.—An attempt was made to extract the gold from these iron turnings by fluxing with flowers of sulphur and a litharge flux, working with two No. 50 Battersea plumbago pots which were filled up several times before pouring. The lead buttons obtained varied in richness from ·81% to 2·78% fine gold. The slag carried 4·9 dwt. fine gold per ton. The matte, of which a large quantity was formed, went 76·7 dwt. fine gold per ton. The fumes of sulphur dioxide inhaled from this smelting were decidedly disagreeable and, after trying this method, the lead bullion obtained and the balance of iron turnings were sent to the Tavener furnace. This attempt at melting, with formation of ferrous sulphide, was more or less of a failure, chiefly owing to the insufficient temperature and no blast.

It was, I believe, the great Michael Faraday who first said that in physical and chemical experiments our failures should be recorded just as carefully as our successes. Possibly some metallurgist who has extracted the gold from old retort turnings will give us his best method.

The turnings from inside of the first old retort assayed 189·75 oz. fine gold per ton. The balance of old retort turnings sent to the cyanide department gave an average value of 96·4 oz. fine gold per ton.

Steel Retorts.—It would be interesting to hear if the lighter steel retorts put in a few years ago, on some of the newer mines, are doing better work and lasting as well as cast iron retorts.

In conclusion, my thanks are due to our general manager, Mr. Chas. Glyn, for permission to publish these notes.

The Chairman : Mr. Donaldson will read the next paper, as Mr. Cullen is still, I am sorry to say, unable, through ill-health, to be present.

THE "LUMINATOR" PURIFICATION TREATMENT OF WATER FOR STEAM BOILERS.

By William Cullen (Past-President).

Two or three years ago the engineering world learned for the first time that when certain feed waters for boilers were run over aluminium plates, under fairly well defined conditions, scaling was entirely prevented, and scale which had already been formed was loosened and converted into a sludge. Such in brief were the claims made by the discoverers and they were sufficiently startling in all conscience. I need not enlarge on the great disadvantages of scale ; indeed they are obvious enough when considered only in the light of the efforts made to prevent its formation. The new method was so simple that few believed in its efficiency, and to-day there are still many unbelievers. Two years ago I was numbered among the sceptics, but to-day I am convinced that, for certain waters at least, the process is absolutely efficacious. Like many others I instituted experiments about which more anon—but I can say here that those experiments all led to negative results, so that they do not enable me to explain in the very least what chemical or physical change takes place in the water. That others have the same difficulties I know, as will be seen from the following brief outline of their views. So far as I can discover there are only two records of real scientific attempts to solve the mystery, and both are recorded in the *Journal of the Society of Chemical Industry.** From the first of these (see references) it would appear that the discoverer was named Brandes, and he gave the name of "luminator" to a vessel made of aluminium. In the original patent which was taken out in the names of Neff and Brandes, there was no reference to the prevention of scale in boilers, but the water was supposed to have wonderful cleansing properties, soiled linen being cleansed in a very short time. I have never heard that anything came of this. I have never seen a luminator and I do not think there is one in South Africa—but it appears to be a launder made of corrugated aluminium. While flowing through this vessel something happens which even the inventor could not explain—but the effects I have already mentioned :—

> "The theory of the action of the luminator is that by the passage of water over the metal channels at certain speeds a current of electricity is induced, the water being negative and the plates positive and a kind of ionisation of the salts takes place which makes them fall out in an amorphous form : at the same time aluminium is by friction and electrical action abraded from the surface as colloidal aluminium, which after a time is changed in the water."

Just at this stage let me say that in order to make this theory fit in with facts the aluminium must lose weight, but I found in laboratory experiments that this was not so ; neither have my somewhat crude luminators lost weight after a year's continual work. The author of the paper indeed had to admit this for he said that the loss was infinitesimal. The best results (according to the author) are obtained when the luminator is facing either north or south, but an essential condition is that it be exposed to direct light.

In his reply to a very remarkable discussion the author gave some very interesting facts, among which I quote the following :—

(1). A litre of water run through the luminator ten times lost 20° temporary hardness.
(2). There is no limit to the hardness which could be treated in the apparatus, but for particularly hard waters, say, 80 gr. per gallon on Clark's scale, a somewhat lengthy apparatus would be required.
(3). Foaming always ceased when a luminator was used.
(4). Milk when passed through the apparatus kept an extraordinarily long time.
(5). Traces of nitrites and nitrates were decomposed and a 2% solution of magnesium bicarbonate had been converted into carbonate.
(6). The corrugations — which are at right angles to the flow—have merely the function of increasing the surface.
(7). Certain acid waters and salt water cannot be treated.

The second communication (H. B. Lake) was made as recently as December last and is a record of a very interesting series of experiments to determine what change, if any, is made in the water by it passing through the luminator under conditions which vary both as to light and speed. The differences found were infinitesimal and were in my opinion quite within the limits of observation. However, the author introduced into one of his series of experiments a new factor, viz., the addition of aluminium as sulphate in the proportion of 3 per 100,000 his idea evidently being that by the presence of sulphate in the water part of the aluminium was converted into sulphate. Whether this is so or not the action on the water which was subjected to the luminator treatment after admixture with sulphate of alumina was remarkable indeed—the soluble lime being increased seven-fold. What I mean by soluble lime is, of course, a transference of so-called "temporary hardness" or bicarbonate into "permanent hardness" form or "sulphate.

About 18 months ago I had some rough experiments carried out on the lines of those indicated by the two authors, to whom I have just referred, and although the results were practically negative the experiments are perhaps worth recording. The corrugated aluminium launder was about 5 ft. long and about 10 in. broad. It was exposed to sunlight during the experiments and the water trickled down slowly over the corrugations. The surface was inclined to become dirty—no doubt from suspended matter in the water—so it was burnished afresh for each experiment.

Experiment 1.—Treated water was compared with untreated by titration with soap solution. The latter indicated 9·17 $CaCO_3$ per 100,000 against 8·47 for the former—or practically no difference.

Experiment 2.—Two litres each of treated and untreated water were boiled down to a bulk of about 100 cc. and were then transferred to beakers and still further boiled down. In both cases the scale stuck to the side of the beakers but in the case of the treated water it was more patchy and less adherent than the other.

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* (1) Vol. xxv., p. 933—T. B. Duggan.
(2) Vol. xxvi., p. 57—H. B. Lake.

Experiment 3.—This is a repetition of Experiment 2 but on a much larger scale, 400 litres of each kind of water being employed. The final boiling down was done in porcelain basins but no difference was observed either in the behaviour or the nature of the residue.

These experiments, as I have already said, lead to nothing. So it was determined to try the luminator treatment on a practical scale. Fortunately there were two small 25 h.p. (nominal) Tangye Colonial tubular boilers situated at different parts of the factory available for the experiment. They worked at 100 lb. pressure and used about 3,000 gallons of water in the 24 hours. Both were provided with corrugated launders 8 ft. long by 1 ft. 6 in. wide set at an angle of 45°. One faced north and the other south. The water trickled over them into a feed water tank and as little steam was used at night the run took place practically all in day-light. I may say also that ordinarily speaking there was so much scale produced that the boiler had to be cleaned out every three or four weeks. Moreover the scale was of a very adherent nature, so that it was difficult to chip off. It consisted in the main of ferric oxide, and sulphates which have this adherent property. As will be seen from the following incomplete examination the water itself has very little temporary hardness or bicarbonate of lime.

	Before Boiling mgm. per litre.	After Boiling mgm. per litre.
Total solids in 100° C.	405	390
$Fe_2O_3 + Al_2O_3$ ...	1·2	·9
CaO	52·5	51·0
MgO	76·8	75·0
SiO_2	4·0	3·35

Even if the above bases were combined with SO_3 and Cl which were not determined, there would still be a big deficiency which presumably consists of alkaline chlorides and sulphates—so that the water is abnormally high in these constituents. Admittedly neither the water nor the scale have been examined very thoroughly, but attention was more directed to the results of the practical trials which can be given in a very few words. First of all, however, I should state that the launders were only cleaned once a week and no special care was taken to burnish them.

The experiments were started when the boilers were nearly ready for scaling, and the first effect noticed was that the scale in position commenced to loosen and much of it was in consequence removed at the blow-off. Then, when the boilers were opened up for cleaning much less scale than usual was observed and what little there was could easily be removed without the slightest difficulty. The aluminium launders have now been in use for fifteen months, and, although on the whole, the improvement is not now so marked as at first that may be the result of want of care and attention to the launders. Certainly the improvement noted at first was very remarkable and striking. Now I do not say that this luminator treatment is nearly so effective as the Kennicot and others, which are well known out here, but for small isolated boiler plants for which it would not pay to instal a water purification system the luminator treatment is certainly worth trying. I do not think anyone could guarantee that it would be effective, there is so much conflicting evidence which is due in the main, I think, to the fact that the boiler water used by some investigators is quite different from others. Now, why should running water over aluminium effect any change at all? Frankly, I do not know. So far as I can gather no one has yet succeeded in proving a chemical change. Then it must be physical, one would say, but here again nothing has been proved. Better aeration occurs to one but this is really no explanation, so I leave the problem to be solved by the members of this Society. I might remark in conclusion that the mystery of the divining rod is also awaiting solution.

The Chairman: I beg to propose a vote of thanks to Mr. Cullen for this paper. Is there any discussion on the paper, to-night?

Mr. H. A. White (*Vice-President*): As Mr. Cullen seems to deny that even the doubtful results he has obtained can be traced to any possible physical or chemical cause, it is perhaps a pity that we have in this country no Psychical Society where reasons for believing in such things as divining rods and patent boiler compounds could be more suitably investigated!

ACCIDENTS IN TRANSVAAL MINES.

(WITH SPECIAL REFERENCE TO FALLS OF GROUND.)

(*Read at November Meeting, 1911.*)

By JAMES CHILTON (Member).

DISCUSSION

Mr. R. A. Barry (*Member*): It is presumptuous on my part at this stage perhaps to say aught on this subject after so many distinguished critics have shed their light upon the causes of the high accident rate on the Witwatersrand, and the way in which a better state of affairs could be brought about, but I make bold to claim your attention to point to a cause for many of our troubles which I believe lies at the root of the whole matter:—

We are all sadly aware of the fact that our accident rate is very high, how high, in comparison with other countries which enjoy different conditions appears to me to be most difficult to arrive at. To reach a fair basis of comparison would need a much closer investigation into the conditions prevailing in each separate country than appears to me to have been entered into.

Coal and metal mining are admittedly in quite different categories when considering accidents. Skilled and unskilled white employees also should each have their separate divisions, and whether such men were actually conducting their own operations and therefore were more or less limited to the risks which they themselves could control in some degree, or whether they were in charge of gangs of workers, and if so to what degree of intelligence and training such workers had attained, are points which, I think, should be stated before attempting to throw the onus on mining conduct. That accidents, with a force of highly intelligent beings, such as the Chinese proved themselves to be, could be far better kept in check than with the continually shifting hordes of savages which it is mostly our unhappy fate to handle now-a-days, there can be no argument.

I think that ill considered comparisons can only do harm by laying a misleading idea before the world, and thus tending to drive workers away from us and into other fields, workers whose energy and brains our own industry can ill afford to dispense with.

Figures have been adduced to prove that the number of changes which occur constantly on all our mines amongst our white labour force is inordinately high, and again, further figures have been brought to show that where such changes occur most there the accident rate has been highest, and what looks perhaps like the obvious deduction has been made from these facts, *i.e.*, that our accident rate is largely due to our unfortunate system of moving men on so frequently. Stop changing your men, they say, and you will largely decrease your accidents. Now I hold that we must look much deeper for the root of the trouble. It is admitted by all who look into the matter that the hope of arresting the flux and efflux of our coloured labour force has no place in practical politics to-day, so that some 85% of our underground labour force is doomed to a perpetual flow, for reasons outside our control, and though, if we could secure more stable conditions amongst the balance of the underground labour force (the 15% represented by the white workers), we should undoubtedly achieve something, yet see what a vast field of impossibility still faces us after achieving this small but difficult part of our task. But apart from the unapproachable field and dealing only with the white labour force—the controllable 15% of our underground force—I would contend that we must look deeper than the mere shifting of our workers from mine to mine for our accident rate. Why do they shift? No manager ever moves a man on just for the fun of the thing, but he is ever anxious to keep any man who has any pretensions to being a good miner, though, alas, the percentage of such men amongst the underground hands is by no means what it should be. A certain number of underground hands are absolutely necessary in each mine, and therefore after getting as many good miners as possible, the balance of the force has to be made up with the next best available, and it is this balance which the manager is ever trying to replace with better men. The fact is that there are not enough miners to go round, good men can pick and choose and move about in the most extraordinary way to suit any whim they may have, for billets are always obtainable; whilst poor miners are for ever being turned over by one mine after another in the search for the unobtainable. Hence it follows that mines most favourably situated, or which have the best and nicest conditions get the first pick of the good miners and show the steadiest labour force, whilst those with bad working conditions find themselves left with the residue. Let the ground be a bit too steep, a bit wet, or the least bit difficult and it will be found at once that the best miner shuns it, saying quite candidly that at such and such a mine he can make the same money with much less discomfort and hard work, so why should he stay with you. The inevitable result of this very natural process of selection is that the most difficult mines which call for the best class of men, get left with the poorest selection and up goes their accident rate: they change and change in the effort to get better men, and your statistician is led to the belief that the trouble is due to the rate of change. He is sadly mistaken. The trouble is due to the fact that there are not enough good men to go round, or anything like it, and the rate of change and the rate of accident are chiefly due to this fact.

Every mine on the Rand is affected by this scarcity of good men and to my mind the industry is as hard, if not harder hit, by the scarcity of really skilled white underground workers, as it is by the much more generally recognised scarcity of unskilled labourers. And to what is this scarcity attributable? There can be but one answer, *miners' phthisis*. This hits at the root of everything.

Miners' phthisis is destroying our best miners just when they are reaching the point when they are most valuable to the industry. After seven or eight years underground there are but few, if any, who come out untainted with this dread disease. We should by this time have a reservoir of good experienced men from which to draw, men who have had good experience here and have decided to make this country their home; but what do we find in the reservoir after all these years during which our mining industry has been going—nothing but "miners' phthisis" It hits us in every direction. We must pay bigger wages to try and blind men to the risk which they take. We get inefficiency in most of our underground work despite our tremendous wages. We find our Rand filled with miners who do not mean to make the country their home but only come to snatch what they can from the jaws of disease, and we find the very heavy accident rate under which we labour and which we try to attribute to a hundred and one things, but which lies really at the door of phthisis—a door, mark you, which can be closed, if the effort be made.

Mr. W. Cullen (*Past-President*): I am taking part in this discussion for the fourth time—not to create a record—but to correct an impression. It will be remembered that at last meeting I gave figures which purported to show that a very large number of mine managers were changed every year. Now these changes are in reality great, but not so great as my figures would indicate. I am credibly informed that owing to the method adopted for recording changes in the Mines Department office—one real change may be recorded as 3 or even as 6. For instance if a manager goes away on leave and comes back two changes will be recorded, but if his subordinates are advanced a step during his absence to subside again at his return, this would mean a further multiplication of changes. Such changes were not in my mind when I submitted my figures, so I take this opportunity of correcting a wrong impression—but one under which I myself also laboured at the time.

The Chairman: The discussion on this paper will be closed at the next meeting.

SOME OBSERVATIONS ON ANCIENT MINE WORKINGS IN THE TRANSVAAL.

(*Read at January Meeting, 1912.*)

By T. G. Trevor, A.R.S.M., Acting Inspector of Mines, Pretoria (Member).

DISCUSSION.

Prof. G. H. Stanley (*Member of Council*): The author's welcome paper brings up a subject of great interest, and one which has not received in the past the attention which it certainly deserves. Personally, I have read it with great pleasure, as I have always felt a keen interest in this ancient mining and have already put some observations on record before another Society.*

These referred to the Messina copper old workings which I had the good fortune to visit about five years ago.

I quite agree with the author as to the extreme antiquity of the commencement of the workings, as it is, I think, fairly well proved that gold mining was carried on further north in quite a highly skilled manner and on a very considerable scale in perhaps Phœnician times and possibly even earlier still. But I do not think that Transvaal copper mining was necessarily quite contemporaneous at the outset, for there is also plenty of evidence that gold lode mining in Rhodesia must have persisted for centuries subsequently, though with decreasing skill, shown by the state of the workings themselves and of ruins of probably contemporaneous buildings; but at the time of the Portuguese appearance towards the end of the fifteenth century all lode mining had apparently stopped, though it was occasionally revived by them. It seems quite possible, though, that working proceeded on a small scale at Messina at various intervals down to recent times, and in that way a very considerable total of ore may have been extracted by other than the original workers; the total time involved being possibly over 1,000 years. The present native inhabitants know very little about the workings, but state that the deposits were worked by their grandfathers, apparently in rather desultory fashion, and not more recently, the work having stopped when one of the workings fell in and buried a number of men. I have heard of traditions in other localities relating to pale-faced men who wore some head covering, probably turbans, and who directed the operations and took most of the output, and this would seem to point to a possible Asiatic connection with the work. There is no record of any great mining activity on the part of the early Portuguese, but before their advent there was considerable traffic along the East Coast from Arabia and India, considerable quantities of gold passing out of the country in this way, although the trade had long been interfered with by severe native wars. Native traditions persist for a very long while, and it is therefore possible that the traditions referred to originated in pre-Portuguese days and have persisted in spite of tribal migrations and wars.

I am afraid the fascination of speculation on the origin of these workings, an intention specifically disclaimed by the author, has caused me to wander from the real subject matter of the paper, which concerns facts, so to return to these, I should like to add that I also noticed the almost complete absence of copper minerals from the dumps of waste and was greatly struck by their comparatively small size. I was fortunate enough to be able to take a photo of some stone hammers, iron gads and copper ingots, which is reproduced hereunder, and I have brought one of the hammers for your inspection.

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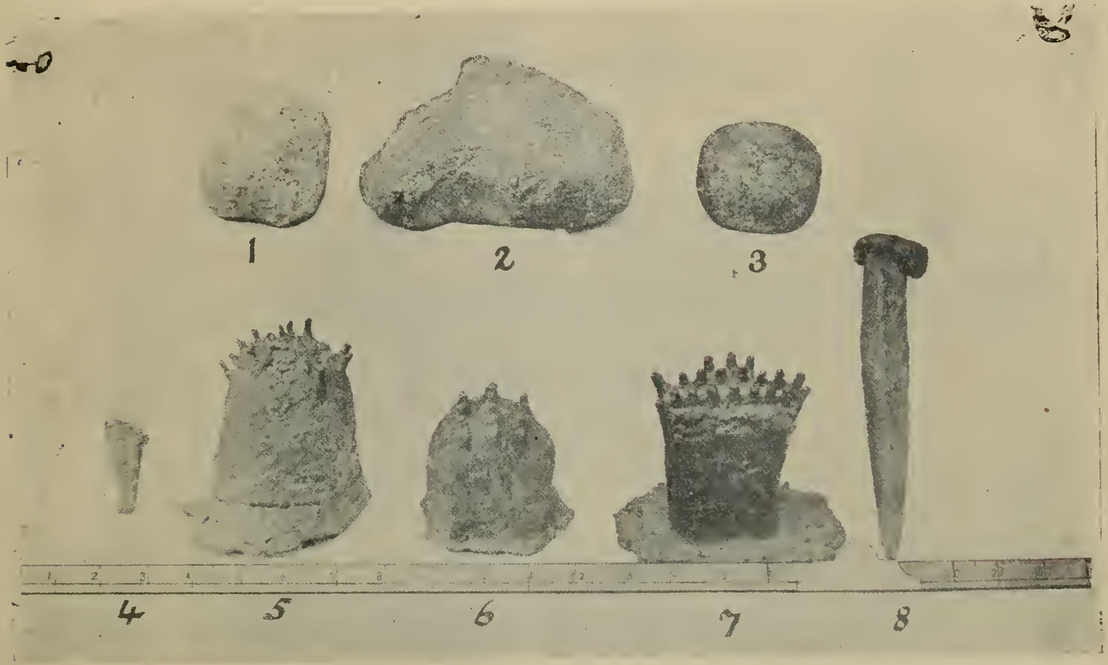
*University of Durham Philosophical Society, Vol. III., Part V.

I should have liked to make an analysis of the copper and therefore obtained permission to sample one of the ingots by drilling; however, the drill was very quickly stopped by some hard material in the interior of the copper—probably granite from a microscopic examination of the fragments—and this idea had to be abandoned. This may indicate an intentional sophistication in the same manner that other natives adopt with rubber !



Nos. 1, 2 and 3, Stone Hammers. Nos. 4 and 8, Iron Gads. Nos. 5, 6 and 7, Copper Ingots.

It is difficult to surmise the true cause of the abandonment of works (*i.e.* on the large scale) as there was plenty of ore available at workable depth ; several rather obvious causes present themselves, such as native wars stopping either work or transport ; but it may be that only the smelting of the oxidised outcrop was understood, though certainly no rejected glance is to be found in the waste dumps. The smelting was apparently carried out on the slopes of neighbouring hills, at the settlements, which were so placed for strategic reasons. There is no sign of the smelting floors above surface now, but at a depth of six inches or so layers of ash, cinders, slag, etc., with fragments of twyers made of a mixture of clay and quartz, are to be found in several places. These I should think are comparatively recent. I did not find anything which could be identified as part of a crucible, and as the twyer noses were covered with slag stained with copper oxide and containing beads of copper, it would appear that smelting was performed in some sort of hearth. A fair sample of slag was analysed with the following results, which seem to show that limestone must have been used as a flux and indicates the possession of considerable metallurgical knowledge by the workers,

	Per cent.
Silica, SiO_2	41·25
Alumina, Al_2O_3 ...	10·10
Iron, equivalent to FeO ...	18·61
Lime, CaO	21·08
Magnesia, MgO ...	1·02
Copper, equivalent to Cu_2O	2·58
Potash, K_2O	1·20
Soda, Na_2O	·90
Sulphur, equivalent to SO_3	·87
Water (combined) ...	·56

The figures represent the analysis of the sample of slag after separating beads of metallic copper, which amounted to 1·8%, and drying at 100° C.

In one of the smaller workings I noticed a Baobab tree growing, about 6 ft. in diameter. These trees live to a great age, one in Rhodesia of diameter about 38 ft. having been estimated to be 1,400 years old, and though I do not know whether it would be safe to estimate the age of the smaller one mentioned at about 200 years, in any case the working was obviously older than the tree, a photo of which is shown.

Though I have seen in the Vryheid district and elsewhere small native workings in iron ore for pigments, I have not visited any other really ancient workings, but I was recently asked to examine samples taken chiefly from the pillars of a very large working in the north eastern Transvaal with a view to discover what the mineral mined for had been. Though I made a very complete examination I could find nothing of any possible value except iron, and this, though chiefly in the form of specular hematite, I should scarcely think could have been of value as a source of iron, since richer iron deposits probably occur in the same district. These samples were taken, I am informed, from old stopes reached by vertical shafts, the bed mined being inclined. The roof was properly supported by pillars, and the whole appearance gave the impression of having been the work of skilled miners. I have also seen caves, etc., in the dolomite country in the Western Transvaal which might be mistaken for old workings, particularly certain surface depressions which I concluded had been produced by the falling in of caves below; very little examination is required to show the true nature of these.

Before concluding, I should like to refer to the analyses given in the paper. That of the gad shows it to be surprisingly free from the slag contained in greater or less quantity in all wrought iron (or steel produced similarly) and rather higher in carbon than would be expected from its probable mode of manufacture unless the natives understood how to convert iron to steel; the bronze is a most peculiar product: assuming that *alumina* is a misprint for *aluminium*, I should like to know whether any really was present in the alloy, and whether the metal appeared to be fit for anything at all with so much arsenic present. The presence of nickel is also noteworthy, and I do not remember another case in which it has been found in pre-historic bronzes, though arsenic was used for hardening purposes.

In conclusion, I think the best thanks of the Society are due to Mr. Trevor for his most interesting paper.

The proceedings then terminated.

Visit to Coalbrook Colliery.

Saturday, 9th March, 1912.

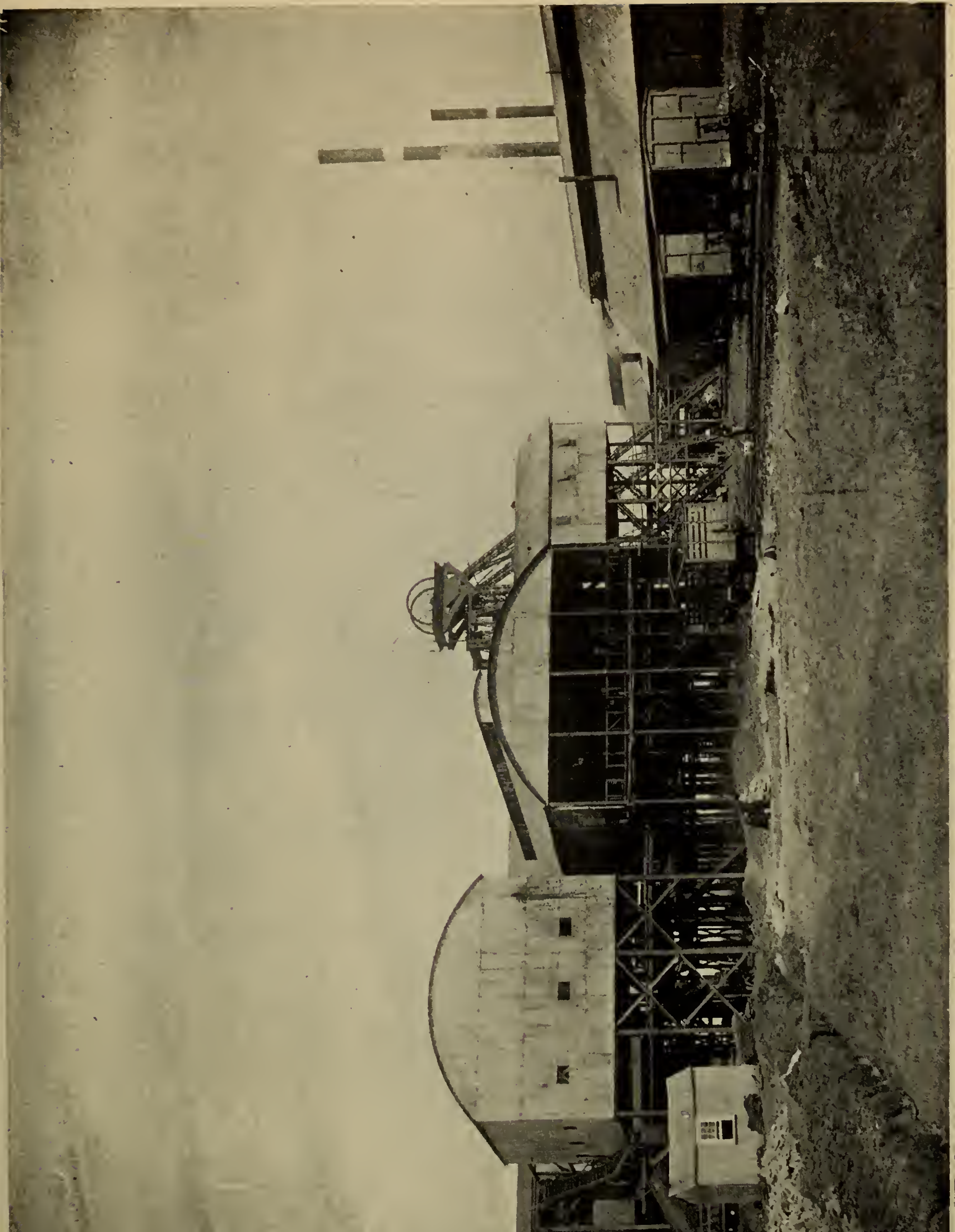
The Clydesdale Company's colliery, at Coalbrook, in the Orange Free State, was visited on Saturday, 9th March, by about 100 members of the Chemical, Metallurgical and Mining Society, for whom special train arrangements had been made. The visitors were headed by Mr. W. R. Dowling, one of the Vice-Presidents, and were accompanied by Mr. A. Mackie Niven, managing director of the company. The colliery was reached shortly after 2 o'clock, and the visitors immediately proceeded to inspect the surface machinery, plant and buildings in five parties, conducted by Mr. Mathew Cullen, general manager; Mr. J. Boles, manager; Mr. T. Wilson, mine engineer; Mr. John Donaldson, mine captain; and Mr. T. Hunter, mine secretary. Afterwards most of the visitors inspected the underground workings.

The following description of the property was circulated to the visitors in a handsome *brochure*:—

The Coalbrook Colliery is owned by the Clydesdale (Transvaal) Collieries, Limited, which also has a colliery at Springs, Transvaal. It is situated on the farm Taaiboschspruit, district of Heilbron, Orange Free State, about 5 miles south of Coalbrook Station, on the main line of railway from Germiston to Bloemfontein. Coalbrook Station is 13 miles south of Vereeniging.

The Company owns an area of ground 17,659 acres in extent, viz.:—

Farm Taaiboschspruit No. 205, mineral rights, 10,108 acres. Farm Rietfontein, No. 150, freehold, 7,551 acres.



General view of Surface Works at Coalbrook Colliery.

Coal has been proved in seven boreholes, over a very large area, the life of the mine being practically unlimited.

Sinking.—Work was begun in the year 1904, two shafts being sunk to the coal which was entered at 437 ft. 3 in. from the surface and passed through at 470 ft., the shafts being stopped in conglomerate at a total depth of 487 ft. 3 in., the coal seam is 32 ft. 9 in. thick. The section to the coal shows sandstone, dolerite, shales and conglomerates: the dolerite is 77 ft. 4 in. thick and sinking in it was difficult and costly. The coal was cut in the shafts in January and March, 1905, respectively, and an output was begun in April of the same year; the mine has been worked regularly since, and the output now averages 11,000 to 12,000 tons per month.

Shafts.—No. 1, the up-cast shaft, measures 11 ft. × 12 ft. 10 in. in the clear and is timbered throughout. It can be used as a winding shaft when required, and is also fitted with a ladderway. No. 2, the down-cast shaft, is the main winding shaft; it measures 11 ft. × 14 ft. 2½ in. in the clear and is fitted with wooden guides. It is equipped for winding 2,000 tons per day. The cages are single decked but can be double decked if desired. They carry two tubs each holding about 2,000 lb. of coal. The winding engine is direct coupled with 20 in. cylinders and 28 in. stroke. Winding drums are 10 ft. in diameter, the makers being Messrs. Barclay & Co., Kilmarnock.

Ventilating Plant.—This consists of a high speed Waddle fan 21 ft. in diameter, capable of running at 100 revolutions and producing 100,000 cub. ft. of air per minute. The fan is direct driven by a steam engine.

Banking and Screening.—The banking is done at one level and the tubs are turned on to jigging screens which separate the small coal, the large coal passing over to the main belts where it is cleaned, the inferior coal and shale being passed down shoots to a travelling belt. The cleaning belts are of the bar pattern, and are each 71 ft. in length.

Washing Plant.—The whole of the small coal is carried by conveyor belts and an elevator to the washing plant, which consists of three Bash tanks and a Felspar washer. The small coal is sorted through a revolving screen into three sizes —nuts, peas and pearls before washing. All these qualities are loaded direct into wagons from the hoppers under the washer. The small coal refuse passes along launders and through pipes into the settling ponds, the water being pumped back by a centrifugal pump.

Headgear and Buildings.—The headgear and buildings, which are of steel construction throughout, were supplied and erected by Messrs. J. and R. Niven of Johannesburg, and comprise:—

Headgear No. 2 shaft of steel lattice construction, 72 ft. high, having 2 winding pulleys 12 ft. diameter. Heapstead building, 149 ft. 3 in. long × 50 ft. wide × 42 ft. high to eaves, arranged for 4 tipplers and creeper haulage. Picking house, 100 ft. 2 in. long × 46 ft. 4 in. wide × 32 ft. 6 in. high to eaves. Dross house, 36 ft. long × 21 ft. wide × 40 ft. 6 in. high to eaves. Washer house, 43 ft. 6 in. long × 38 ft. 9 in. wide × 44 ft. high to eaves, arranged with additional bin extension of some 50 ft. long. Boiler house, 90 ft. long × 44 ft. span × 23 ft. high to eaves.

Electric Lighting and Power Plant.—A complete lighting and power plant has been recently erected by Messrs. R. H. Gould & Co., Johannesburg.

Generating Plant.—This consists of:—

One-220 B.H.P. Bellis & Morcom 2-crank compound engine running at 425 R.P.M., at present exhausting into the atmosphere. Direct coupled to one Johnson & Phillips continuous current 3 wire generator. The generator is 5% over compounded and has only 1 commutator and 3 slip rings. The potential difference at the brushes on the commutator is 500 volts at no load and 525 volts at full load and the normal full load output is 300 amps. From the three slip rings a 3 phase current is taken to three choking coils immersed in oil. The choking coils are connected in star and from the star-point the neutral wire is taken. This is earthed at the main switchboard end and the voltage between any one of the 2 outers and the neutral is 250/262½ volts. All the motors are connected on the outers and they are therefore ordinary D.C. 500 volt machines. All the lights on the other hand are connected between any one outer and the neutral, 250 volt lamps being employed.

Motors.—There are 2-80 h.p. motors compound wound, one for driving the screening plant, one for driving the washing plant. The washing plant motor is fitted with a speed regulating device which permits of running the plant from zero up to the full working speed. There is a 50 h.p. motor driving the underground haulage, a 25 h.p. motor driving one Gould's triplex pump 6in. × 8 in., a 16 h.p. motor driving the workshops, a 5 h.p. motor driving pump 1½ miles distance, and one 25 h.p. motor driving compressor underground. In the engine room there is a main lighting and power distribution switchboard. All the switchgear is of Messrs. Ferranti, Ltd's. manufacture and the switchboards, both in the engine room and for the motors were made and designed in Messrs. R. H. Gould's workshops, Johannesburg. The leads of the 80 h.p. motors and of the underground power supply are made by means of paper insulated and lead covered cable. Each motor is protected by Ferranti automatic overload switches and the motor circuits themselves are protected by Ferranti type non-arcing fuses. There is no porcelain about these fuses and they are removed by means of a handle specially provided for the purpose.

Lighting Plant.—All the buildings on the property are electrically lighted, the protective fuses being placed in centrally arranged positions, eleven to twelve lamps being the maximum on any one circuit.

Underground.—The main drives have been well lighted with lamps 40 to 50 ft. apart. Each group of lamps, from 15 to 20, being protected by their own cartridge type fuse placed in a cast iron water tight switch and fuse box. This arrangement is particularly valuable in a coal mine where the wires are loosened at times through the hanging wall coming down. In such a case only the lamps on that one circuit effected by the fall go out, not affecting the whole plant.

Telephones.—A telephonic installation has been installed all over the property and underground.

Boiler House.—Five Babcock & Wilcox water tube boilers, each 110 h.p. have been erected; they are hand fired and are each capable of evaporating 3,500 gallons of water per hour. The feed water is heated by the exhaust steam from the winding and electric engines.

Shops.—These consist of fitter's, blacksmith's, carpenter's and electrician's shops. They are fitted with sawing, wood morticing, lathe, drilling, punching, shearing and screening machines, also a power hammer; all of which are driven by a motor.

Buildings.—The married and single quarters, the native compound, the hall and boarding house, and other principal buildings are built of local freestone.

Quarry.—There is a freestone quarry opened on the property, from which all stone for the foundations and the principal buildings has been taken.

Branch Railway and Siding.—The branch railway, which is operated by the company's locomotives, is 5½ miles in length and has a gradient of 1 in 75. The sidings are so graded that the wagons, when loaded, run from under the shoots to the weighbridge by gravitation. At a point about a mile from Coalbrook Station, Steenpan, the well known freestone quarry, is connected with the branch railway, and the company's locomotives haul the stone to Coalbrook Station.

Water Supply.—This is taken from a dam on the Taaibosch Spruit, at a point 800 yards distant from the mine. The dam has a capacity of 10 million gallons. At a point 1½ miles distant on the same spruit there is a practically inexhaustible supply of water, which is treated as a reserve. A 5 h.p. motor and centrifugal pump is erected here which can be started up at short notice when required.

Underground.—The coal is 32 ft. 9 in. thick, but only the lower portion averaging 9 ft. is being worked. The seam is worked on the pillar and stall method. The mine is laid out on the panel system, to allow of the sealing up of any section in case of need. The benching method of breaking the coal is adopted, there being a good holing about 5 ft. from the pavement; the coal breaks large and cleanly. The measures dip slightly south west and the main drives, which run east and west, are fitted with mechanical haulages driven by a 50 h.p. motor. A number of mules are used to draw coal to the main haulages. The mine makes very little water, about 3,000 gallons per day, which is dealt with by a Gould pump.

Agricultural.—The company has some 1,000 acres under cultivation on Rietfontein, the ploughing being done by oxen. A sufficient quantity of maize and other crops is grown to supply the native compound.

The inspections over, refreshments were served, after which:

Mr. W. R. Dowling (Vice-President), briefly proposed a vote of thanks to the management, and congratulated the company on the excellent equipment and working.

Mr. A. Mackie Niven humorously remarked, in reply, that one felt, in bringing a society such as theirs to a coalfield, there was very little to show them. On that mine they had spent only £180,000 during the last few years, and of course on the Rand they had mines where more than a million of money had been spent, and where they had much more to show. Anyway, they could show something for the money, and he claimed that the colliery was second to none in efficiency in South Africa. They had at present an output of rather better than two tons per day per native boy, and as far as underground boys only were concerned the return was rather more three tons per day. He did not think that could be bettered.

Mr. Mathew Cullen, manager of the mine, also replied and expressed the hope that at some future date they would be able to welcome the Society at the mine again.

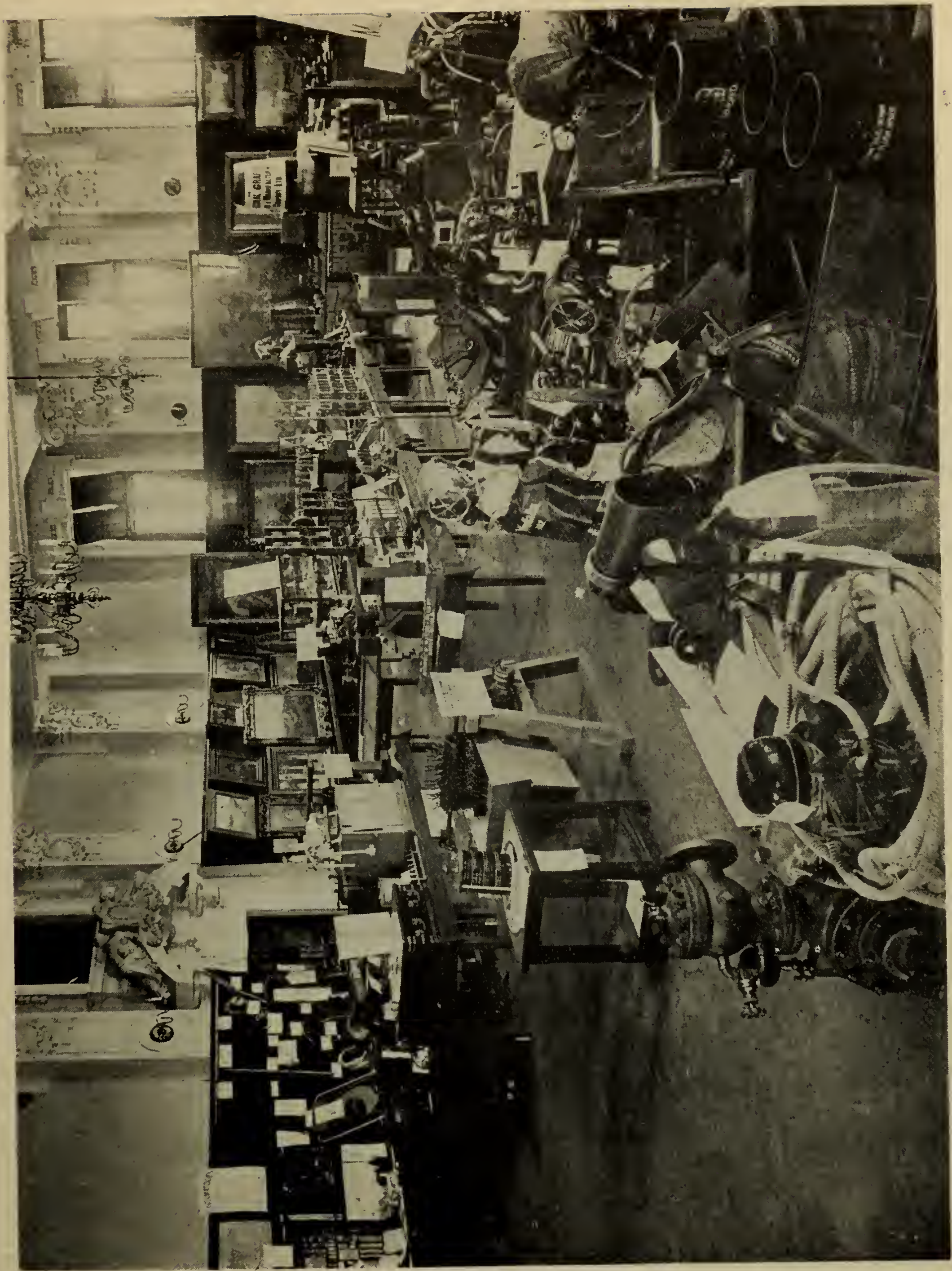
The party rejoined the train about 6.30 p.m., and arrived at Park Station shortly before 10 p.m.

The Mining Exhibition.

The First Exhibition of Chemical, Metallurgical and Mining Apparatus and Appliances, etc., held in Johannesburg, inaugurated by the Council of the Society, was opened in the Assembly Hall of the South African School of Mines and Technology, Johannesburg, Friday afternoon February 9th, 1912, in the presence of a large number of ladies and gentlemen, the latter for the most part representative of the local mining world.

In the regrettable absence through illness of the Chairman of the Exhibition Committee (Mr. Wm. Cullen, Past-President), Mr. Andrew F. Crosse (Past-President), presided. In introducing Mr. H O'K. Webber, the President of the Transvaal Chamber of Mines, Mr. Crosse said that the gathering had been called together to start a new experiment in Johannesburg, namely a Mining Exhibition. The question of holding a Mining Exhibition was brought up before the Chemical, Metallurgical and Mining Society several months ago. The first idea was to have a permanent exhibition in Johannesburg, but for the present that was found impossible, as no suitable building could be obtained. However, half a loaf was better than no bread, and the School of Mines Council had kindly placed the hall at the disposal of the Society until March 2. Mr. William Cullen—the genial manager of the Modderfontein Dynamite Factory—Chairman of the Exhibition Committee, has been, however, unfortunately prevented by illness from addressing the gathering, and no one could regret his absence more than the speaker. The members of the Society had felt that the mining and metallurgical industry should make a move and hold an exhibition of various appliances of interest to those concerned. Mr. Cullen had been specially interested in promoting the exhibition of various rescue appliances, and it was to be regretted that he was not present to explain them. He (the speaker) believed that if the various raw products which were found in South Africa could be exhibited, and the locality which produced them could be mentioned, that manufacturers as well as the farmers would be induced to make many things which had now to be imported from oversea, and amongst those commodities were glass, fire-bricks and crucibles. In a community like this, where so many ingenious men were striving to solve so many problems connected with technical necessities which were continually cropping up, an exhibition of such a kind afforded an excellent opportunity for exhibiting new appliances, or showing models of the same. There were some very interesting models on show, and a full and minute description of them would no doubt be given by the Press. The success of the exhibition would be largely due to the enterprise of the commercial firms who had come forward with their exhibits. The other evening the *Star* had given a very sympathetic account of the objects of the exhibition, and had mentioned that the Government ought to contribute its quota towards the financial expenses of the undertaking. The Society did not consider it necessary to ask the Government for any financial assistance, but the Government mining representatives had assisted by sending some very interesting exhibits. The exhibition was merely the beginning of a very much larger and possibly, a permanent undertaking. Thanks were due to the members of the Exhibition Committee, especially to Mr. H. G. Brickhill for their work in getting such a good display together; and it was to be hoped that the exhibition would be well patronised by the public as well as by mining men. Before sitting down, he mentioned the name of the Secretary, Mr. Fred. Rowland, who had taken infinite trouble to make the exhibition a success. If he had stuck to the proverbial eight hours a day work, said Mr. Crosse, this exhibition would not have been ready for inspection for another week. The Chamber of Mines had been of great assistance pecuniarily, and he had great pleasure in calling upon Mr. Webber, President of the Chamber of Mines, to declare the Exhibition open.

Mr. H. O'K. Webber said that it was a great honour to be present on behalf of the Chamber of Mines to see the Exhibition which had been organised by the Chemical, Metallurgical and Mining Society of South Africa. He thought it did the Society great credit for having started it because an exhibition of such a nature had been a long felt want. The Geological Society had a museum in the same building, and he hoped that that Society would see its way to unite with the Chemical, Metallurgical and Mining Society for the purpose of keeping a permanent museum, so that everybody might go and see the various appliances which were used in mining and the kindred sciences of mining. In Johannesburg, they knew, they had more brains to the square acre than any other place in the world, so he believed! There was much greater scope for the men who had these faculties for invention to exhibit what they had invented, so that everyone might learn from each other and the mining industry in which they were so much wrapped up might benefit. Looking around them they would see exhibits of life-saving apparatus, while there were many other inventions of interest. There were inventions for restoring life from gassing, and respirators which were useful in trying to get rid of the scourge of miners' phthisis. Most of the exhibits were of local origin, although there were some from oversea. They would see among the exhibits labour saving appliances. In South Africa they had in the past been used to the native machine, which was cheap, but all the same it had hindered invention.



General view of the Exhibition looking North-East.

Photo by T. Brittain.

If the mining industry were a white man's one, and he (Mr. Webber) hoped that it would be such one day, then he felt that the stimulus for invention would present itself more quickly than it had. The native was becoming a very expensive machine, and his cost was increasing daily. As his labour became more expensive, more labour saving appliances would have to be invented. Every man would have to set his mind to work to find a means of reducing the costs of production. There were large stores of low grade ores which could be brought into the sphere of production. It was only by the reduction of working costs and specialising in labour-saving appliances that those great stores of mineral wealth could be exploited. There was a great rush upon the mines to-day, and workers had little time in which to study labour-saving appliances. But the heads of the industry wanted the industry to be a white man's one, and that could only be done by inventions. Almost every batch of miners contained somebody who was very ingenious, and amongst the surface workers one would always find someone trying to patent some labour-saving device. The *Government Gazette* showed that every month. The Patents Committee of the Chamber of Mines—of which he was a member—was always willing to aid inventors, and often when the invention was not an invention according to law, the committee helped its designer with suggestions and looked after him. What he (Mr. Webber) thought was wanted was somebody to pick the brains of underground and surface workers—so that the best schemes for reducing working costs could be disclosed. The public, in visiting the Exhibition, would see nearly everything with regard to mining. There were even models of underground workings. Persons who studied those models would be able to get knowledge of what went on underground, but he advised them to go underground and get their knowledge first hand. Everything could not be seen from models, and he wanted the people to be critics from experience and not from hearsay. It was to be hoped that the Exhibition would be an annual one and of great use to the community. Further, he hoped that underground workers would visit it and next year show models of their inventions. He had great pleasure in declaring the Exhibition to be open.

Upon the motion of Mr. H. A. White, Mr. Webber was accorded a hearty vote of thanks for his presence there that day.

THE EXHIBITS.

The Hall of the School of Mines being the only place conveniently available for the Exhibition, space was extremely limited, particularly so when a considerable amount of wall space was occupied by the pictures for the Art Gallery which are exhibited in the Hall pending the building of the Gallery. Many commercial firms were in consequence precluded from exhibiting.

The School of Mines exhibits of Models used in the teaching of mining and metallurgy occupied a long table and were always the centre of many interested visitors. The Government Mines Department lent many interesting examples of "how not to do things," the exhibits being for the most part relics of accidents which have occurred on the mines. A varied collection of mineral specimens found in South Africa was lent by the Geological Survey Department, which was supplemented by many lent by Dr. J. Moir—the Chemist of the Department. Dr. Moir's collection included the specimens of iron ore and coke from Transvaal coals, etc., upon which the Report upon the proposed Iron Industry was written for the Government by Mr. F. W. Harbord; as well as the simple air test devised for CO_2 detection. The General Post Office (Engineering Department) installed a Public Call Telephone and also an exhibit of a small Mine Exchange with underground telephone instruments connected to it. Adjoining Messrs. Jenkins & Co., the agents for Ericsson's Telephones, showed mine telephones disconnected and open; a synchronous electric timepiece (B. P. System Impulse Silent Electric Clocks) with two clock faces, and Webb's patent jockey for mine trucks. Messrs. Orenstein—Arthur Koppel, Ltd. had a large exhibit of Westfalia rescue apparatus, including the Shamrock apparatus for mouth and helmet breathing; Dr. Brat's oxygen reviving apparatus; pure air supply equipment with air pump or bellows; the Shamrock stretcher with oxygen equipment; mine survey instruments, an eight stage centrifugal Sulzer pump; Purser's dust arrester, and air valves, model mine trucks, etc. Messrs. Chas. Butters & Co., Ltd. had a full set of working drawings for the equipment of the Langlaagte Estate and G. M. Co., Ltd. with the Butters' filter plant, and showed a full-sized leaf which had been in use at the Crown Mines plant for over 18 months. Fully laid out on an adjoining table by The British South African Explosives Company was the chemical apparatus used for detecting carbon monoxide in mine air, sampling apparatus for mine gas, electric firing apparatus, the D'Antriche test for the time of detonation of explosives, and Brock's firing stick (to replace the "cheesa" stick) with patent holder. An interesting series of micro-photographs of iron, steel, copper and alloys were shown by Mr. Percy A. Tucker (an Associate of the Society). The Anglo-German Tin Co. had an exhibit of very fine tin specimens from German S. W. Africa, one "nugget" weighing 390 lbs. being generally the centre of many interested visitors. The Messina Copper Co. showed various specimens of copper ores from their mine, while Prof. Stanley had the specimens on view of titaniferous iron ore from the Pretoria district and the pig iron extracted therefrom, a description of his experiments in connection therewith having previously appeared in this *Journal*. A bedstead with patent laths for use in mine hospitals and compounds, patented by Mr. S. A. Binks, was shown by Messrs. Harper and Tanner of Randfontein. Mr. Macnamara of Zuurfontein had many cores from holes drilled by his shot drills, together with specimens of the shot and the bits or cutters used. Various types of rock and hammer drills were exhibited by the Ingersoll-Rand Co., the Denver Rock Drill and Machinery Co., Ltd. (agents for the Waugh drills) and The International Channelling Machines Co., Ltd. who showed the Siskol drill and a model of their coal cutter. During the time the Exhibition was open these three firms also demonstrated their drills in actual working practice on granite blocks in the School of Mines drill testing shed, which demonstrations were very largely attended. The Cannon patent automatic gripping jockey for mine trucks, shown by Auto-Grip, Ltd., was exhibited in model form at work driven by a small motor, and was one of the attractions of the Exhibition. Messrs. Rennert and Lenz occupied a large space with exhibits of Cape asbestos in many forms suitable for mining purposes. Acetylene lamps in various shapes and sizes for underground use, a model Babcock and Wilcox boiler, coal testing apparatus, a small "Sirocco" fan worked by motor, anti-friction metal, etc., and the "Pneumatogen" rescue apparatus, which weighs but 22 lbs. The Stobie respirator was exhibited by the manufacturers showing its evolution from the first one made until that in use to-day. "Proto" rescue apparatus, air masks with bellows for supplying air, oxygen cylinders, the "Handy" first-aid box and oxygen output, the Keiker sleigh stretcher for use in stopes, which enables injured men to be taken up incline or vertical shafts with minimum discomfort, and the "Handy" mine surgical outfit. Of interest to chemists and assayers was the Keller balance, Iler's cupel machine, a prospector's combination crucible-muffle furnace, and a prospector's blow-pipe cabinet, together with various forms of respirators for the use of miners. A mine surgical first aid box containing everything necessary for rendering first aid on mines, specially designed by Dr. L. G. Irvine, was also a feature of this stand.

Messrs Stewarts and Lloyds exhibited samples of their well known piping, pipe joints, valves and fittings, Greenfield and Kennedy's water meters etc., and Hopkinson and Cos., high pressure steam valves.

General View of the Exhibition looking South East. *Photo by T. Brittain.*

Some beautiful silica-ware was to be seen, sent out particularly for the Exhibition by the British Thermal Syndicate, Wallsend on Tyne. The ware, which is named Vitreosil, comprised combustion tubes—glazed and plain, basins, plates, crucibles, capsules, muffles, triangles, trays, and a pipe socket bend, and received much attention from chemists and assayers An experimental Johnson's filter press and various forms of Johnston's patent troughing idlers for conveyor belts, and "Simplex" patent haulage rope rollers were shown by Messrs. J. and R. Niven. Mr. C. F. Wienand, the agent for J. H. Andrew & Co's Toledo steel, exhibited a handsome case of this well-known steel, together with specimens of wire screening and asphaltic solutions. Messrs. Lennon, Ltd., had a large exhibit which included the

In the School of Mines Engineering Department a national suction gas plant, erected by the firm was shown working on Wednesdays and Saturdays.

The laboratory sifting apparatus invented by Mr H. Stadler was shown by Mr. J. H. Oehmen, and a electro-magnetic separator which had been working successfully on the Porges Randfontein mine for seven months was exhibited by Mr. T. Coppin. The Transvaal Graphite Mining and Manufacturing Co., Ltd., had an interesting exhibit of the ore and various uses to which the graphite mined from its property in the Northern Transvaal, could be put. The rough graphite ore was shown and the preparations therefrom, such as graphite flake, paint, lubricants, axle grease, jointing compound and "Murella" graphite boiler composition for removing incrustation from boilers, etc. Messrs. Bellamy and Lambie showed samples of Latch and Batchelor's wire ropes and wire drawing for use in aeroplanes, Henry's patent belting, and North British rubber goods, together with models of Barker's safety hook, Reliance rope capel, France metallic packing and Sell's roller bearings. The well known Brunsviga calculating machine was exhibited by Messrs. Richard Heymann & Co. African Paints, Ltd., had a pretty little stand upon which were samples of the many beautiful paint powders made locally, with samples of putty, soft soap, washing soda crystals and acetic acid. Of use to the metallurgical worker were the goods exhibited by the Leyland and Birmingham Rubber Co., Ltd. On this stand were various screens and screening in I.M.M. standard sizes (Locker's), rubber hoses, valves, vat door rings and conveyor belting, filter paper and cloths, amalgam scrapers, enamelled ware, brushware and plate cover locks. Sections of electric cables—one show case exhibiting samples of the shaft and surface cables suplied to the Robinson G. M. Co., Ltd. under the Rand Mines Electrification Scheme, were displayed by Henley's (S.A.) Telegraph Works Co., who also showed a new terminal box and a specimen of cable as supplied to the Marconi Co., which had been tested to 130,000 volts. Wolf's acetylene and safety lamps were among the principal items on Messrs. Geo. Cradock and Co. Ltd's., stand, together with samples of Bradford's patent cocks and valves, Kinghorn's patent metallic valves, Cradock's patent haulage clips and steel wire ropes, steam cylinder lubricators. The model of Humble's safety detaching hook created much attention. In another part of the Hall a large case of samples of the steel wire ropes for which they are so well known, was shown by the same firm. Adjoining this was Dickers patent valve with simple reseating device, shown by Messrs H. Incledon and Co. Ltd., Usher's patent candle save was displayed in a novel manner by the Inventor, while Messrs. Stephens and Hoar exhibited the L.S. Pierce amalgamator which is reported to be doing good work experimentally upon several of the mines, the W.S. Tyler ton-cap screening, the Western automatic lubricating valve, and the Bowser oil pump and tank. The African Share Agency Ltd. were responsible for a large attractive show case containing useful articles for presentation, jewellery, ornaments, etc., manufactured from Verdite, a beautiful green stone from the Barberton District. There were also included specimens of the gold schist from the same Co's. gold mine together with samples of talc powder from the slime, with which soap, toilet powders etc., were manufactured.

General View of the Exhibition looking West. *Photo by T. Brittain.*

INDIVIDUAL EXHIBITS.

Turning to the exhibits sent in by individual workers and inventors on the Rand there were many of great interest, some of which have been successfully applied, others of which the inventors hoped would be of use and benefit.

Mr. Geo. Buchanan: A patent tool holder for zinc lathes; Mr. B. V. Blundun: Patent grading or sifting machine; Mr. E. Harrison: Improvement for ventilating drives, raises, etc., by pumping in good air and clearing out smoke, dust and fumes; Mr. Walter Philip: A safety device for reversible winding and hauling plants; Mr. J. W. Scorer: A safety appliance for mills for facilitating the placing of wedges used in the connecting of the shoes to the heads of stamps; Mr. Hugh McLean: An improved form of coupling—for coupling up broken cam shafts in stamp mills; Mr. C. F. Bayly: A nozzle for regulating the underflow of pulp from classifying and dewatering cones, etc.; Mr. S. Fisher: The Guin machine jet—a spray device for attaching to the bars of machine drills; Mr. W. M. Butler: An amalgam trap for use in launders, etc.; Mr. James S. Reah: (1) A device for taking drills down vertical shafts. (2) A new ventilation door for drives; Mr. Harry Rusden: (1) A water jacket assay mould, and (2) a model assay furnace for use with coal; Mr. A. L. Thorne: The Thorne patent vacuum dust abstractor; Mr. W. H. Jane: A thrust collar for cam shaft; Mr. J. H. Dinwoodie: A patent silicosis respirator; Mr. Donald McLachlan: A patent dust allayer for use with rock drills (a spray device); Mr. O. Tonnesen: (1) Patent slime sampler. (2) The "Stereometer"—a mine survey instrument.* (3) The Simplex stope measuring instrument, and (4) a beacon apparatus consisting of stands with rods and yoke; Mr. Andrew Crosse: A magnetic separator for laboratory,* with photographs of a patent fine grinding machine; Messrs. J. W. Eykelesboom and G. A. Coles - A dust arresting apparatus to be attached to rock drills; Mr. E. Budd: Specimens of magnesite from the Kaapmuiden Magnesite Mine; Mr. Joseph Michell: (1) A device for catching dust when drilling dry holes in drives or bore-holes. (2) A pick with detachable points, and (3) a sketch of a bar for stoping, with detachable parts.

Other interesting exhibits were the Scoular patent concentrator (a concentrating machine used on the Rand many years ago) and a polished specimen of banket ore from the Village Main Reef shaft, being a portion of the South Reef where penetrated by the first bore-hole put down on the Witwatersrand in 1889, both shown by Mr. J. S. Curtis. Side by side were models of a 10-stamp battery unit as erected at the Modder "B" mine shown by the Central Mining and Investment Corporation, Ltd., and a 4 stamp Nissen battery, shown by Messrs. Head, Wrightson & Co., Ltd. A section of the apparatus for the assay of solutions by electrolysis* was exhibited by Mr. C. Crichton for the Kleinfontein Group of Mines, and the Manager of the Robinson Deep G. M. Co. (Mr. J. J. Wessels) displayed models of sprays for drives, and specimens of sharpened drills as used on that property. Mr. M. H. Coombe, the Manager of the Government Gold Mining Areas (State Mines) showed a model of a seven compartment shaft. The "Synchron" life saving or artificial respiration apparatus was exhibited by Mr. R. A. Barry of the Nourse Mines, Ltd.

The Exhibition, to which admission was free, remained open daily (except Sundays) from 10 a.m. till 10 p.m., from Friday the 9th February until Saturday the 2nd March. During that period it is estimated that from 15,000 to 17,000 visitors passed through the Exhibition, which on some days was somewhat uncomfortably crowded. Demonstrations of the various rescue appliances were given by the various firms representing the manufacturers at intervals during time the Exhibition was open which were followed with much interest.

Altogether the Council of the Society are extremely gratified by the great success achieved by its first Exhibition. The Society's grateful thanks are due to the Council of the School of Mines for so kindly placing the Assembly Hall at their disposal, without which the Exhibition could not have been held; to Professors Orr and Stanley and other members of the Staff of the School for much valuable assistance; and to the Press (and particularly to the *Star*) for the great help and ready assistance in publishing notices and bringing the Exhibition so prominently to the notice of the public. The Council is also under a debt of gratitude to its Exhibition Committee for their arduous labours in the organising of the Exhibition. The Committee consisted of Mr. Wm. Cullen (Chairman), Messrs. R. Allen, H. G. Brickhill, Andrew F. Crosse, Tom Johnson, Dr. J. Moir, Prof. G. H. Stanley and A. Whitby, with the Secretary, Mr. Fred. Rowland.

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*Previously referred to in this *Journal*.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

On the Behaviour of Fused Silica Ware.—"Vitreous silica may be obtained in two main types:

(*a*) As a transparent glass almost or quite free from bubbles. This is prepared by a difficult and lengthy process, but when finished is as clear as good glass.

(*b*) As an opaque or translucent glass containing bubbles. The difficulty of manufacture is in this case not so great, and very much larger apparatus can be made. In most cases, unless transparency be an object, it may be used for the same purposes as the transparent material.

An abstract of a table (see p. 160) gives the value of *f* in kilogrammes per square centimetre for most of the tests made. In most cases the probable error is from 10 to 20 units except in the case of silica 'T,' where it is somewhat larger.

On comparing the relative strengths of different kinds of silica as shown in this table, it is at once noticeable that the transparent material is between two and three times as strong as the others at all temperatures.

Mean Furnace Temperature.	Period of Heating.	Strength in $\frac{\text{Kilos}}{\text{Cm.}_2}$			
		'U' (Unglazed).	'G' (Glazed)	'S' (Satin-like.	'T' (Transparent).
Unheated	—	342	382	281	902
1,124° C.	8 hours	376	310	319	818
1,122°	$3\frac{1}{2}+4\frac{1}{2}$ hours	358	337	307	750
1,188°	4 hours	425	309	285	897
1,186°	8 ,,	369	311	277	802
1,275°	4 ,,	257	270	211	745
1,353°	4 ,,	218	186	188	339

2. *Microscopical Examination. — General Conclusions.*—1. That except for a few small grains of quartz and an occasional grain of carborundum from grinding no foreign nuclei were observed, and that the former were only occasionally starting points for crystallisation.

2. That at the lower temperatures to which the substance was heated, crystal growth was confined to the outer surface and the walls of bubbles, especially of those open at the ends and near the surface of the section.

3. That sometimes at 1,275°, and nearly always at 1,353°, crystals were formed in the ground mass and a microcrystalline structure developed in some areas. It is possible that this may be partly attributable to some small traces of impurity in a state of solid solution. Silica 'T' did not show this apart from bubbles.

4. That the crystals formed consisted of tridymite.

5. That they were very minute, but increase slightly in size to about 1,188°, and then usually become shattered at higher temperatures.

6. That the changes in mechanical strength very closely follow the crystalline changes.

7. That the formation of the crystals seems to be accelerated by contact with the air or in parts of the specimen which may become permeable to air, the inside of the specimens usually being attacked last.

8. That above the transition point of vitreous silica to tridymite the change can probably not be prevented entirely.

It may be mentioned that as far as the work was on the same lines as that of Day and Shepherd on the lime-silica series of minerals, it was confirmatory of their results.

3. *Relative Coefficients of Expansion of the Opaque and Transparent Varieties of Fused Silica.*--Experiments lead to the main conclusions, which are as follows :—

(1) That the differences between the coefficients of expansion of transparent silica and the opaque unglazed material are very small.

(2) That the opaque material has a slightly greater expansion than the transparent up to about 500° C., and that above that temperature the transparent silica expands most.

(3) That in both groups of experiments the expansion of one or both materials lags behind the temperature.

The transparent silica has the greater lag in experiments carried to the higher temperature (above 500° C.), but in one experiment where the highest temperature was only 360° the reverse was found to be the case.

(4) That on re-cooling to atmospheric temperature the lengths are usually not quite the same as before heating, and that a slow change of length takes place in one or both materials for some time afterwards, though from practical points of view this is too small to be of any moment.

It may be taken that so far as the coefficient of expansion is concerned the differences between the two classes of silica are for practical purposes negligible, though from the theoretical standpoint and that of the more exact work required in connection with standards of length they may be of more importance and worthy of further study."—A. BLACKIE, B.A.—*Transactions of the Faraday Society*, Nov., 1911, p. 158. (H. A. W.)

IMPROVED METHOD OF FORMING A HALOGEN CYANIDE.—"In cyanidation it has been found advisable in some cases to make use of bromocyanide, or similar halogen compound. The usual method of forming preparing this compound has been by mixing sulphuric acid, potassium cyanide and an alkali bromide and bromate. The mixture usually is made only when it is needed, and the procedure is one requiring care. It is the aim of Dr. Hans Foersterling, of the Roessler and Hasslacher Chemical Company, of Perth Amboy, N.J., to provide a better means of forming halogen cyanides by melting an alkali halogen with an alkali cyanide in the proportion of one molecule each and permitting them to solidify. No decomposition takes place in this process and the double salt is used instead of making the mixture above referred to. In order to liberate bromocyanide, for example, from a mixture of sodium bromide and sodium cyanide, an oxidising agent such as an acid solution of hydrogen peroxide is used. The reaction follows:

$$NaBr + NaCN + H_2O_2 + H_2SO_4 = BrCN + Na_2SO_4 + 2H_2O.$$

By this method it is possible to provide cyanide mills with a correctly proportioned mixture ready for use when dissolved and mixed with an oxidiser. (999,215, Aug. 1, 1911.)"—*Metallurgical and Chemical Engineering*, Nov., 1911, p. 601. (T. F. A.)

WEIGHT OF A NORMAL LITRE OF HCl.—"The whole of our results on the density of this gas may be summarised as follows :—

Series	Source of Gas.	Number of Experiments.	Volume of Bulb.	Mean Weight of a Normal Litre.
				grams.
I.	$NaCl$ and H_2SO_4	4	465·856	1·63907
II.	$SiCl_4$ and H_2O	7	465·856	1·63914
III.	NH_4Cl and H_2SO_4	10	457·420	1·63920
IV.	NH_4Cl and H_3PO_4	10	551·358	1·63915

The gas was prepared from four different sources. In Series I., II. and III. it was dried by phosphoric anhydride, in Series IV. by anhydrous aluminium chloride. The gas was fractionated many times and in a variety of ways, and it was also sublimed at low pressure without any detectable alteration of density. The gas itself was absorbed completely by solid caustic potash, and the unabsorbed residue remaining after treatment with boiled aqueous caustic potash was negligibly small. The density bulb in Series I., II. and IV. was of glass, in Series III. a fused silica one was used. In all cases the gas was weighed condensed on charcoal in a bulb of small volume, but in Series IV. the method was varied by displacing the gas from the density bulb at constant pressure by means of mercury, and by using a different set of weights and a different manometer scale. In all thirty-one determinations were made, and the arithmetic mean gives the value $1·63915 \pm 0·00004$ gm. as the weight of a normal litre of hydrogen chloride.

This result does not modify our previous conclusions, and we wish to express our belief that the most probable value for atomic weight of chlorine yielded by physico-chemical evidence is 35·460, a value which is identical with that adopted by the International Committee for Atomic Weights in their report for 1910."—F. P. BURT, B.Sc. and R. W. WHYTLAW GRAY, Ph.D.—*Transactions of the Faraday Society*, Nov., 1911, p. 40. (H. A. W.)

MERCURY SURFACE AND Na_2S.—"Our experiments, subject to the reservations we have made, may be summarised as follows :—

The effects of small quantities of Na_2S upon the potential difference between mercury and dilute solutions of various electrolytes have been studied and an attempt has been made to analyse and explain them.

The rate of change of potential difference resulting from addition of Na_2S is at first roughly proportional to the amount of Na_2S added, but suddenly increases at a critical concentration lying near 0·0004 equivalents Na_2S per litre in solutions containing about 0·0011 equivalents of O per litre.

This effect is temporary and almost disappears within twenty-four hours. It is succeeded by a similar effect, permanent so long as atmospheric oxygen is excluded, which occurs in the solution containing initially about 0·00055 equivalents Na_2S per litre.

The second effect is due to the conversion of the Na_2S into $Na_2S_2O_3$. by interaction with the oxygen contained in the solution. The amount per litre of Na_2S required to produce the effect is thus a measure of the amount per litre of the dissolved oxygen. A very slight excess is sufficient to alter the potential difference by more than half a volt.

The temporary effect is apparently due to selective adsorption of Na_2S at the mercury surface. Such adsorption, besides accounting for this effect, would explain the striking variations in the rate of tarnishing of the mercury in the different solutions. It would also be in accord with the time effects first observed in all the solutions and with the latter reversal in sign of the time effect, in solutions containing more than 0·00055 equivalents Na_2S per litre.

The way in which the tarnishing progresses in the different cases illustrates incidentally the origin of phenomena analogous to the 'oxidation tints' of tempered steel."—S. W. J. Smith, M.A., and W. F. HIGGINS, B.Sc.—*Transactions of the Faraday Society*, Nov., 1911, p. 77. (H. A. W.)

PLATINUM IN COINS.—"Just look accurately at your 20 franc pieces: some of them are worth 40 francs because they are counterfeited. They were made all nearly in the last years of the second empire. At this time platinum was not nearly so valuable as gold; it was just an ordinary metal, and you could buy a whole kilo for some hundred francs. And the coiners of false money of this time, who did not foresee that they would enrich a later generation of decent people by their criminal activity, thought to deceive the world by the manufacture of 20 franc pieces from platinum. Now this metal is dearer than ever before; just now it costs 7,500 francs per kilo, and perhaps it may soon go up to 8,000 francs. And in every one of these base coins there are 6 gm. of platinum!"—(Communicated by Dr. GARNY.)—*Le Figaro.* (H. A. W.)

DETERMINATION OF COPPER.—"The modification of the iodide method described consists in the destruction of the nitrous acid without boiling. This is accomplished by the addition of a small amount of sodium hypochlorite. The addition of sodium hypochlorite to a nitric acid solution produces hypochlorous acid. The interaction of hypochlorous acid and nitrous acid results in the oxidation of the nitrous acid and the formation of hydrochloric acid, and the reaction between hypochlorous and hydrochloric acid results in the destruction of the hypochlorous acid and the formation of free chlorine and water. As the solution of sodium hypochlorite contains small amounts of chlorides, hydrochloric acid will always be present when the solution is acidified, thus insuring the destruction of the hypochlorous acid and the formation of free chlorine. The effect of adding sodium hypochlorite to the solution is the complete destruction of the nitrous acid and the formation of free chlorine.

To remove the free chlorine in solution some compound must be added which will take up the chlorine, but will not affect subsequent operations. Such a compound is found in phenol. Under the conditions of the determination, phenol will add chlorine directly to the benzene ring, but is not affected by iodine or any of the other compounds in the solution. Chlorophenol not being ionised removes all traces of free chlorine. This modification of the method greatly reduces the time and attention required for a determination, and, in addition. the copper solution is prepared in such a way that iodine can be liberated by copper alone.

In the determination, the copper, copper oxide, or sulphide is dissolved in nitric acid. After the addition of the sodium hypochlorite and phenol, which requires but a moment, the solution is made slightly alkaline with sodium hydroxide, and is then made acid with acetic acid, when the solution is ready for titration."—E. C. KENDALL, *Mining and Scientific Press*, Dec. 30, 1911, p. 838. (H. A. W.)

ASSAY METHOD FOR TUNGSTEN.—"When tungsten is in solution in the condition of tungstate it is possible to precipitate and estimate the tungsten in the form of the trioxide WO_3. To effect this, all that is needed is to render the solution acidic by means of hydrochloric or nitric or even sulphuric acid.

Tungstic acid is soluble to some slight extent in mineral acids. On that account it is advised in the usual works on analytical chemistry to render the tungstic acid insoluble by repeated evaporation on a hot water-bath, of the solution to which an excess of acid has been added, after which the dry residue is warmed for some length of time at a temperature of 120° C.

To avoid these operations, which are rather tedious and time-consuming, the writer recommends for study a method of determining tungsten based on the precipitation of tungstic acid by an excess of a solution of freshly prepared stannous chloride. The stannous chloride precipitates the tungsten from solution in the form of the tungsten oxide W_2O_5. It is well known that this reaction is quite sensitive.

In order to assure himself to what extent precipitation is complete, and also that the oxide be not reduced during washing, the following experiments were tried by the writer:—

Two gm. of absolutely pure tungstic acid was dissolved in just enough concentrated ammonia to dissolve that amount of acid, and the volume of this solution was brought up to 1 litre. Caustic soda or potash might have been chosen just as well, the choice of alkali having no effect on the precision of the operations.

For each test determination 50 cc. of this solution were taken, containing therefore 0·1 gm. of tungstic acid. To precipitate the tungsten oxide from this solution the author added 20 cc. of a solution of stannous chloride. The latter was so concentrated as to contain 50 gm. of crystal salts per 200 cc. of concentrated hydrochloric acid. After boiling the mixture for one or two minutes the precipitate was permitted to settle.

The precipitate was then washed in warm water. Since the oxide of tungsten is deposited very readily it is possible to wash the precipitate several times within a few minutes without having the water cool sufficiently to float the precipitate. Under these conditions the precipitate remains flocculent, and no cloudiness is produced in the water-bath. The oxide thus precipitated was calcined and weighed as the trioxide. The results indicated that this method was susceptible of an unusually high degree accuracy.

In order to be sure that in the calcined residues measured in the above tests no tin was carried along, the following analyses were made:—Several of the residues were collected and heated in a current of hydrogen. The production was subjected to hydrochloric acid and filtered. A current of sulphuretted hydrogen gas was passed through the filtrate. This reaction gave no precipitate. Hence no tin was carried along in the precipitation of tungsten trioxide.

The author also tried some experiments, the aim of which was to ascertain whether the presence of iron affects the accuracy of the method. With this end in view there were added to a couple of tested samples 10 to 20 cc. of a solution of ferric chloride, half normal. The results showed that iron produces no effect on the accuracy of the method."—B. H. Divani, *The Chemical Engineer*, xiv., No. 4.—*Chemical News*, Vol. 105, 2723, p. 56. (J. G.)

Determination of Sulphur in Pyrite.—"In Dennstedt's method of sulphur determination by burning the pyrite in a stream of oxygen, passing the gases formed over hot platinum, and absorbing the sulphur trioxide in heated sodium carbonate contained in boats in the forward portion of the combustion tube, there is always some sulphur trioxide retained by the Jena glass tube, which must in consequence be washed out after each operation, and the washings added to the solution of the contents of the boats for precipitation by barium chloride. The author finds that this troublesome process can be entirely avoided by using a quartz combustion tube, in which case the amount of sulphuric acid retained, if any, is inappreciable. The process of the combustion can be quite well observed through the translucent tube. The sodium carbonate is dissolved in water, bromine water added, then hydrochloric acid to neutrality. The pyrite residue is dissolved in 2 cc. of warm concentrated hydrochloric acid, diluted, ammonia added to precipitate the iron, filtered, and the filtrate added to the main solution. This is then made up to 400 cc., and after addition of 1 cc. of concentrated hydrochloric acid, heated to boiling, and precipitated by 10% barium chloride solution added all at once, in quantity about double that required for the sulphur present. The results agree satisfactorily with, but are usually slightly higher than, those obtained by Lunge's method of dissolving in nitro-hydrochloric acid, precipitating the iron, and precipitating the filtrate with barium chloride."—A. Heczko, *Zeitschrift für Analytische Chemie*, Dec., 1911, pp. 748-753. (G.F.A.)

Origin and Composition of Coal.—"Vivian B. Lewes delivered a series of Cantor lectures before the Royal Society of Arts during December on the 'Carbonisation of Coal.' The study of the effective destructive distillation of coal involves an inquiry into its nature, composition, and origin, and he therefore reviewed this part of the subject at considerable length. The origin of coal is not only of interest to the scientific geologist, but the disquisitions in connection with it are of value as above mentioned to the gas and coke producer and also to the investigators of coal-dust explosions. In view of the recent discussion on the origin of oil, a perusal of Mr. Lewes's paper will prove of interest to many of our readers. Though it is universally held that coal is the remains of plants, it is difficult to determine which particular portion of the vegetable kingdom provided the basis for the deposit, and which part of the individual vegetable growths were preserved; also what were the chemical reactions and physical forces that produced the effect of disintergration and subsequent preservation and carbonisation. It is generally held, however, that the chief constituents were: (1) the humus derived by the degradation of cellulose and lignite, and (2) the resinous and gummy substances produced by plant life. It is conceded that coal is a variable product and that therefore the original constitution of the matters forming it varied within wide limits. Until recently its analysis in the true sense of the word has been impossible, and has been confined to the determination of fixed carbon, volatile matter, and ash, or to the ultimate analysis of the elemental constituents. During the last few years, two new methods of examination have been indicated, one being the discovery by Bedson that pyridine will dissolve part of the coal, and the other the system introduced by Burgess and Wheeler who have closely studied the principles of fractional distillation. The two latter investigators have found that at temperatures below 450° C. the gases evolved consist largely of the paraffin series, and that when the temperature of distillation is about 700° C. the gases contain a large preponderance of hydrogen. The paraffin gases are considered as being derived from that part of the coal formed from resinous substances, and the hydrogen from the cellulose degradation derivatives. The behaviour of various coals under carbonisation action is irregular and has been much of a mystery, but these new ideas go a long way to explain some of the phenomena. It is believed that the portions of the coal soluble in pyridine are those that yield the paraffins in Burgess and Wheeler's reactions. Mr. Lewes does not agree with all their deductions, so this paper, quoted by him, read recently before the Chemical Society, should also be studied."—*Mining Magazine*,—Jan. 1912, p. 65. (A. R.)

The Behaviour of Certain Alloys when Heated in Vacuo.—"It was observed during the course of some experiments conducted a year ago, that on melting brass in vacuo, the whole of the zinc volatilises, leaving the copper. This separation is quantitative, if the heating is not too prolonged and the temperature not above 1,200° C.

The behaviour of other copper-zinc alloys when similarly heated was, therefore, investigated. A sample of 'poisoned' brass—that is, brass containing iron, lead, tin, arsenic, and other impurities—was heated in a porcelain tube in vacuo at 1,200° C., and the residue examined. All the zinc, lead, and arsenic, and a little of the tin, volatilised, leaving a residue of copper, iron, and most of the tin.

Larger scale experiments are also described, in which 70/30 brass was melted in a crucible, transferred to a large and well lagged desiccator, and the whole quickly exhausted. The residue in the crucible was cooled and analysed, and the loss in zinc recorded. As a result of these experiments, it is suggested that heating in vacuo might be advantageously applied for the refining of crude copper, brass scrap, etc.

'Hard' zinc may be refined by heating in vacuo to 500° C.—that is, to a scarely visible red heat. Hard zinc is the residue from galvanising baths, and contains about 5% of iron. At higher temperatures some iron is carried over with the zinc, but at about 500° C. all the iron remains in the residue. As the temperature necessary is so low, and as there is no loss of zinc by oxidation, there is reason to believe such a process would be commercially successful.

Some preliminary experiments on the equilibrium between zinc vapour copper, and copper alloys, are also described.

Zinc itself volatilises easily at 375° C. in vacuo. A 60/40 brass readily loses zinc at 520° C., and a 70/30 brass at 550° C. On heating zinc with copper in vacuo, the metals not being in contact, the copper rapidly turns yellow at 380° C., whilst the zinc is still solid, enough zinc volatilising to form a surface layer of brass. Zinc distils readily in glass vessels in vacuo, the vapour being colourless and transparent. The zinc condenses in globules, having the appearance of mercury." — Prof. T. TURNER. — *Page's Weekly*, Jan. 19, 1912, Vol. xx., p. 99. (J. E. T.)

RAPID COLOURIMETRIC DETERMINATION OF MANGANESE IN IRON AND STEEL.—"When the manganese content of the iron or steel is between 0·02% and 1·50%, 0·5 gm. of the sample is dissolved in 20 cc. of nitric acid (1 : 1), in a 100 cc. flask, and the solution heated till the red fumes have disappeared. 20 cc. of silver nitrate solution (6 gm. per litre) are then added, and when the solution has become only lukewarm, about 1 gm. of ammonium persulphate is added and the solution made up to 100 cc. with water free from hydrochloric acid. The contents are then well shaken. The maximum colour is developed in a very short time. 5 cc. of the solution are transferred to an Eggertz tube and comparison made with a standard. The standard may be prepared by treating a steel of known manganese content in the same manner as the sample, or 10 cc. of a manganese solution containing 0·1 gm. of manganese per litre may be taken and treated like the steel solution with silver nitrate and ammonium persulphate. The manganese solution is made by dissolving a pure oxide of manganese in nitric acid. Higher percentages of manganese may be estimated in the same way by taking smaller weights of the sample, using a stronger solution of silver nitrate, and suitably diluting the final solution. Silica and graphite, if present, may be allowed to settle, or a portion of the solution may be filtered through ignited asbestos, or guncotton. If the solution is too hot when the ammonium persulphate is added, manganic hydroxide may separate. The solution should then be transferred to a Jena flask, boiled, cooled, and treated with a few drops of hydrogen peroxide. The clear solution is then boiled to destroy excess of hydrogen peroxide. again cooled, and the determination continued in the ordinary manner by the fresh addition of ammonium persulphate. The determination succeeds equally well in sulphuric or nitro-sulphuric acid solutions. Large quantities of nickel, chromium or copper do not interfere. The time required for a determination does not exceed 15 minutes."—STANICHITCH, *Rev. Mét.*, 1911, *8*, 891·2.—*Journal of the Society of Chemical Industry*, Vol. 31, p. 75. (J. G.)

ANALYSIS OF FERRO-URANIUM.—"Aluminium, which is a frequent impurity in ferro-uranium, should be determined by fusing the ammonium carbonate precipitate with sodium peroxide, not the alloy as stated in error in the former communication (*J.S.C.I.*, 1911, 132). Rose's method of separating iron from uranium by means of ammonium sulphide in the presence of ammonium carbonate gives very accurate results :—The solution of 0·5 gm. to 1 gm. of the alloy in *aqua regia* is diluted, completely oxidised, and treated with a large excess of solid ammonium carbonate. A small excess of ammonium sulphide is next added, and the contents of the flask shaken, made up to 500 cc., and filtered. An aliquot part (250 cc.) of the filtrate is boiled for a long time to precipitate the uranium, which is filtered off after the addition of a few drops of ammonia solution. Or the liquid may be acidified with nitric acid and boiled, and the uranium precipitated by means of ammonia. The precipitate is washed with dilute ammonium nitrate solution, since it is somewhat soluble in water. Any black glittering particles undissolved by *aqua regia* consist of silicon carbide. When ignited in a platinum crucible they leave a residue of silica."—W. TRAUTMAN, *Z. angew. Cnem.*, 1912, *25*, 19.—*Journal of the Society of Chemical Industry*, Vol. 31, p. 76. (J. G.)

DETERMINATION OF NICOTINE IN MANUFACTURED TOBACCO AND IN GREEN TOBACCO LEAVES.*—"The weighed material, cut in small pieces, is damped with boiling water, and set aside for 24 hours in a closed flask. A considerable excess of freshly-made milk of lime is then added ; the flask is again closed and set aside for a further twenty-four hours, with occasional agitation, after which the nicotine is distilled over in a current of steam. The distillate is neutralised with dilute sulphuric acid, and evaporated on the water-bath, protected from light. The concentrated solution is transferred to a separator, rendered alkaline with a strong solution of potassium hydroxide, and shaken out with ether. The ether extract is evaporated on the water-bath until no more vapour of ammonia is given off, as indicated by Nessler's test-paper ; then the rest of the ether is allowed to evaporate spontaneously. The oily residue thus left, which is nicotine, is dissolved in water and titrated with $N/10$ sulphuric acid solution."—R. MELLETT, *Schweiz, Woch. Chem. Pharm.*, 1911, *49*, 117 ; *Ann. Chim. analyt*, 1911, *16*, 233.—*Journal of the Society of Chemical Industry*, Vol. 31, p.p. 89-90. (J. G.)

DETECTION, SEPARATION AND DETERMINATION OF ARSENIC AND ANTIMONY.—"The author states that the method of Blattner and Brasseur (*J.S.C.I.*, 1904, 338) for the determination of small quantities of arsenic in hydrochloric and sulphuric acids does not give accurate results. In the case of concentrated hydrochloric acid, the loss of arsenic may amount to 2%. Correct results are obtained with sulphuric acid of 45° B. even in the presence of hydrochloric acid, provided that the amount of the latter does not exceed one-third of that of the sulphuric acid. The washing of the arsenic tri-iodide with hydrochloric acid containing potassium iodide is not to be recommended, as iodine may be thereby liberated ; and instead of the 5 cc. prescribed by Blattner and Brasseur, the author uses 2·5 cc. or less of the potassium iodide solution. For the determination of arsenic in hydrochloric acid, one part of the latter is mixed with two parts of sulphuric acid of 45° B., whilst in the case of sulphuric acid, if lead and tin be absent, the arsenic is precipitated as iodide without addition of hydrochloric acid. Commercial sulphuric acid is brought to a concentration corresponding to 45° B. and mixed with one-third of its quantity of hydrochloric acid before precipitating the arsenic ; if nitrous compounds be present, these must be removed by addition of urea. For washing the precipitate, a mixture of two parts of sulphuric acid of 45° B. and one part of hydrochloric acid is used. In this modified form the method is said to give good results. Antimony tri-iodide is soluble in concentrated hydrochloric acid, and hence can be determined by this method only in sulphuric acid. For the determination of arsenic and antimony when present together, the substance is dissolved in sulphuric acid of 50° B., and in an aliquot portion of the solution, the arsenic and antimony are precipitated as iodides, and the precipitate titrated as described previously. In another aliquot portion of the solution, after addition of one-third of its quantity of hydrochloric acid, the arsenic alone is determined, and then the amount of antimony is obtained by difference."—G. BRESSANIN, *Boll. Chim. Farm.*, 1911, *50*, 691-694; *Chem. Zentr.*, 1911, *2*, 1964-1965.—*Journal of the Society of Chemical Industry*, Vol. 31, p. 93. (J. G.)

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* See this *Journal*, February., 1912, p. 340.

METALLURGY.

RE-PRECIPITATION.—"The following notes may be of interest. Some time ago I treated ore from two old dumps in this district. No. 1 dump has small pine-trees growing out of it and is the refuse from the rock mined by the 'ancients' when the method of mining was to light a huge fire against the ore, and when the face was sufficiently hot to apply water. The ore thus produced was broken by hammers, the richest being saved. The residue is the dump, which contains 0·2 oz. gold and 9 oz. silver per ton. There are many roots, small pieces of decaying wood, and charcoal as well. The ore is a friable iron-stained quartz.

I had many misgivings as to being able to treat it successfully, as I was afraid of re-precipitation, but had ten tons brought to the mill as a test. This was reduced so that 90% would pass 150 mesh, agitated with liquor of strength 0·1% free ond 0·25% combined cyanide, and protective alkali 0·06% in terms of NaOH. The proportion of solid to solution was 1 : 3. After 24, 36, and 48 hours air-agitation, samples were taken of the pulp: these were filtered, washed, and assayed for gold and silver, the filtrate being tested for free and total cyanide and protective alkali. The following was the result:

Time of agitation in hours.	Free KCN.	Total Cyanide.	Protective Alkali NaOH.	Gold Oz.	Silver Oz.
24	0·07	0·12	0·06	0·05	4·50
36	0·05	0·10	0·03	0·01	2·35
48	0·03	0·08	0·02	0·01	2·00

The total time of agitation was 52 hours. The residues contained 0·01 oz. gold and 2·15 oz. silver. The pulp was filter-pressed and water-washed.

No. 2 dump is also an old one, produced similarly to the former, but consists of a different class of ore, being a pyritic quartzite oxidised by the weather and also containing vegetable matter and charcoal. 17 tons containing 0·35 oz. gold and 12 77 oz. silver per ton were brought to the mill and given a treatment similar to the former. The only difference was that one pound of lead acetate was added at the time of crushing and lime equal to 4 lb. per ton was added, after 24 hours' agitation (this in addition to lime added at time of crushing) to the latter.

Time of agitation in hours.	Free KCN.	Total Cyanide.	Protective Alkali NaOH.	Gold Oz.	Silver Oz.
24	0·05	0·13	0·005	0·25	5·21
36	0·03	0·10	0·02	0·10	3·50
48	0·02	0·08	0·01	0·10	2·90
60	0·05	0·16	0·03	0·06	1·90

The strength of the solution was made up to 0·1% free and 0·25% total cyanide after 50 hours agitation, the protective alkali strength being raised at the same time. The agitation was continued for 72 hours, the residue containing 0·025 oz. gold and 1·05 oz. silver.

From 5·2 kilograms of the residue of No. 1 half an assay-ton (14·583 gm.) of fine charcoal and splinters of wood was obtained; this assayed 0·01 oz. gold and 2·50 oz. silver per ton.

The precipitation of the precious metals in the zinc-boxes was 99%.

From these two tests I am led to believe that if free cyanide is present with protective alkali the chances of re-precipitation are small.

A few years ago I was cyanide assistant at the Ashanti Goldfields and in leaching the current tailing, if care was not taken, re-precipitation would take place in the vats. The strange thing about the plant was the action in the zinc-boxes. If these were not cleaned-up once a week the precipitated gold would re-dissolve. The ore was complex and subsequently it was crushed dry, roasted, and cyanided. But the trouble in precipitation continued and was only cured by keeping the solution entering the boxes at a strength of 0·2 to 0·25% KCN."—D. F. FOSTER, *Mining Magazine*, Jan. 1912, p. 54. (A. R.)

PREMATURE PRECIPITATION OF GOLD IN CYANIDING.—"In the nomenclature of the cyanide process, the term 're-precipitation' is used to indicate the precipitation of the precious metals in the leaching and agitation tanks, which I shall hereafter designate under the general term, 'treatment tanks.' The term 're-precipitation' is very misleading for the reason that it implies a previous precipitation followed by a 're-solution' and 're-precipitation.' The idea which it is desired to convey by the use of the term is that the metals have been accidentally precipitated in the ore before the solution has been drawn off and passed through the precipitation boxes. The term 'premature precipitation' is one which more accurately describes the phenomenon, and as a precedent for its future use, I shall make use of it in that sense in this discussion.

Gold occurs in two different forms—the ordinary yellow, so-called crystalline variety, and the black amorphous kind obtained by precipitation. Of the former we have a very complete knowledge regarding both its chemical and physical properties, but of the latter our information is so meagre that we cannot even say for a certainty that it is pure gold. The idea has been advanced that amorphous gold is a mixture of the metal with one of its oxides, but the evidence is very incomplete. The fact that it can be converted into the yellow variety by the application of heat makes the 'metallic gold-oxide' hypothesis rather untenable. It is to be regretted that our knowledge regarding the composition and properties of amorphous gold is not more complete, for it might throw considerable light on the subject which we are now discussing.

Gold forms two well-defined series of compounds, in one of which it is univalent and in the other trivalent. It also forms a few intermediate compounds in which it is bivalent. It has few solvents and consequently forms few compounds. With one exception, which will be discussed later, the compounds of gold are extremely unstable. So weak are the bonds which ordinarily hold gold in combination with other elements that in many cases the mere application of either light or heat is sufficient to bring about decomposition.

In view of the foregoing, it is a distinct surprise to note extraordinary stability of the compounds of gold and cyanogen, particularly potassium aurocyanide, which is formed according to Elsner's equation as follows:

$$4KCN + 2Au + O + H_2O = 2KAu(CN)_2 + 2KHO.$$

The reagents which ordinarily precipitate gold from solution have absolutely no effect on this compound. This is one of the reasons why the chemistry of the process has made so little progress. Chemists have hesitated about engaging in experimental work for the reason that, although they might be able to discover more effective compounds of cyanogen, or chemicals, the addition of which to the working solutions would greatly increase their efficiency, the problem of precipitating the precious metals would be vastly greater than that of getting them into solution. We are consequently working with the same solutions to-day that we used 20 years ago.

The fact that there are so few reagents which will precipitate gold from cyanide solutions is conclusive proof of the stability of potassium aurocyanide. In view of the foregoing, there is a great deal of interest attached to the statements which are so often made now-a-days by chemists to the effect that gold and silver are subject to precipitation in the tanks by reaction between the solution and certain mineral constituents of the ore. The list of compounds which are supposed to possess the power of decomposing potassium aurocyanide is a very long one and includes carbon in various forms, and also the sulphides, arsenides and tellurides. The foregoing are the minerals which metallurgists have generally recognised as the cause of 'premature precipitation' but to this list, H. Forbes Julian has recently added grease, oils, and silica and alumina in a fine state of division.

The 'premature precipitation' theory is of comparatively recent growth, but, although there are a great many who believe in it, the insufficiency of the evidence so far brought forward to support it, has led many thinking chemists to withold their opinion until such time as the facts would warrant them in taking a stand on the matter.

The purpose of this discussion is to throw some light on the subject and to determine whether or not it is something with which we may begin to reckon, or, like the lawyer's John Doe, it is merely a convenient fiction which we can employ as occasion requires to assume the responsibility for the shortcomings of one to whom, perhaps, we would not like to give name. In this paper I shall merely present the main facts in connection with the subject under discussion, leaving them to be taken up in *minutia* by others. There are many points on which it will be impossible for me to say anything whatever, for the reason that details are entirely lacking, as, for example, the effect of oil on precipitation. In such cases, it is hoped that those who have any information on the subject will make it public.

Of the various substances which it is claimed will produce 'premature precipitation,' carbon is unquestionably the most important, for the reason that in the form of charcoal it is one of the recognised commercial means of recovering the precious metals from cyanide solutions. Although not employed very much at the present time, it was used a few years ago quite extensively in Australia, and its efficiency as a precipitating agent cannot be questioned.

Although the effect of charcoal on cyanide solution has long been known, it took chemists some time to recognise the fact that its presence in ore undergoing treatment was responsible for a loss of gold and silver. It was at a time when tailing piles were much more common than they are now that metallurgists first began to appreciate the effect that it might have on their profit sheet. The old beds contained it in considerable quantities and for some years it was quite a problem to contend with. At the present time tailings are very scarce and we seldom hear about charcoal causing 'premature precipitation,' but we have in its stead graphite, which, as I shall soon show, is a legacy handed down to us by the workers of the 'charcoal days.'

Until charcoal came to be recognised as a common constituent of old tailings, the 'premature precipition' theory was unheard of. It originated without doubt in the discussions which took place at that time when the effect of charcoal and organic matter was being taken up. There is no question whatever but that 'premature precipitation' was responsible in many cases for material losses, but it must also be admitted that losses very often took place which were not brought about in that manner. It is easy to see how the theory grew in popularity, for it served as a convenient way of explaining the failure to obtain a high extraction when the real cause was unknown. We can also see how it came about that graphite came to be recognised as a cause of 'premature precipitation.' If carbon in the form of charcoal precipitated gold, why should it not do so when it was in the form of graphite? We will all agree that the argument is ingenious, though we may not agree as to the similarity of the behaviour of charcoal and graphite with cyanide solution.

Will graphite precipitate gold from an alkaline solution containing free cyanide under the conditions which exist in cyanide treatment? If it will, then the 'premature precipitation' theory is firmly established, and furthermore we will have to admit that potassium aurocyanide can be decomposed by any compound in a finely divided condition. The effect which this would have on 'fine grinding' is problematical, but one thing is certain and that is, we would have to revise many firmly rooted ideas and start investigations along new lines of treatment.

My reason for stating that if graphite will precipitate gold under the conditions stated, any mineral in a finely divided state will do the same, is that if graphite does bring about a reaction, it is because of its physical condition and not because of any chemical change which takes place and in which it participates to the extent of passing into solution. Graphite, as we all know, is reduced by grinding to an extremely fine state of division, and the reaction, if any actually does take place, is made possible only by its existence in that condition. The only other explanation which can be advanced is that the action is catalytic, but this is highly improbable. The well known fact that charcoal precipitates gold has often been cited to prove that graphite does the same, but a consideration of some of the facts in connection with the action of charcoal will establish the point that if graphite actually does react with potassium aurocyanide, the reaction is of an entirely different character from that with charcoal.

I wish first to make two statements which will have some little bearing on what follows. First of all, it has never been conclusively proven how it is that charcoal precipitates gold and silver. Second, all charcoal contains more or less occluded gas, which consists of hydrogen and various powerful reducing hydrocarbons.

The first person to recognise that charcoal could be used to precipitate gold from cyanide solutions appears to have been W. D. Johnson of San Francisco, who in 1894 took out a patent covering its use in this country. In his patent specifications, Johnson merely states that carbon precipitates gold from cyanide solutions and he makes no attempt whatever to explain the reaction, and the inference is tha it was just as much a mystery to him as it is t chemists at the present day. There are three possible explanations of its action: (*a*) It is due to a chemical reaction in which the carbon participates. (*b*) the carbon is passive and the reaction is brought about by osmotic pressure, which is set up when the solution passes through the pores of the charcoal. (*c*) It is due to a reaction with the occluded gases.

Regarding the first explanation that it is due to a chemical reaction in which the carbon participates, I hardly think that any one, after giving the matter a little careful thought, will be willing to admit that he thinks that such a reaction actually takes place. At ordinary temperatures, carbon, in whatever form you choose to consider it, represents the most extreme type of a chemically inactive element. That it should decompose a compound of such remarkable stability as potassium aurocyanide and pass into solution seems absurd. It should not even be necessary to have to establish this by experiment. A general knowledge of the character of carbon and cyanogen should enable one to clearly see that a reaction between the two under the circumstances which exist in cyanide treatment is an impossibility. There is absolutely no evidence to prove that the reaction which results in the precipitation of gold by charcoal is one between carbon and potassium aurocyanide. On the contrary there is much which goes to prove that such a reaction does not take place.

The second explanation, that the carbon is passive and the reaction is brought about by osmotic pressure, which is set up when the solution passes through the pores of the charcoal, is quite plausible, but its acceptance makes it necessary to assume that there is something present which will react with the gold compound. The occluded gas supplies the necessary member to complete the reaction.

That the reaction is due solely to the occluded gas does not seem possible. Hydrogen is present in the charcoal and no doubt exists in the precipitation of the gold. Under ordinary circumstances this gas is inactive but when it is brought in contact with a compound with which it can react and an osmotic pressure is set up, it is easy to see that combination will take place. Eliminate osmotic pressure as a factor in the precipitation and it is difficult to explain the action of charcoal in precipitating gold from a cyanide solution. My explanation of the phenomenon is then that it is due to a reaction between potassium aurocyanide and occluded gas under the influence of osmotic pressure.

That the presence of a reducing gas in the charcoal is absolutely essential to the reaction, a brief consideration of some of the observed facts will confirm. Freshly burned charcoal has a high efficiency as a precipitating agent; old charcoal has a very low one. Analysis has established the fact that as charcoal gets old, it loses its hydrocarbon gases. This explains the efficiency of fresh charcoal. In the plants employing charcoal as a precipitating agent, it has been found that after a time the charcoal becomes inert and loses its power to precipitate. Its efficiency is restored by re-burning. From this we deduce, that whatever it is that precipitates the gold, it is something which is formed at a high temperature. Hydrocarbon gases, as everyone knows, are formed when charcoal is burned under certain conditions. One of the most important properties of charcoal is its power to absorb gases, and for which purpose it finds many important applications in the arts. Its power in this direction varies with the character of the wood from which it is made but to give an idea of what it will do, I can quote the extreme case of charcoal made from cocoa-nut wood which will absorb 172 times its volume of ammonia gas. This will give us an idea of the large volume of hydrocarbon gas which it is possible for the charcoal to absorb. It has also been observed that charcoal which has been used to precipitate gold does not show any marks of corrosion. The carbon therefore does not enter into combination with the compound which it decomposes.

The above are the main facts in the case and from them but one conclusion can be drawn, and that is that the occluded gas is the real precipitant of gold. This is the generally accepted opinion of those who have given the subject careful consideration. As to whether osmotic pressure plays any part in the reaction there is no positive proof, but as it seems quite possible that it does, I have advanced it as a partial explanation of the phenomenon.

We now come to the subject of graphite and its effect on potassium aurocyanide. If we admit, as we must, that the precipitating power of charcoal is due to occluded gas, how can we explain the action of graphite, admitting for the sake of argument that it does precipitate gold. Graphite does not contain any hydrocarbon gas nor anything else that would tend to decompose a chemical compound. If it does produce a reaction, then it must be of a catalytic nature, but from our general knowledge of this subject, this seems highly improbable. Graphite has not the peculiar celluar structure common to most catalyzers and furthermore, compounds susceptible to catalytic action, should be extremely active chemically, or in other words, in a state of unstable equilibrium. Potassium aurocyanide is anything but an unstable compound.

It has been shown experimentally, that certain compounds in a minute state of division, sometimes possess the property of effecting the decomposition of salts in solution. This property is not common to all substances and furthermore it is a reaction which proceeds very slowly and only under favourable conditions. It is quite unlikely that graphite can effect the decomposition of potassium aurocyanide in the manner described and under the conditions which exist in cyanide treatment. It is quite probable, however, that many will contend that this is the explanation of its action; but it will be for them to prove it; and until the necessary evidence is forthcoming, the effect of graphite on potassium aurocyanide will have to be recessed in the realms of conjecture.

From the foregoing it will be seen that there is a grave doubt as to whether graphite really precipitates gold from cyanide solutions. There are many cases, however, on record, where chemists have reported having observed its action, and one of these I shall quote as a typical illustration of the character of the evidence advanced to prove that graphite produces 'premature precipitation.'

In an article entitled 'graphite—An Obstacle to Good Cyaniding,' M. W. von Bernewitz says: 'In the treatment of the sulpho-telluride ores of Kalgoorlie, it was noticed in certain mills, that the extraction occasionally fell off for no apparent reason, although all conditions were favourable for good work. Ore from certain parts of the mines was known to contain graphite, and it was when this class was being milled that the trouble occurred. * * * I think there is little doubt that a precipitation of the gold takes place while the graphite is in violent agitation with the pulp in the pans. * * * (The ore is reground in pans with weak cyanide solution.) As to it taking place in the agitators with 0·08% KCN, no means are available for determining.

'Extraction generally drops from 2% to 5% when graphite is present, varying with the grade being milled. Scum collected from the pans was filtered and washed carefully and assayed. In the cupel a fair sized bead was seen but as the quantity assayed was small and there was nothing definite as to the amount of graphite in the ore, no calculation of its value was made. It seemed clear, however, that the graphite had precipitated this gold from the circulating solutions.'

Let us examine closely into Mr. von Bernewitz's reasons for believing that graphite precipitates gold. First of all there is the coincidence of the falling off in the extraction with the appearance of the graphite. This is not evidence which will prove his contention. It might be said that the graphite coming into the ore, indicates peculiar conditions existing at the points where it is found, during the period when the circulating waters were depositing their mineral contents, and these conditions effected certain changes in the character of the surrounding tellurides which makes them more difficult to treat than the tellurides from other places where the conditions were different.

The only direct proof which he offers is the high value of the scum collected from the pans. Everyone who has had any experience in the treatment of telluride ores will appreciate the weakness of this evidence. Telluride ores are notorious for their sliming qualities which even roasting does not prevent. I have, myself, frequently collected scum from tanks containing roasted telluride ores from Cripple Creek, which assayed over $200 per ton, while the average value of the ore itself was not over $20 and there was no graphite present in the ore to lend a suggestion that it might be responsible for the high value.

I do not know how others may regard Mr. Bernewitz's opinion, but to me it appears that he has failed to establish his claim that graphite precipitates gold, and I might say the same of many other writers. Before leaving the subject of graphite, I wish to state a case which came under my own observation.

Some years ago the tailings of the Argonaut mine at Jackson, California, were running very high, owing, as those who were familiar with the ore thoroughly understood, to the presence of arsenical pyrite which carried the gold. The treatment consisted of ordinary amalgamation and concentration, and it was thought that by supplementing this with cyaniding a further profit could be obtained. A chemist was employed to make a series of experiments and a report was eventually rendered in which the statement was made that owing to the presence of graphite which brought about a 'premature precipitation,' the process was inapplicable. Some time afterwards I was asked to render an opinion on this 'premature precipitation' phase of the report, and in order to arrive at a clear understanding of the matter I secured some of the offending wall rock from which the graphite was supposed to be derived, and subjected it to a number of tests.

Several pounds of the rock were crushed dry to pass a 30 mesh screen. The material was then carefully dried to expel the moisture, and the sample was then placed in a glass percolator. A quantity of cyanide solution containing a known quantity of gold was then allowed to filter through for a period of 4 days. The value of the filtrate was unchanged. The experiment was then varied by introducing 50 gm. of finely divided graphite into the pulp. The solution was run through again with similiar results.

Before leaving the subject it might be well to consider briefly the effect of the solution on the precipitated gold, assuming that decomposition of potassium aurocyanide is brought about by carbon.

When gold is precipitated by charcoal it comes down in the amorphous form, and we may assume that if graphite has any effect, the gold will appear in a similiar state. Precipitated gold is extremely light and flocculent and judging from the solubility of leaf gold in cyanide solution we would naturally suppose that amorphous gold would be even more soluble. My attention had never been called to a possible difference in their solubilities before, but it occurred to me that there might be a difference and I thought it advisable to look into the matter. The various authorities are silent on this point and I found it necessary to investigate it myself. Unfortunately my laboratory facilities were limited and my experiments were purely qualitative. A small amount of precipitated gold was obtained by dissolving some cornets in aqua-regia, evaporating, re-dissolving in cyanide solution, and precipitating with zinc shaving after acidulation. The precipitate was then digested in sulphuric acid to remove the excess of zinc after which it was thoroughly washed with warm water. Approximately 100 mgm. of the precipitate was transferred to a litre flask and 800 c.c. of a 10 lb. cyanide solution was added. Several hours elapsed before there was any perceptible evidence that the gold was dissolving. At the end of 24 hours I estimated that one fourth of the gold had passed into solution. At the end of 4 days, when the experiment was brought to a close, there was still quite a considerable quantity of gold undecomposed. The conditions were extremely favourable at all times for a reaction.

The flask had been left in a warm place and the contents had been agitated from time to time. The experiment was tried again with a weaker solution and with similiar results. The two experiments were then repeated with 500 mgm. of precipitate and 1 sq. in. ot leaf gold. The leaf gold was placed in the flask along with the amorphous gold, my idea being to note the difference in the solubility of the two under similiar conditions. At the end of 2 hours the greater part of the leaf gold was in solution and apparently none of the precipitate. At the end of the 24 hours, all of the leaf gold had disappeared and only a small quantity of the amorphous gold. As before, at the end of 4 days some of the gold was still visible to the eye.

These experiments, while very crude, give some very valuable information. They show that amorphous gold is not as soluble in cyanide solution as the yellow variety. If 'premature precipitation' does take place, it probably occurs towards the end of the treatment when the solution has lost the greater part of its oxygen and become less active. If the gold is precipitated at this stage of the operations, we can readily understand now how it is that in the short time elapsing before the vats are discharged it is impossible to redissolve this gold by addition of fresh solution. In my experiments I used pure solutions and the conditions were much more favourable for the solution of the gold, than would exist in commercial work. It is of considerable importance that the facts surrounding the solubility of precipitated gold in cyanide solutions, be ascertained and I would suggest that some of the members of the society examine into it quantitatively and study the effect of impure mill solution. If we eventually decide that 'premature precipitation' is of common occurrence, this information will be of the greatest possible assistance to us in finding a means of recovering this gold before bringing the treatment to a close.

Before entering into a consideration of the effect of some of the other minerals, it might be well to say a few words about the effect of organic matter as, in a way, it is allied to the subject of carbon.

Like charcoal, organic matter is a common constituent of old tailings. It is found in the form of decaying vegetation, animal remains, sewage, etc. That some of it causes a loss of the precious metals there is no question whatever, for leaves and similiar matter removed from tanks after treatment have shown their presence by fire assay. But that all of the gold and silver found in leaves and similiar matter is the result of a chemical reaction, I have very serious doubts. Because of the presence of this organic matter, which is of a reducing nature and causes a dearth of oxygen in the solution, it is the general practice to apply the solution in the form of intermittent washes, allowing the tank to drain and absorb oxygen after each application. This treatment gives the leaves and other woody matter a chance to absorb several successive charges of gold bearing solution, with an opportunity to partly dry out between each one The presence of the gold can therefore be charged in part to a purely mechanical agency, evaporation, and not to a chemical reaction. I do not wish it thought that I am advancing this as a theory. I merely offer it as a possible explanation of certain cases where the presence of the gold cannot be accounted for in any other way. I know that there are a great many who believe that gold is precipitated in wooden vats and zinc boxes by the action of the organic compounds in the wood, whereas the real explanation is that it was deposited there by the process of evaporation.

No doubt exists in my mind but that organic matter sometimes causes 'premature precipitation' but I consider such cases extremely rare. Organic matter, particularly that in an advanced stage of decomposition, is a powerful deoxidising agent. Oxygen, as we know, is absolutely essential for the reaction with gold and it is easy to understand, how, in the presence of considerable organic matter, which is robbing his solution of oxygen and thereby impairing his extraction, a chemist is very apt to credit the 'premature precipitation' theory with a lot of things for which it is in no way responsible.

The sulphides have been charged with having been responsible for certain cases of 'premature precipitation.' Like similiar accusations of this kind, the charges have been so loosely drawn that it is difficult to say just which of the large and populous sulphide family it was intended to incriminate, I shall take up one case however, that of the soluble sulphides.

The alkaline sulphides are of common occurrence in mill solution. Their presence may be accounted for in two ways; They occur as an impurity in the cyanide, and are a product of the decomposition of certain minerals. If a soluble sulphide reacts with a compound of gold we would expect it to precipitate the sulphide of gold, Au_2S_2. In the experiments of Prof. Christy it was shown that the sulphide of gold formed by the action of hydrogen sulphide on the chloride, dissolved completely in 1 hour in a 4 lb. solution of potassium cyanide. In stronger solutions the reaction occupied but a fraction of this time. From this we may deduce the fact that soluble sulphides will not precipitate gold from cyanide solutions, at least not in the presence of an excess of free cyanide.

This opens up the question as to what effect the soluble sulphides actually have on the solution. We know for one thing, that their presence causes a decided falling off in the extraction. Julian and Smart have shown that the addition of 5 gms. of sodium sulphide to a ton of solution of 5 lb. strength causes a loss of 34% of its dissolving power. The probable explanation of the effect of sulphides of this type is that they are deoxidisers, that is, they remove the oxygen from the solution, thereby retarding the dissolution of the gold. It is easy to see now how a chemist who is accustomed, we will say, to obtaining a certain extraction in 5 days, suddenly finds his tailing running very high and is led to believe that it is a case of 'premature precipitation' when as a matter of fact, soluble sulphides or some other deoxidising agent has appeared in his solution, and simply retarded its dissolving action. I think that investigation will show that a great many cases of so-called 'premature precipitation' are of this character.

It is not my intention to go into the behaviour of all the various substances which are supposed to produce 'premature precipitation' but rather to leave them open for discussion. I have outlined in a general way the subject as it is viewed by many chemists, with the expectation that others will take up the matter and either criticise my views or amplify my statements. 'Premature precipitation' is a live issue. It has been discussed, in a vague, hesitating manner, for some years and the time is ripe for us to either recognise or reject it altogether. Those who have any knowledge regarding cases which have come under their observation should make the facts public. Direct evidence is invaluable, but we must not forget that discussion from a purely theoretical standpoint has its value also, and it is to be hoped that those who may be able to throw a little light on the subject will do so for the general good of the profession."—R. STUART BROWNE.—*American Metallurgical Society.*—*Mining and Engineering World.* Dec. 2, 1911, p. 1108. (A. R.)

HIGH-GRADE BULLION FROM ZINC-BOX PRECIPITATES.—"Owing to the combined treatment of acid oxidized ore and alkaline roasted ore at the Chaffers gold mine in Western Australia considerable difficulty was often experienced in keeping the alkalinity of the gold-bearing solutions at the most suitable point for good precipitation of the gold. In order to aid precipitation, an extra zinc-box was constructed and brought into use.

With the introduction of the extra box combined with a reduction in the grade of ore sent to the mill, a very noticeable reduction in the value of the bullion produced was observed.

In spite of all precautions as regards acid treatment, washing, roasting and smelting, the value of the bullion fell from £3 16s. 6d. ($18·614) per oz. to £3 13s. 8d. ($17·923) per oz , and on one occasion was as low as £3 9s. 6d. ($16·921).

As this meant a considerable loss in refining and banking charges, steps were taken to raise the standard of bullion produced; the various well-known methods were given careful consideration, and eventually the methods detailed below were put into practice with highly satisfactory results. (See table F.)

Clean-up Procedure.—A brief description of the procedure at clean-up is here given in order to follow the various points in connection with tables showing results obtained from actual and trial smelts, and to emphasize the deductions drawn therefrom.

All the precipitate and short zinc is sent to the acid vat, and only the well-washed long zinc is returned to the boxes; no scrubbing or screening of short zinc is attempted, it being desirable to obtain the maximum amount of gold possible.

After acid treatment—the finishing point being determined by the absence of action on the addition of acid—the diluted precipitate is pressed in a small filter-press, where the pressed precipitate is given three or four washes of water, brought close up to boiling point in order to wash out the soluble sulphates.

The cakes are air-dried, and dropped into shallow iron trays, weighed, and placed in a cast-iron muffle, where the precipitate is subjected to either a roast at dull red heat, or is merely dried as is shown in Table F. No screening of the dried or roasted precipitate is carried out, as practical smelting operations prove that the practice of screening sometimes adopted has no bearing on the production of high-grade bullion. Screening, though lending itself to accuracy in sampling and assaying of dried or roasted precipitate, would probably lead to loss through dusting.

The roasted or dried precipitate. after being weighed and sampled, is mixed with flux as follows: 100 parts precipitate, 50 parts borax, 20 parts sand, and in some cases 7% to 10% of nitre has been added to the charge. (See Table F.) This mixture is smelted in clay liners in plumbago pots, No. 100. When fusion is quiescent, the charge is poured into conical moulds producing high-grade buttons of gold bullion ranging in value from £4 3s. 7d. ($20·388) to £4 4s. 4d ($20·520) per oz. which are subsequently smelted in small plumbago pots, No. 30, and the gold poured into bar moulds of 500 oz. capacity.

Full details of acid treatment, washing, roasting or drying conditions under which smelts have been conducted, and the results of bullions produced are to be found in the above mentioned Table F.

Investigation of the physical nature of the roasted or dried precipitate, and of the bullion under consideration, makes it plain that it is easier to produce high-grade bullion direct from the roasted or dried precipitate than to produce low-grade bullion and afterwards refine it; moreover, there is less chance of loss where the refining is carried out through the medium of the flux than in the refining of the metal.

Given that the acid and wash treatment have been carefully carried out, the roasting at a dull red heat often resorted to is unnecessary, as will be shown later, with regard to obtaining high-grade bullion. Drying at 100°C. is all-sufficient for obtaining the desired result.

Supposed Losses Due to Nitre.—The production of high-grade bullion is mainly a question of thorough acid treatment, thorough washing, and the selection of the proper fluxes in the fusion process. The use of liners inside plumbago pots, whether nitre is used in the fluxing or not, is a powerful factor in the production of high-grade bullion, guarding, as it does, against the reduction of metallic zinc from the zinc sulphate or zinc silicate.

Numerous authorities have enlarged on the losses of gold when nitre is used for the refining of gold bullion. Losses of from 10% to 25% are stated to have occurred. "*Avoid the use of nitre*" is a common phrase in the text books.

Finding no actual data as to how the nitre was applied, whether to the flux or direct to the bullion, one is unable to oppose the foregoing conclusion, but from the following Tables A, B and C, it will be observed that when nitre was used as a flux and the smelting was effected in plumbago pots with clay liners, there was practically no loss of gold.

Table A shows the result of trial smelts made in order to ascertain the loss of gold by volatilization, using nitre as a flux; the smelts were carried out in clay pots.

TABLE A.

Charge—	(1)	(2)
Precipitate, gm.	50	50
Borax, gm.	25	25
Sand, gm.	7·5	10
Nitre, gm.	Nil.	5
Weight of bullion, gm. ...	20·5	20·9
Percentage	41	41·8
Fineness of gold	995·6	997·6
Fineness of bullion	996·6	995·8
Slag values per 2,000lb ..	22 dwt.	16·4 dwt.

The results of Table A show that the losses generally attributed to the use of nitre as a flux are invariably overestimated. This view is strongly supported by the following comparisons drawn between the gold called for by assay of dried precipitate and the gold recovered (June half month clean-up), Tables B and C.

TABLE B.

ASSAY OF ROASTED PRECIPITATES.

Raw sludge, 151lb.
Roasted sludge, 89·5lb.
Fine gold. 38·1lb. (42·66% or 555·49 fine oz.)
Value as assayed, £2,359 15s. 2d. ($11,480·073.)

TABLE C.

ACTUAL SMELT RECOVERY.

		£	s.	d.
Fineness of gold	993			
Value, Bar 113		1,000	11	5
Value, Bar 114		1,334	17	5
From slag		3	16	7
Left in slag		2	16	0
Mint surplus		2	10	0
Total Clean-up		2,344	11	3
		($11,409·854)		

The deficit on the assay of roasted precipitate 0·64%, indicates that little or no loss has taken place; 0·64% is within the error of weighing, sampling and assaying.

Sampling Dried or Roasted Precipitate.—The roasted precipitate, after being weighed, is broken down gently with the back of a square-mouthed shovel, and well mixed, every precaution being taken to avoid loss by dusting. It is then carefully quartered down, and the final sample weighing about 4lb. is taken to the assay office, where it is first ground down to pass 10 mesh, again quartered down and ground to pass 20 mesh, after which the final assay sample is cut out, ground to pass 80 mesh, metallics (if any) removed, and the sample assayed.

The small percentage of error is probably due to the practice of weighing the precipitate on ordinary counter scales, and on the whole the method, if carefully carried out is satisfactory.

Drying v. Roasting.—The drying of the precipitate, as against roasting at dull red heat, gives a more friable and more uniform product for sampling, and should give more satisfactory results.

In conjunction with the final clean-up for the month of May, a number of trial smelts that were conducted gave some interesting results (see Table D) from which useful deductions may be drawn. All the trials were conducted on the roasted precipitate recovered at the same clean-up; therefore they underwent the same processes of acid treatment, washing and roasting.

TABLE D.

	(1)	(2)	(3)
Smelted in ...	Plumbago.	Clay.	Clay.
Flux used—			
Precipitate (parts)	100	100	100
Borax (parts) ...	50	50	50
Sand (parts) ...	15	15	20
Nitre (parts) ...	Nil.	Nil.	10
Fineness of gold ...	890·8	995·6	995·8
Value per oz. ...	£3 15s. 7d. ($18·30)	£4 4s. 7d. ($20·58)	£4 4s. 7d. ($20·58)

(1) represents the final actual smelt for May; (2) represents a trial smelt in clay pot, and (3) a trial smelt in clay pot with the addition of nitre. All the material tested was roasted at red heat.

The subjoined deductions are drawn from the foregoing Table D:

(*a*) That the liners were in the main responsible for the high-grade bullion produced.

(*b*) That the use of nitre as a flux was unnecessary.

(*c*) That the acid treatment was practically perfect, otherwise metallic zinc would have come down in the trial smelt. No. 2 bullion produced was 995·6 fine.

(*d*) That the washing was imperfect, as is shown by the low-grade bullion produced in actual smelt (890·8 fineness) in plumbago pot.

Deductions (*a*) and (*b*) are again supported by the results obtained in practice from the July half-month and July final clean up (see Table F), when bullion of the value of £4 4s. 4d. and £4 3s. 7d. per oz. respectively was produced. Liners were used, but no nitre was used in the flux.

At the July final smelt one of the liners in use was eaten through, with the result that the bullion fell a little in grade (£4 3s. 7d.), due to the reduction of metallic zinc from the zinc sulphate or the zinc silicate. That this reduction takes place when the smelt is exposed to the reducing action of the plumbago pots is demonstrated when a bad break occurs in the liner, or when the liner is eaten through.

Most authorities on the cyanide process are emphatic on the necessity of a dull red heat for roasting the precipitate before smelting operations. In the event of the acid treatment being imperfect either owing to insufficient acid (an unpardonable error), or to protective coatings of insoluble lead or calcium sulphates deposited on the zinc shavings, a thorough good roast would be necessary, or the use of clay liners and nitre flux would have to be resorted to in order to obtain high-grade bullion.

Roasting at dull red heat would also hold good in the event of the acid treatment being discarded as is sometimes done. The use of liners and nitre would also meet this particular case.

The object of roasting at dull red heat is to oxidize any metallic zinc present, but roasting opens an avenue for loss of gold through the gold being carried over with the copious fumes of zinc; therefore, it is advisable to give a thorough preliminary acid treatment, and thus avoid the necessity for roasting at red heat.

Given that the acid treatment is practically perfect, then the drying of the precipitate and subsequent smelting in clay liners in plumbago pots is all that is necessary to obtain bullion of high grade.

The following Table (E) of trial and actual smelts carried out on the raw precipitate obtained from the July half-monthly clean-up will clearly demonstrate the point. The precipitate used for trial and actual smelts received the same acid treatment and washing. (See Table F.)

The deduction to be drawn from Table E is that roasting at red heat is unnecessary; drying of the thoroughly acid-treated precipitate being all that is required to produce high-grade bullion (see smelts No. 1 and 4, Table E), and a reference to Table F will show that bullion of the value of £4 3s. 7d. per oz. was produced at the July final clean-up, when the precipitate was dried at a dull heat, and smelted in clay liners in plumbago pots, without the use of nitre in the flux. Table E also clearly supports the deductions (*a*, *c* and *d*) drawn from the results of Table D.

Elimination of chief base metals.—Zinc.—With regard to the base metals present in the precipitate under consideration, the only troublesome base is apparently zinc, which was dealt with by thorough acid treatment, washing as far as practicable, drying of the precipitate, and the subsequent smelting in clay liners in plumbago pots. The result of this treatment is satisfactory, as will be seen by Table F, reference Nos. 9 to 12.

Copper.—Practically no copper is present, and proof of this is seen from the high-grade bullion produced by the use of liners without nitre flux. Were copper present in the precipitate, then the fine filaments of this base metal would remain unchanged throughout the acid and wash treatment and metallic copper would come down in the bullion unless precautions were taken for the elimination of this base metal. The use of clay liners and nitre flux, in this

TABLE E.

Drying.	(1) Dried at dull heat.	(2) Roasted at red heat.	(3) Dried at 100 C.	(4) Dried at 100°C.	(5) Roasted at red heat.
Smelted in	Plumbago.*	Plumbago.	Plumbago.	Clay pot.	Clay pot.
Flux—					
Precipitate (parts)...	100	100	100	100	100
Borax (parts)	50	50	50	50	50
Sand (parts)	25	25	25	20	20
Nitre (parts)	Nil.	Nil.	Nil.	Nil.	Nil.
Fineness	992	779·1	785·5	988·85	987·75
Value per oz.	£4 4s. 4d. ($20·52)	£3 6s. 2d. $16·10)	£3 6s. 8d. ($16·22)	£4 4s. 0d. ($20·44)	£4 3s. 11d. ($20·42)

NOTE.—No. 1 represents the actual smelt for July; Nos. 2 to 5 inclusive represent trial smelts.
*Clay lined.

TABLE F.—DETAILS OF CLEAN-UP, COVERING THREE MONTHS: MAY, JUNE, JULY, 1911.

Reference No.	Clean Up.						Fluxes Used at Smelting.						Valuation of Precipitate by Assay.					
	Period.	Raw precipitate lb.	H_2SO_4 used, lb.	Wash used gallons.	Loss in drying per cent.	Dried precipitate, lb.	Roasting nitre, per cent.	Precipitate, parts.	Borax, parts.	Sand, parts.	Nitre, parts.	Fine gold, per cent.	Fine gold, lb.	Fine gold, oz.	Value. £	s.	d.
1 2	May half-month...	116	208	360	39·6	70·0	nil	100	50	15	nil	43·45	30·41	443	1,883	0	0
3 4	May final... ...	166	234	360	41·5	97·0	nil	100	50	15	nil	36·87	35·76	521	2,214	8	0
5 6	June half-month	151	208	360	40·7	89 5	nil	100	50	20	7	42·66	38·1	555·49	2,359	15	0
7	June final ...	151	208	480	35·7	97·0	5	100	50	20	3	36·69	35·5	518·75	2,203	10	0
9	July half-month	110·5	130	480	43·8	62·0	nil	100	50	25	nil	35·9	22·25	324·4	1,378	0	0
11	July final ...	131·5	156	480	37·3	82·5	nil	100	50	25	nil	36·52	30·1	439·28	1,866	0	0

Reference No.	Bullion Bars.													Total clean-up including slags.			Slag contents per ton
	Period.	Bar No.	Bullion, ounces (Troy).	Fine gold, ounces (Troy.)	Gold, fineness.	Bullion, fineness.	Base, fineness.	Total value. £.	s.	d.	Value per oz. £	s.	d.	£	s.	d.	ounces.
1	May half-month	109	256·65	201·19	819	824·66	175·34	892	18	3	3	9	7·4	1,863	13	0	70
2		110	263·50	218·96	831	837·36	162·64	930	3	7	3	10	7·6				
3	May, final	111	273·25	243·19	890	892·70	107·30	1,033	16	7	3	15	7·4	2,170	18	3	20·2
4		112	292·65	260·75	891	895·12	104·88	1,107	13	5	3	15	8·6				
5	June half month	113	237·2	235·53	993	994·61	5·39	1,000	11	5	4	4	4·5	2,344	11	5	13·6
6		114	316·4	314·23	993	994 33	5·67	1,334	17	5	4	4	4·5				
7	June final	115	522·05	514·74	986	992·9	6·1	2,186	12	1	4	3	6	2,193	14	9	12·44
9	July half-month	117	336·2	333·51	992	995·74	4·26	1,416	15	0	4	4	4	1,418	10	0	7·7
11	July final	118	440·15	431·79	981	983·22	16·78	1,834	4	8	4	3	7	1,867	4	8	10·0

Nos. 1 to 4.—The smelts were carried out in plumbago pots, the precipitate roasted at red heat.
Nos. 5 to 8.—The smelts were carried out in clay liners in plumbago pots, the precipitate roasted at red heat.
Nos. 9 to 12.—The smelts were carried out in clay liners in plumbago pots, the precipitate dried only at dull heat.

case, absolutely essential, would result in high-grade bullion. Personal practical experience has proved that precipitate containing a large percentage of copper which in the plumbago pots gave bullion worth only £2 2s. ($10·22) per oz. was raised to £4 3s. 3½d. ($20·27) per oz. by the above-mentioned method.

Lead.—The practice of washing zinc shavings in a solution of lead acetate often accounts for the presence of lead as base in bullion. The base metal may be removed by liners in plumbago pots using nitre as flux, but the method is slow and tedious. Manganese dioxide is often added to facilitate operations, and is preferable, because the elimination of the lead is performed more quickly.

General Deductions.—1. No screening or scrubbing of short zinc is necessary, the cost of extra acid being more than compensated for by the reduction of refining and banking charges.

2. Acid treatment must be as complete as possible, and that it is possible is proved by high-grade bullion produced by plain fluxing in clay liners in plumbago pots.

3. Washing in the press is by no means perfect and leaves much to be desired.

4. Roasting at red heat is unnecessary (exceptions given in text).

5. Drying of the precipitate is all that is necessary under ordinary conditions.

6. Clay liners are essential for the production of high-grade bullion unless the press wash can be much improved upon. The presence of $ZnSO_4$ being against direct smelting in plumbago pots.

7. Screening of the dried or roasted precipitate before fluxing is quite unnecessary.

8. Nitre flux is absolutely unnecessary when zinc in the form of zinc sulphate in the dried precipitate is the cause of the base bullion.

Advantages of Using Clay Liners With or Without the Use of Nitre in Flux.—1. Slags of lower value are produced. (See Table F.)

2. The charge fuses more quietly than in plumbago pot.

3. The absence of obnoxious fumes; a point much in its favour.

4. Lower bullion realization cost.

The time of smelting is slightly longer, the cost may be a trifle higher, but this disadvantage is more than compensated for by the lower realization cost.

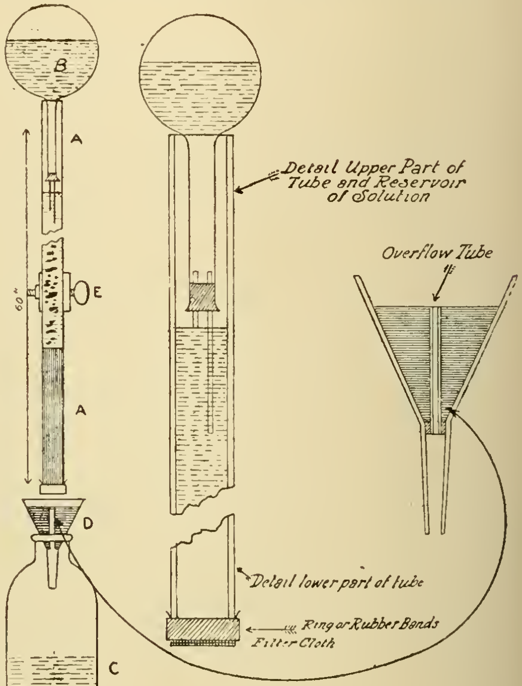
It may be argued that handling of small parcels of precipitate makes the smelt in clay liners and plumbago pots an easy matter. The latest patent tilting furnace furnished with special salamander crucibles fixed in position, and salamander hood, covering, etc., combined with clay liners, fills a long felt want, and leaves no ground for the above argument. The furnace can take charges up to 400lbs., and costs somewhere in the vicinity of £140 ($681).

These furnaces are reported to be giving excellent results in Rhodesia."—J. Boyd Aarons and Herbert Black, *Journal of the Chamber of Mines, of Western Australia*; *Mining and Engineering World*, Nov. 11, 1911, p. 955. (A. R.)

Laboratory Cyanide Tests.—"It has been my object to devise an apparatus for laboratory cyanide tests that will duplicate the conditions of actual practice, and that will handle quantities large enough to obtain accurate results, in checking solutions and tailing after treatment. I have used such an apparatus over a year with gratifying results.

The accompanying sketch shows the apparatus now in use. A large glass tube, *A*, 60 in. long and not less than 1¼ in. internal diameter (1½ in. is preferable) at the lower end, is covered with a filter medium of cloth or canvas, which is held in place by a tight-fitting ring of metal or by rubber bands. The measuring flask, *B*, should have a neck small enough to enter the glass tube and a capacity of 2,000 cc. This is fitted with a cork, through which two small tubes enter the neck of the flask. One protrudes about an inch and the other must be at least three inches longer. The purpose of this arrangement is to provide a self-regulating feed to the percolation tube, *A*, and to prevent the contamination of the solution by the laboratory gases.

As the shorter tube provides the only inlet for air, its length regulates the height of the liquid column in *A*. Its length can be regulated as desired, but the two tubes must differ in length sufficiently to assure a flow of solution. A difference of three inches is sufficient. *D* is a funnel with an overflow tube covered with a small piece of rubber tubing, which may be pressed into the neck of the funnel, thus retaining sufficient solution for tests. This always represents the last liquid received, and can be allowed to enter the solution reservoir by releasing from the neck of the funnel, or it can be withdrawn if so desired. These large tubes are very susceptible to changes of temperature and should be protected. This can be done by pasting thick paper over them, allowing enough glass exposed for observation.


B
A
60"
E
A
D
C
Detail Upper Part of Tube and Reservoir of Solution
Overflow Tube
Detail lower part of tube
Ring or Rubber Bands
Filter Cloth

Before using the tube for experiment, it should be graduated. As the tubes are not always of regular diameter, it is well to mark them permanently at the points reached by successive additions of not more than 500 cc. of water. This provides a simple method for determining the specific gravity of ore charges, when using the tubes for experiments. In using the tubes the ore charge is introduced, preferably with a funnel, from the top. The feed regulator, *B*, is then inverted in the neck of the tube, and no further attention is required. It is well to bend the point of the discharge tube of the supply flask so that the liquid will not drop directly on top of the ore charge, but against the wall of the tube *A*.

After introducing the ore charge and solution as described, the time is noted. For this purpose an experiment card is provided for each apparatus, which it is best to hang conveniently at hand. This card should be provided with columns in which are entered all the amounts of liquid removed for tests, the total content of solution in chemicals, and the total content of the ore charge in gold and silver. I use 1 kg. of ore in most tests, and estimate the total milligrams of gold and silver contained. As soon as the solution is exhausted from the solution-reservoir, the speed of percolation can be noted. This can be taken as correct for similar depth of ore-beds, no matter what diameter the beds may have, and can be safely rendered as the speed of percolation for similar charges on a large scale. It is important to keep track of this detail during the time of the experiment. When the entire solution has passed through the ore charge, the amount recovered is measured (with allowance made for solution appropriated for tests), and the difference from the original amount charged to what is held by the ore. If wash water has been used, the amount held by the ore charge must be known, and its effect considered in increasing the quantity of solution, with its bearing on the cyanide strength in tests, also in precipitation for gold and silver extraction.

It is my custom to precipitate the gold and silver after the passage of each total quantity used, with a weighed amount of zinc dust, remove the precipitate by filtration, and reduce in the assay furnace, thus obtaining the actual metal leached, and introducing the effect of zinc found in solutions used in practice. These incomplete recoveries are compared to the original amount in the ore charge, their proportion to the remaining metal in the tube noted, and furnish an accurate comparison of the progress of the experiment. After a few tests a factor is found that will make them accurate enough to determine the progress of the work and indicate the point desired. The assays of the tailing remaining in the tube should be expected to check closely.

Where it is desired, a portion of ore charge can be removed by a sampling device. I use for this purpose a glass tube, attached to a rubber tube sufficiently long to permit its entrance from the top of the percolation tube. This is allowed to rest on top of the ore charge. By suction it will enter into and draw up the pulp. This can be withdrawn, with its contents, by pressing the rubber tube to prevent the entrance of air. By slipping a heavy rubber band over the filter-cloth of the percolation tube, to form an air-tight connection, the end may be introduced into a suction flask and the rate of filtration accelerated by an air-pump."—G. A. James, *Mining and Scientific Press*, Dec. 30, 1911, p. 839. (H. A. W.)

MINING.

The Use of Explosives.—"The United States Government issues from time to time bulletins of advice to miners. A specially useful one lately published had reference to the use of explosives. A local authority on this subject has prepared a list of "Don'ts" for miners as far as explosives are concerned; and they are printed below. Attention to them may prevent injury, and even loss of life, not only in the case of the miner, but the native workers in his charge, and for whose safety he is legally responsible.

More than half the number of accidents due to explosives are caused by carelessness and ignorance on the part of miners and natives.

1. Don't store anything with explosives.
2. Don't store detonators with other explosives.
3. Don't open a case of explosives with any implement except a copper or brass chisel, or a wooden wedge.
4. Don't store explosives in a hot or damp place.
5. Don't allow dirt or waste paper, etc., to accumulate round your boxes, and don't collect old gelatine, use it up at once.
6. Don't forget to look under and behind your explosive boxes at frequent intervals, to make sure that no explosives are lying there.
7. Don't make up your primers sooner than is necessary.
8 Don't cut that end of your fuse which is to go into the detonator, on the slant, cut it square.
9. Don't forget to shake the sawdust out of a detonator before putting it on the fuse.
10. Don't push the fuse hard into the detonator.
11. Don't use anything except your nippers to crimp the cap on to the fuse.
12. Don't force the cap into the gelatine when making a primer; make a hole first with the copper end of your nippers, or with a piece of stick.
13. Don't put the cap more than three-quarters of its length into the gelatine.
14. Don't forget to grease the cap at its junction with the fuse, if you are going to load a hole in a wet place.
15. Don't sit on your explosive box, or use it for capping up on.
16. Don't smoke when handling explosives.
17. Don't have a light close to your explosives, or don't hold or place your candle over your explosives.
18. Don't carry your light when you are carrying your primers, or explosives, but have another person to carry the light for you.
19. Don't handle explosives or detonators carelessly.
20. Don't spare your fuse, fuse is cheap. Don't roll or coil your fuse, but tuck it into the hole.
21. Don't forget to sludge out your holes before charging.
22. Don't force explosives into a hole, if they won't go down easily put a primer on and blast them as they stand.
23. Don't hammer your explosives home—squeeze them in with a steady pressure.
24. Don't press hard on the first cartridge of tamping.
25. Don't use more tamping than is necessary to make the hole air-tight.
26. Don't use anything other than a wooden stick as a tamping rod.
27. Don't forget that you are responsible for any accident that may happen to a native while he is lighting up for you, and don't forget that you must supervise him.
28. Don't try and put your "cheesa-stick" out, throw it away and let it burn out.
29. Don't stand talking to anyone with a burning "cheesa-stick," you are trying to gas yourself and him
30. Don't forget to guard the entrances to your working places at blasting time.
31. Don't forget to count your shots, and don't forget to report miss-fires to the proper persons.
32. Don't go back to your working place until the smoke has cleared.

33. Don't forget to turn the air on at blasting time.

34. Don't go back to a miss-fire under 30 minutes.

35. Don't enter a winze, raise, or drive, or back stope at the start of a shift, or after blasting without turning on the air, and let it blow for ten minutes before entering.

36. Don't drill in old holes on any account.

37. Don't drill on a bench where there is a miss-fire.

38. Don't drill a hole so that it can come in contact with an old socket.

39. Don't forget to plug all holes that will hold a plug.

40. Don't attempt to take explosives out of a hole.

41. Don't, when firing a missed hole, remove the tamping with a scraper, use water and a pointed stick.

42. Don't remove more tamping than is necessary to get another primer in."—*Mining Journal of the Mozambique Company*, Oct. 1911, p. 75. (H. A. W.)

Flushing or Silting Mine Workings.—"In the October number of *The Mining Magazine* (London), Mr. Edgar Pam, in an article entitled 'Water-Borne Packing for Stope-Filling' says: 'This process, commonly known as 'sand filling' was started in Silesia in 1901, and its value has been proved by the rapidity with which it has been introduced all over the mining world. * * Many collieries in Silesia, Westphalia, and France are employing this method of filling their workings; the majority of the mining groups on the Witwatersrand have adopted it, and from private information and current literature it appears that a large number of mines in Scotland and the United States are falling into line.'

The above statement, while not a direct claim, is a very strong inference that the filling of old workings with sand or other material flushed in with water originated in Europe. Such an inference, however, is at variance with facts.

In 1886, the workings of the Kohinoor colliery under the western part of the borough of Shenandoah, Pa., threatened to destroy the surface and the buildings thereon. The Mammoth seam at the Kohinoor colliery was in this locality normally from 50 ft. to 60 ft. thick, but owing to an overlap there was a bed of fine anthracite from 100 ft. to 120 ft. thick.

The Philadelphia & Reading Coal and Iron Co. owned the colliery, having purchased it several years previously from Messrs. R. Heckscher & Co. Most of the mining in the territory mentioned was done by the original owners, and the real extent of the mine openings was not shown on the mine maps, until a new and more accurate map was constructed from surveys by the engineers of the Philadelphia & Reading Coal and Iron Co. It was evident from this map and the fact that a geological cross-section showed less than 400 ft. of cover over the seam, that there was some danger of a heavy subsidence of the surface.

At this time (1886) the late R. C. Luther was Chief Mining Engineer for the Philadelphia & Reading Coal and Iron Co., and he conceived the idea of putting 8 in. bore holes from the surface to the higher points in the mine workings, and of flushing culm down these bore holes to fill up the very large cavities. There were large culm piles at the colliery, so scraper lines were constructed running from the culm piles to the bore holes, and water was pumped to them from a neighbouring creek. Men at the culm piles screened the culm before it was sent to the bore holes by the scrapers, and men at the bore holes fed it into the holes as fast as the water would carry it down. After the water drained off from the culm it flowed by gravity to the sump and was thence pumped to the surface. In course of time all the openings were filled, and the success of the idea has been proved by the fact that 25 years have elapsed since the work was done, and there has been no subsidence of the surface where the culm was silted into the workings.

The surveys and geological cross-sections, and the locations of the several bore holes required, were made under the supervision of Mr. John H. Pollard, then Resident Engineer of the Shenandoah District, and now Division Superintendent of the Mahanoy Division of the P. & R. C. & I. Co.

Thus, in 1886, fifteen years before the plan was adopted in Silesia, it was succesfully used in Pennsylvania. Some 2 or 3 years later Mr. James B. Davies, then superintendent of the Haddock collieries at Plymouth, Pa., knowing of the results of the work at the Kohinoor colliery, successfully used the same method, and he also antedated the Silesian engineers.

It is interesting to also record the fact that at the Kohinoor colliery the culm in the mine packed so solidly that gangways or headings were driven through it and the miners reported that it "cut like cheese." The fore-poling method of timbering was naturally used in these gangways and through them several hundred thousand tons of coal was mined from the larger pillars, and the new excavations were then filled with culm in the same way. The process was described in *Mines and Minerals*, then known as *The Colliery Engineer*, in the early part of 1888."—*Mines and Minerals*, Jan., 1912, p. 321. (A. R.)

Interim Report of the Chief Inspector of Explosives for the Transvaal and Natal Provinces, for six months ending 31st December, 1910.—*Manufacture.*—The British South African Explosives Co., at Modderfontein, manufactured 118,733·9 cases of explosives of 50 lb. each. Messrs. Kynoch, at Umbogintwini manufactured 103,565·4 cases, comprising Blasting Gelatine (82,679·2), E.O. Gelatine (3,652), Gelignite (16,720·2), and Arkite (514).

Imports of Explosives into the Transvaal.

	From oversea.	From Cape.	From Natal.
		Cases.	Cases.
Blasting Cartridges	—	137,970	105,176
Detonators No. 6 and 6D (number	18,787,500	—	—
,, No. 8 (number)	3,010,600	—	—
,, Electric (number)	105,000	—	—
Safety Fuse (cases of 500 coils).	10,328	—	—

The safety fuse came from Great Britain (6,700), Germany (2,746), and Belgium (880).

The imports into Natal were: Blasting Gelatine (5 cases) Gelignite (320 cases), Dynamite No. 1 (309 cases), Permitted Explosives (2,453 cases), and Blasting Powder (487 cases), Detonators No. 6 and 6D (220,000), Electric Detonators (300,000).

Storage.—152 magazine licenses are now in force.

Samples tested.—97 samples were tested at Johannesburg, and 55 in Natal. 67 samples of safety fuse were tested and 6 rejected."—Major A. B. Denne.—*Journal of the Society of Chemical Industry*, Jan. 15, 1912, p. 47. (J. G.)

STOPE-FILLING.—"I read with interest the able article on 'Water-Borne Packing for Stope-Filling' by Mr. Edgar Pam, but I noticed that the author stated in the first paragraph of his article that the process was started in Silesia in 1901, and also made further reference to its adoption in other mining countries. This would give the impression that the system originated in Silesia. That is not so. The idea is of American origin and was first introduced on the property of the Dobson Company at Plymouth, Pennsylvania, in 1891.

L. W. Mayer, in his book 'Mining Methods in Europe,' in the chapter on the flushing system, states: 'Not many years after the announcement of results at the Dobson colliery, German engineers investigated the field where the process was in successful operation and have since illustrated their approval of the practice by its universal introduction along most elaborate lines in Germany, where it continues to gain favour constantly.' "—L. A. WOMBLE, *Mining Magazine*, Jan. 1912, p. 61. (A. R.)

MISCELLANEOUS.

LABOUR SHORTAGE. — "I have been patiently waiting for the first signs of the solution of the labour problem in the columns of *The Farmer's Weekly*. Within the last two months, one or two correspondents have got hold of the clue, but it has not apparently impressed your numerous readers nor has it even given rise to a discussion. This is the more strange as, to any thinking person, the reintroduction of Chinese labourers for the Rand mines is a simple and effective solution.

I know, of course, that it may be a distasteful step for the Government to take as, unfortunately in the past, the Chinese labour question was mixed up with party politics. Is that, however, sufficient reason for the prohibition of this cheap and efficient labour when it can be proved that it is at once a simple remedy and economically sound? No one, I suppose, will deny that both the mining and agricultural industries are considerably hampered by the shortage of labour. Consequent upon this shortage comes the fact that both these and other industries are suffering from a too high scale of wages.

The position of the South African native appears unique in the industrial world. For the last 8 to 10 years he has dominated the situation. If he find his work uncongenial or the wages not to his liking, or if he merely elects to have a change of occupation, he finds hundreds of arms open to receive him. He will even find money advanced to him to go to certain places and all at a more or less generous scale of wages as compared with other countries with the same class of labour.

In one sense this, to me, is monopoly, and the simplest method of overcoming a monopoly is to introduce competition. Let the conditions be such for the native as the white man has to contend against, viz., that if he loses his job he may find some difficulty in finding another. Those are conditions that, under normal circumstances, obtain all over the world, so why not in South Africa with the native?

The Government, and for that matter perhaps, others than supporters of the Government, contend that there is sufficient labour in South Africa to supply all the wants of industrial expansion. Time and opportunity have been given to prove this contention, but still the shortage grows more acute, and now one hears clamours for some form of compulsion to get the native to come to work. To talk of compulsion now is to go back 25 years. The days of compulsion are past, except in so far as economic conditions can be brought about to keep an even balance between the industries and the native labourer, and at present the balance is a great deal too much in favour of the native.

The position in South Africa to-day is that the mining industry is a goose that is laying a very real golden egg, and it, therefore, must be tended.

A prosperous and active mining industry is of the most vital importance to South Africa. We are not yet in a position to have a large fruit export to take the place of gold, as in the case of California, nor are we so situated that our wool export could fill the gap, as perhaps could be done in Australia. We have still a long way to go before the produce of the land can take the place of the gold export. This means that the gold industry must be supported in so far as such support is not economically unsound. If a cheap and more reliable labour supply were available, some of the Witwatersrand mines, now on the verge of shutting down, may be able to keep going at a profit and so increase or, at least, maintain our great produce market.

Now for the farmer. The introduction of say, 80,000 Chinese labourers to be employed on mines outside Johannesburg town, east and west, would change the whole position for the farmer. Instead of looking vainly round for some assistance with his cattle, his horses, or his field of wheat, he would have a comparatively constant number of applicants for work.

When we had the maximum number of Chinese labourers on the Rand—in 1907-1908, I think—Kafir labour was quite plentiful among the farmers in the Lydenburg district, able-bodied natives offering themselves at 20s. per month.

There is another question of grave importance. I do not think anyone with a knowledge of compound life on the Rand will deny that it has anything but a deteriorating effect on the morals of the native.

I have often heard the question of Chinese labour discussed, but I have never heard really sound reasons for its prohibition. As far as the mines are concerned, I think I am right in saying that it is cheaper labour per ton mined than Kafir labour and, of course, far more reliable.

It has been repeatedly stated that it is dangerous to flood a place with foreign labour, but it is not preposterous to suppose that our legislators are incapable of making provision against any such labour remaining in the country when it is no longer wanted, or of efficiently supervising it while here? It is equally preposterous to suppose that our workmen's unions are not capable of watching any encroachment of such workers on what is looked upon as skilled labour.

As I have previously stated, the reintroduction of Chinese to the Rand is the one simple remedy of the universal South African labour trouble. The native commands the situation because, in most cases, he need only work a few months of the year to supply his few wants.

The industrial expansion of South Africa has made it acutely apparent that we have, at least, a most unreliable supply of labour, and the very natural solution of the difficulty is to supplement it. As it happens this can be done by supplying the Rand with Chinese—a cheap and efficient labour that can be minutely supervised—and thus leave to the country at large the thousands of natives now streaming to and from the Witwatersrand.

Let the farmers in all the Provinces think well before crying out against the re-introduction of Chinese labour. It will at once increase their market and give them all the labour they want."—J. EDGAR, *Farmer's Weekly*, Feb. 14, 1912, p. 975. (C. B. S.)

Reviews and New Books.

GEOLOGICAL AND TOPOGRAPHICAL MAPS, THEIR INTERPRETATION AND USE. By ARTHUR R. DWERRYHOUSE. Cloth octavo, 136 p., with many illustrations. (London: Edward Arnold). Price 4s. 6d.

"This book has been written by the author, who is the lecturer on geology in the Queen's University, Belfast, for the purpose of making clear the interpretation of geological and topographical maps. Though he particularly refers to the British official survey, his instructions are equally useful to civil and mining engineers and geologists in other countries. We have always felt that a book of this kind would be useful to the student in the field, for a map containing details of contours, outcrops, and dips can be made the basis of more deductive argument than is usually supposed except by those who have prepared them. The book also discusses the physical phenomena and mathematical problems in connection with dip, strike, faults."—*Mining Magazine*, Jan., 1912. (A. R.)

MINING WITHOUT TIMBER. By R. B. BRINSMADE. Cloth Octavo, 320 pp., with many illustrations. (New York: McGraw-Hill Book Co.) Price 12s. 6d.

"As the supply of timber in the United States is rapidly diminishing, the author has taken this opportunity of drawing the attention of American mining engineers to the various ways of winning ore without the use of timber. He starts with a description of surface work, usually called quarrying in this country, but called mining by steam shovel and glory-hole mining in America; then he passes to underground quarrying with underhand stoping as exemplified in the lead mines of Southeast Missouri, and in the Calumet and Arizona mine where the Mitchell modification is employed. He next describes shrinkage stoping, with and without waste filling, and gives particulars of practice at Wolverine, Homestake, Alaska Treadwell, and many other mines; then he quotes cases of overhand stoping on waste in Mexico, West Australia, and Broken Hill. Examples of pillar-caving are next given, the Miami and Boston Consolidated being of special interest. Back-caving, block-caving, and slicing systems are described, and the author then proceeds to deal with the methods adopted in coal mining."—*Mining Magazine*, Jan., 1912. (A. R.)

TYPES OF ORE DEPOSITS. By H. FOSTER BAIN, E. R. BUCKLEY, S. F. EMMONS, W. H. EMMONS, F. H. HATCH, OSCAR H. HERSHEY, J. D. IRVING, J. F. KEMP, A. C. LANE, C. K. LEITH, R. A. F. PENROSE, jun., T. A. RICKARD, and C. H. SMYTH, jun. Edited by H. Foster Bain. 378 pp., with index and 99 illustrations (maps, plans, sections, photographs). $2, post paid. 1911. (San Francisco: *Mining and Scientific Press.*)

"The papers on 'Native Copper Deposits,' by A. C. Lane, and on 'Replacement Ore-bodies,' by J. D. Irving, have been read before the Canadian Mining Institute. Those on 'The Treadwell Mines' (Alaska), by Oscar H. Hershey, on 'The Cobalt District, Ontario,' by S. F. Emmons, and on 'Outcrops of Ore-bodies,' by W. H. Emmons, have been published in the *Mining and Scientific Press*. That on 'Some Causes of Ore Shoots,' by R. A. F. Penrose, jun., appeared in *Economic Geology*. The remaining papers on 'The Clinton Type of Iron Ore Deposits' (C. H. Smyth, jun.), 'The Lake Superior Type of Iron Ore Deposits' (C. K. Leith). 'Flats and Pitches of the Wisconsin Lead and Zinc District' (H. Foster Bain), 'Lead and Zinc Deposits of the Ozark Region' (E. R. Buckley), 'The Saddle Reef' (T. A. Rickard), 'Contact Deposits' (James F. Kemp), 'The Conglomerates of the Witwatersrand' (F. H. Hatch), have been written specially for this volume. Other papers on these subjects already published in various survey reports and publications of mining and scientific institutions, have been drawn upon by the authors. Acknowledgment is made to these periodicals and institutions, and to the *Journal of Geology* for the loan of certain illustrations.

Space precludes a discussion of each paper, but it may be noted that Mr. S. F. Emmons regards the silver bonanzas of the Cobalt district as due to secondary enrichment, thus differing from Canadian geologists, who apparently regard them as formed from ascending solutions. This may have an important economic bearing on the future of the field. The gold in the Witwatersrand conglomerate is evidently still held by Dr. Hatch to be attributable solely to infiltration. Many of his readers, will, however, require weighty evidence against the placer theory, advocated by Becker, Gregory and others, before they can accept his conclusions, regarding the origin of all the gold in these conglomerates. "Replacement Ore-bodies" (Irving) is a masterly treatment of a most interesting subject, and it is well illustrated."

Allan, Horace. Metallurgical Manual of Iron and Steel; Their Structure, Constitution and Production. Cr. 8vo., pp. 374. *Technical Publishing Co.* Net 6s.

Bellasis, E. S. Hydraulics with Working Tables. 2nd Edition. 8vo., pp. 324. *Spon.* Net 12s.

Blount, B. The Institute of Chemistry of Great Britain and Ireland: Lectures on Cement. 8vo., pp. 64. *30, Bloomsbury Square, W.C.* Net 2s. 6d.

Brooks, A., and Others. Mineral Resources of Alaska. Report on Progress of Investigations in 1910 (Bull. 480, U.S. Geological Survey). 13 plates and map. 8vo., sd., pp. 333. *Wesley.* Net 5s.

Burstall, F. W. The Energy-Diagram for Gas. Royal 8vo. *Constable.* Diagram only, net 2s.; net 5s.

Cantlie, James. British Red Cross Society. First Aid Manual. No. 1. With 123 illustrations. 12mo., pp. xvi-219. *Cassell.* Net 1s.

Church's Laboratory Guide. Revised and largely rewritten by E. Kinch. 9th Edition. 12mo., pp. 368. *Gurney and Jackson.* Net 6s. 6d.

Elliott, C. G. Engineering for Land Drainage. 2nd Ed. Cr. 8vo. *Chapman & Hall.* Net 8s. 6d.

Faraday, Michael. Experimental Researches in Electricity. (Everymans' Library.) 12mo., pp. 358. *Dent.* Net 1s. Leather, Net 2s.

Fleming, J. A. Waves and Ripples in Water, Air and Aether. 2nd Issue. Revised. Cr. 8vo., pp. 312. *S.P.C.K.* Net 2s. 6d.

Gear, H. B., and Williams, P. F. American Electric Central Station Distribution Systems: their Design and Construction. 8vo. *Constable.* Net 12s. 6d.

Jamieson, Andrew. A Text-Book of Applied Mechanics. Vol. 4. Hydraulics, etc. 8th Ed. Thoroughly Revised and Enlarged Edition. Cr. 8vo., pp. 342. *C. Griffin.* 5s.

Jones, F. T., and Tatnall, R. R. Laboratory Problems in Physics to accompany Crew and Jones' "Elements of Physics." Cr. 8vo. *Macmillan.* 2s. 6d.

Journal of the Institute of Metals. Vol. 6. Edited by G. Shaw Scott. 8vo., pp. 369. Office.

Macdonald, G. W. Historical Papers on Modern Explosives. Roy. 8vo. *Whittaker.* Net 7s. 6d.

Mechanical Engineers' Price Book, The, 1912. Edited by Geoffrey Brooks. 12mo., pp. 184. *Spon.* Net 4s.

Minchin, William C. The Treatment, Prevention and Cure of Tuberculosis and Lupus with Allyl Sulphide. Cr. 8vo., pp. 100. *Bailliere.* Net 3s. 6d.

Modern Apprenticeships. Being a Scheme Recommended by the English Ruskin Society. Outlined by W. Lupton. Cr. 8vo. *English Ruskin Society.*

Morse, H. W. Storage Batteries. The Chemistry and Physics of the Lead Accumulator. Cr. 8vo. *Macmillan,* Net 6s. 6d.

Neilson, Robert M. The Steam Turbine. 4th Ed. Revised and enlarged. Re-issue with additions. 8vo., pp. 678. *Longmans.* Net 18s.

Prelini, Charles. Dredges and Dredging. Roy. 8vo., pp. 294. *C. Lockwood.* Net 12s. 6d.

Reid, George. Practical Sanitation. 16th Ed. Revised. Cr. 8vo., pp. 366. *C. Griffin.* 6s.

Savage, William G. Milk and Public Health. 8vo., pp. 478. *Macmillan.* Net 10s.

Sexton, A. H., and Primrose, J. S. G. The principles of Ironfounding (and Foundry Metallography). Cr. 8vo., pp. 338. *Technical Publishing Co.* Net 6s.

Shenton, H. C. H. Practical Sewerage and Sewerage Disposal. 12mo., pp. 148. *Rogers.* Net 3s. 6d.

Stocks, Herbert B. Water Analysis for Sanitary and Technical Purposes. Cr. 8vo., pp. 144. *C. Griffin.* Net 4s. 6d.

Text-Book on Rand Metallurgical Practice (a). Vol. 1. By Various Authors. 8vo., pp. 488. *C. Griffin.* Net 25s. Local publisher: *Fred. Rowland.*

Villamil, R. de. A B C of Hydrodynamics. 8vo., pp. 148. *Spon.* Net 6s.

Abstracts of Patent Applications.

(C.) 194/11. James Catford Koller. Improvements in Ore Concentrating Apparatus. 5.4.11.

This invention relates to an ore concentrating apparatus, which is constructed in the form of a cone and is provided with a spiral riffle.

The apparatus partakes of the nature of the old fashioned buddle, to which has been added a spiral riffle for the purpose of catching concentrate.

(C.) 222/11. Edmund Paul Millington Jerrard (1) and Paul Millington Jerrard (2). Improvements in clutches or similar devices for the transmission of power. 18.4.11.

This application relates to clutches for the transmission of power. The clutch described is primarly intended to be used with high speed machines, and the chief object aimed at is the placing of the load gradually on the driving machinery.

The apparatus consists of a cylindrical casing with hollow bosses or projections on opposite sides and having hollow trunnions as integral parts of the bosses on which the whole revolves. The driving shaft passes through one of the trunnions and terminates in the centre of the casing with a cross head of more or less T shape, on which is mounted a compound wedge which is arranged by means of a rod and lever to slide in a direction parallel to the axis of revolution. A disc is mounted on the compound wedge having a peripheral concentric groove in which a diaphragm, which divides the casing into two angular partitions, fits. The disc, which is of such dimensions that when in position in the casing it is in sliding contact with the sides of the latter, forms with the interior of the periphery of the casing an angular chamber, which is divided by the diaphragm into two equal angular compartments. Two abutments are provided one in each angular compartment, and are placed radially and diametrically opposite one another, and are kept in yielding contact with the rim of the disc by springs and serve as sliding partitions interrupting the two angular compartments. Channels are provided from above the abutments into the hollow bosses, as also on the sides of the cross head from one hollow boss to the other to allow of a fluid passing from one compartment to another. Openings are provided at various points to allow of the whole being filled with a fluid such as oil. On one of the trunnions is provided a pulley or geared wheel off which the motion is taken for the driven machinery.

It is claimed that when the disc is maintained concentric to the axis of the revolution, then the disc rotating concentrically in the angular oil filled compartments will practically transmit no torque to the casing, but, that if by movement of the compound wedge the disc is moved into a position of eccentricity to the axis of revolution that the movement of the oil will be such that it will re-act upon the face of the abutment in the angular compartment and produce the torque on the casing in the same direction as that of the revolution of the disc, the torque being gradually increased by the movement of the lever and rod operating the compound wedge until the position of extreme eccentricity of the disc is reached, the disc being then in firm contact with the casing.

A further method is described in which there are two discs operated by separate wedges which are mounted on the driving shaft which, in this case, is carried entirely through the casing and it is suggested that the shaft might have a sliding coupling so that it may be moved horizontally in order to operate the wedges, otherwise this method is the same as the previous one. In both methods described the power is transmitted through the clutch by belt or rope, etc., from one machine to the other. No method of direct coupling is shown.

(C.) 341/11. James Gitsham. Improved process for the recovery of gold and silver from antimonial, arsenical, and other gold bearing ores. 17.6.11.

This application is for a process patent, which consists of :—

1st. Water washing refractory arsenical and antimonial gold and silver ores in a leaching vat.

2nd. Then treating the ore with a cyanide solution to combine with the acid contained in the ore or acid added, to form hydrocyanic acid (which dissolves the gold and silver).

3rd. Drawing off the solution to an alkaline sump where a portion of the cyanide is regenerated.

4th. Passing the solution to another sump for strengthening with cyanide.

5th. Precipitating the gold and silver by standard methods.

(C.) 9/12. James Millar Neil. Process for the precipitation of precious metals from cyanide solution. 5.1.12.

This application is for an improvement in the existing processes for precipitating precious metals from cyanide solution, and consists in agitating the solution with a mixture of atomized metallic zinc and lead.

(C.) 13/12. Hugh MacKenzie Leslie. Improvements in the extraction of metals from their ores. 5.1.12.

This application relates to improvements in the cyanide process which consist in covering the vats, extractor boxes, etc., so as to prevent volatile cyanogen compounds from escaping into the atmosphere, and conducting them to suitable collecting vessels, such as gasometers, from which they can be forced back into the agitators or other vessels in which pulp is treated, thus forming what the applicant describes as a "closed system."

(C.) 14/12. Hugh MacKenzie Leslie. Improvements in the extraction of metals from their ores. 5.1.12.

This application closely resembles No. 13/12 by the same applicant, differing only by the fact that in this case, a sketchy description is given of a complete plant for the use of the "closed system" of saving the waste of cyanide that occurs through escape of volatile cyanogen compounds into the atmosphere.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 115/12. James S. Colbath (1), Wilbur Alson Hendryx (2). Improvements in ore classifiers. 23.2.12.

(C.) 116/12. Morris Columbus White (1), Otho Cromwell Duryea (2). Improvements in or relating to apparatus for heating compressed fluid for operating pneumatic or other mechanism. 23.2.12.

(C.) 117/12. Jonas Marcus Kylling. An improved process and apparatus which is worked by hand for separating gold and other nuggets or diamonds from rubble earth or sand and retaining same. 24.2.12.

(C.) 118/12. Harry Percy Hudd (1), Frans Heinrich Michelsen (2), Edmund William Hudd (3). A new and improved solution for preventing pneumatic tyres deflating when punctured. 24.2.12.

(C.) 119/12. Frederick James Belneman. Traction engine drive wheel. 24.2.12.

(P.) 120/12. James Arnest Hrosfield. Improvements in tube mill feeders. 24.2.12.

(P.) 122/12. Rose Basch (1), Samuel Basch (2). Improvements in and relating to elastic tyres for the wheels of vehicles. 24.2.12.

(P.) 123/12. Robert Hunter. Self locking arrangement for set pins, tap bolts and the like. 26.2.12.

(P.) 124/12. Edward George Clifford-Jones (1), Alfred Edward Barker (2), Lancelot Ussher (3). Apparatus for distributing material upon dumps. 26.2.12.

(P.) 125/12. Edward Silcock. A hingeless gate. 27.2.12.

(C.) 126/12. Francis Laurence Bosqui. Improvements in pulp distributors for amalgamating tables, plates and the like. 27.2.12.

(P.) 127/12. John Scott. Improvements in and relating to the hammer piston type of rock drill. 27.2.12.

(C.) 128/12. Raoul Diaz Wagner. Improvements in screw bolts, nuts, or the like for fixing rails on sleepers or for other purposes. 28.2.12.

(P.) 130/12. Ernest William Massam (1), William McVitty (2). Improvements in stamp guides. 28.2.12.

(P.) 131/12. Thomas Foulis (1), David Napier (2), Samuel Fox (3). Improvements in dust arrestors for rock drills. 29.2.12.

(P.) 133/12. Percy John Bolt. Improvements in apparatus for separating precious stones and precious or base metals from alluvial deposits. 3.3.12.

(C.) 134/12. Joseph Albert Hill. Improvements in and relating to furnace grates for locomotive and other boilers and other purposes. 2.3.12.

(C.) 135/12. Hans Charles Behr. Improvements relating to percussive apparatus. 2.3.12.

(P.) 139/12. Thomas Henry Chad. An apparatus for moulding igniters, or so called "tshisa" sticks, for blasting and other analogous purposes. 6.3.12.

(P.) 141/12. Rowland Owen Pickin. An improved device or tool for loosening and extracting rock drills and the like from bore holes. 8.3.12.

(P.) 142/12. Albert Thomas Harris (1), Charlton Effingham Wollaston (2). An improved spray for liquids. 8.3.12.

(P.) 143/12. William Cullen (1), Thomas Donaldson (2), The British S.A. Explosives Company, Limited (3). Improvements in explosives. 8.3.12.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

ANDERSON, A., *to* The Retreat, Que Que, Rhodesia.
CINDEL, F. W., *l/o* Johannesburg; Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.
HOLFORD, W. G., *l/o* Benoni ; c/o Rand Club, Johannesburg.
HORWOOD, C. D., *l/o* Roodepoort ; 39, Silverdale Terrace, Hanover Road, Mayfair, Johannesburg.
JONES, H. E., *l/o* Rhodesia ; The Lodge, Culmington Road, Eling, London W.
LEACH, G. W., *l/o* Cleveland ; P. O. Box 18, Parys.
MARGERISON, S. B., *l/o* Dutch East Indies ; M. M. Sintoeroe, Sambas, Dutch Borneo.
McLEAN, C. S., *l/o* Rhodesia ; Simmer and Jack G. M. Co., P. O. Box 192, Germiston.
PHILLIPS, J. M., *l/o* Potgietersrust ; 47, 6th Street, La Rochelle, Johannesburg.
SHARP, W. P. F., *to* 74, Willow Road, Germiston.
SMITH, W. J. W., *to* Broad Arrow Mine, Selukwe, Rhodesia.
STEELS, S. H., *l/o* Johannesburg ; Geldenhuis Deep, Ltd , P. O. Box 54, Cleveland.
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THE JOURNAL

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Vol. XII. APRIL, 1912. No. 10.

Proceedings

AT

Ordinary General Meeting, April 20th, 1912.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, on Saturday, April 20th, Mr. W. R. Dowling (Vice-President) in the chair. There were also present :—

39 Members : Messrs. R. Allen, Dr. W. A. Caldecott, A. F. Crosse, J. Gray, J. Littlejohn, Prof. G. H. Stanley, J. E. Thomas, F. W. Watson, H. A. White (Members of Council), W. Beaver, J. Cellier, J. Chilton, F. W. Cindel, L. L. Ellis, R. Gascoyne, J. H. D. James, A. J. Johnson, G. A. Lawson, J. Lea, H Lea, P. T. Morrisby, M. T. Murray, W. Nicklin, W. H. Odgers, A. C. L. Olsen, J. M. Phillips, W. S. V. Price, J. F. Pyles, W. H. Reid, E. Roberts, G. A. Robertson, A. Rogers, A. L. Spoor, A. R. Stacpoole, A. Thomas, Prof. G R. Thompson, K. Tonnesen, and F. Wartenweiler.

26 Associates and Students : Dr. J. L. Aymard, Messrs. E. G. Baskett, G. Bulman, E. Burrows, J. Cronin, C. L. Dewar, I. Dicks, F. E. Doble, O. A. Gerber, M. Green, T. D. Hall, W. Human, W. J. R. Hunter, R. W. Irvin, H. M. Jeudwine, G. J. Mackay, G. F. Matthews, F. Millar, D. D. Nicholas, G. S. Paterson, R. M. Phillips, F. J. Pooler, H. B. Powter, V. E. Robinson, H. Stadler, and A. M. Thomas.

11 Visitors, and Fred. Rowland, Secretary.

MINUTES.

The minutes of both the February and March meetings, as published in the *Journal* for February and March, were confirmed.

Messrs. P. T. Morrisby and J. F. Pyles were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been elected as follows :—

NEW MEMBERS.

DINGWALL, JAMES, Modder "B" G.M. Co., Ltd., P. O. Box 14, Brakpan. Underground Manager.

FROUDE, WILLIAM GODFREY, P. O. Box 40, Pilgrims Rest. Mine Manager.

GERDS, FREDERICK NIVEN, A.R.S.M., Cam and Motor G.M. Co., Ltd., Eiffel Flats, Gatooma, Rhodesia. Mining Engineer.

JOHNSTONE, JOSEPH WILLIAM, Government G.M. Areas, Ltd., P. O. Box 28, Brakpan. Mine Overseer.

The Secretary notified that since the last meeting of the Society the following Associates had been admitted by the Council :—

As Associates—

ARMSTRONG, ERROL D., Bodangora, New South Wales. Assayer.

MANN, CHARLES JOHN ROBERT, Poerna G M. Co., Ltd., Poerna, Westland, New Zealand. Metallurgist.

MOYNIHAN, EUGENE JOSEPH, P. O. Box 2061, Johannesburg. Consulting Engineer.

ORNSTEIN, ABRAM, Cinderella Consolidated G.M. Co., Ltd., P. O. Box 225, Boksburg. Miner.

TRAVERS, JOHN WOLSELEY, Knight Central, Ltd., P. O. Box 91, Germiston. Mill Manager.

YUILL, HARRY HOGG, Crown Mines, Ltd., P. O. Box 103, Johannesburg. Mine Surveyor.

GENERAL BUSINESS.

ANNUAL DINNER.

The Secretary announced that the annual dinner would be held on Saturday, 11th May, at the Grand National Hotel, the tickets for which cost one guinea and could be obtained from any member of the Council or from himself

OBITUARY

The Chairman : I regret to have to record the deaths of Messrs. E. Blume, one of our members, and H. Forbes Julian, the well known metallurgist and author who went down in the *Titanic.* Both these gentlemen were well known to you. I would ask you to signify your sympathy by rising. All members rose from their seats.

NOTES ON COAL FROM WANKIE COLLIERY, SOUTHERN RHODESIA.

Mr. Jas. Gray, F.I.C. (*Member of Council*): Though large deposits of coal exist in Southern Rhodesia, the only coal mined for sale is that obtained at Wankie Colliery and until recently I had not seen any analyses of this coal on record. However, in the *Rhodesian Mining Review* for September 6, 1911, p. 383, analyses of Wankie coal are given by John Hynd, which show it to be superior to the best Transvaal coal and certain analyses which I have made recently confirm this.

My samples were taken from a consignment of household coal purchased in Salisbury and separated as follows:—

No. 1 round coal.
No. 2 nut coal.
No. 3 large dull pieces with greyish colour.

The results of analysis were:—

	No. 1.	No. 2.	No. 3.
Moisture	0·71%	0·23%	0·57%
Ash	10·11%	12·46%	15·31%
Volatile matter ...	22·96%	25·32%	22·02%
Fixed carbon ...	66·22%	61·99%	62·10%
Total sulphur ...	1·20%	2·02%	0·60%
Evaporative power in lb./lb. Mahler Bomb	13·78	13·64	13·00

These figures show the coal to be superior in ash and evaporative power to the best Middelburg (Transvaal) coal.

Though the samples do not show an extremely high percentage of sulphur, yet I have seen pieces of coal which have innumerable small nodules of pyrite, and it is not uncommon to pick out lumps of iron pyrite up to several inches in diameter.

DETERMINATION OF CARBON DIOXIDE.

Mr. Edw. H. Croghan (*Member*): In the January *Journal*, Dr. Moir submits a revised edition of his shaking bottle method of testing mine air for CO_2. This revised method indicates whether the air contains under or over 0·20% CO_2, as distinct from the original standard of 0·15%.

Some years ago I made a few experimental comparisons in one of the mines between the original method and that of the barium-hydrate-oxalic acid procedure. The capacity of the bottles used in this instance for the latter method was 560 cc. and for that of Dr. Moir's 195 cc., *i.e.* the barium method was done on a smaller scale than usual. 50 cc. of the BaO_2H_2 (25 cc. = 6 cc. $C_2O_4H_2$ of which 1 cc. = ·001 grm. CO_2), and 10 cc. of the freshly prepared M/1000 pink Na_2CO_3 were respectively employed.

Both solutions were previously measured out in the laboratory—the former being retained in parting flasks with necks cut short and fitted into a rubber cork, thus allowing the flask to act as the stopper and minimising any exposure of the BaO_2H_2 to air; and the latter in small glass phials, rubber stoppered, and just capable of holding the 10 cc. which were transferred to the sample bottle by quickly pouring.

In accordance with the method, I shook and noted the following:—

(1) the colour disappeared in 2 minutes = 0·15% CO_2
(2) the colour disappeared in 1 minute = 0·28% CO_2
(3) and (4) half the colour disappeared in 2 minutes = 0·10% CO_2.

It was sometimes possible to note these differences under mining conditions!

In order to obtain more definite figures I brought these *same* solutions back to the laboratory to compare them with standards made from the M/1000 pink Na_2CO_3. The time interval before being able to make this comparison was as far as I can remember, about three hours.

The standards, kept in very small glass phials, rubber stoppered, were made by simple dilution with distilled water, the series being as follows:—

·02%, ·04%, ·06%, ·08%, ·10%, ·11%, ·12%, ·13%, ·14%, ·15%.

The samples were collected from one of the mines on the 13th level, at the places mentioned:

Place.	Dr. Moir's. % CO_2	Baryta-Oxalic. % CO_2
(1) Main Reef Leader	0·14	0·157
(2) Dead End ...	0·29	0·310
(3) Upcast shaft ...	0·12	0·124
(4) ,, ,, ...	0·12	0·124

The two samples from the upcast gave identical results which seems too good to be true—anyway it is a very good coincidence! In the case of No. 2, all colour on a 10 cc. basis had disappeared, and so I added another 10 cc. at the laboratory and shook for 2 minutes—the result is rather interesting. The number of samples taken is very small, but still this little comparison may prove interesting at the moment.

Dr. Moir states that the working solution only keeps half a day, and I find with reference to this that I made the following notes:—

(1) When the standards are kept well corked, away from light (in this case a wooden covered box was used) the colour remains constant for three days and longer. This was judged by comparison with new standards.

(2) The 10 cc. M/1000 pink Na_2CO_3, contained in the small phials were quite as good after 24 hours if similarly kept as in (1). These were also tested by comparison with the freshly made solution.

I am not sure whether Dr. Moir tried standards in the first instance or whether it was my idea, but it seems to me they might deserve some consideration. A series of 10 are made in a few minutes, employing two burettes, and there seems to be no serious objection why the method might not be proved colorimetrically quite good within certain limits.

Dr. J. Moir (*Past-President*): I have to thank Mr. Croghan for the interesting confirmatory data on this test which he provides.

As a matter of fact, however, I am not satisfied with the present form of the test. It was actually more exact under the old (1906) conditions; and my attempt to standardise it has, I am sorry to say, not improved it. By using strong phenolphthalein I have introduced a new large minus error, apart from the tendency to over-estimate the percentage of colour left. The allowance of 5% for incomplete reaction is also too small. I therefore recommend those who are using the test to cut the phenolphthalein strength down to about $\frac{1}{50}$ of that suggested, *i.e.* use 3 drops of the specified solution instead of 6 cc. In addition the "dose" of test solution should be diminished to 11·5 cc. (instead of 12·9). Under these circumstances the test seems to give correct results. The test could be made equal to the baryta method if a new indicator, of lower acid function than phenolphthalein, could be found. I have tried the following, which are of the required type: Alizarine, oxyazobenzene, paranitrophenol, benzeneazo-α-naphthol, phenacetoline, and methyl-red: but with no advantage.

Mr. Jas. Gray, F.I.C. (*Member of Council*): Dr. Moir's intention in originally working out his method was, I take it, to arrive at a means whereby a layman would be able to determine the percentage of CO_2 in mine air with a moderate degree of accuracy and I think he has been successful in his purpose. It would appear, however, that Mr. Croghan desires to raise the method to the dignity of being capable of competing with the well-known Pettenkofer process, with the sacrifice of simplicity, and thus render it unsuitable in the hands of a layman. My opinion is that the introduction of these colorimetric standards will render the method very cumbersome and still fail to attain the accuracy desired and therefore, in cases where a high degree of accuracy is required, Pettenkofer's method should be adopted. It is quite easy for one man, provided he has native assistants to carry the necessary paraphernalia about to take from 15 to 20 samples of air in the course of a morning's work and the remaining part of the work, that is, the addition of barium hydrate, etc., could be done on the surface, for if we except the arithmetical part of the work, there is nothing in Pettenkofer's method beyond the powers of the ordinary layman.

REGENERATION OF CYANIDE SOLUTIONS.

The Chairman: Mr. W. D. Williamson of New South Wales has sent us a contribution through Mr. J. S. McArthur, who has forwarded it to the Society through Dr. Caldecott.

Mr. W. D. Williamson (*Member*): In the November, 1911 issue of the Society's *Journal*, page 205, there is a reprint of an article on the above subject by R. P. Wheeloch. The description of the process, which is said to be patented, so closely resembles a process for which I obtained "British provisional protection" in 1903, that a description of my *modus operandi* may have some interest to members.

In 1899, while treating cupriferous tailing in North Queensland, the excessive cyanide consumption, due to the presence of copper carbonates, made its profitable treatment exceedingly difficult. Solutions rapidly fouled, precipitation was very unsatisfactory, extraction decreased until the only alternative was to discard the fouled solution. and make up fresh solution. Changes of solution were very frequent and involved considerable expense and loss of time. This trouble was finally overcome by preliminary leaching with sulphuric acid, about 12 lb. of acid being used per ton of tailing treated. Later, when sulphide ore had to be treated, sulphuric acid leaching was useless.

Experiments made on solutions, which had become charged with copper, demonstrated the fact that a very considerable saving of the combined cyanide could be effected by the addition of sulphuric acid to the solution, in sufficient quantity to precipitate all the copper as $Cu_2\,Cy_2$, viz. :—

$$Cu_2Cy_22KCN + H_2SO_4 = Cu_2Cy_2 + K_2SO_4 + 2HCN$$

The insoluble Cu_2Cy_2 was separated by filtration, and the clear filtrate containing the HCN neutralised with a slight excess of alkali.

The experiments were sufficiently encouraging to induce me to continue the method on a working scale. As solutions became overcharged with dissolved copper, sulphuric acid was added to the sumps in sufficient quantity to complete the above reaction. The precipitated Cu_2Cy_2 was settled, and the clear supernatant liquor s drawn off, and neutralised in a separate sump, and was then used as a fresh clean cyanide solution. The settled Cu_2Cy_2 was collected, and saved for shipment to smelter. The possible recovery of the cyanogen in combination with the copper as Cu_2Cy_2, which under the above method was palpably lost, still gave me material for experiment.

Heating with acid as mentioned in Mr. Wheeloch's paper I consider impracticable, as it would involve distillation of the liberated HCN, and probably excessive loss of cyanogen owing to the decomposition of the cyanogen radical in the presence of heat and strong mineral acids. Eventually, I found that an almost complete recovery of the cyanogen in Cu_2Cy_2, could be obtained by suspending the Cu_2Cy_2 in water, and passing sulphuretted hydrogen when the following reaction takes place,

$$Cu_2Cy_2 + H_2S = Cu_2S + 2HCy.$$

The insoluble Cu_2S was separated by filtration and the clear filtrate containing the HCN neutralised with alkali.

Finally, I combined the two operations by adding to the cupriferous cyanide solution sufficient Na_2S, to precipitate all the copper present, afterwards adding the requisite amount of sulphuric acid to complete the following reaction :—

$$2KCN{\cdot}Cu_2Cy_2 + Na_2S + 2H_2SO_4 =$$
$$Cu_2S + 4HCN + Na_2SO_4 + K_2SO_4$$

The precipitated Cu_2S carries down with it the precious metals contained in the solution, and is collected, dried, and shipped to smelter. The clear liquor is decanted or filtered, and sufficient alkali added to combine with the free HCN liberated by the previous treatment.

The above described process, which theoretically should recover all the cyanogen in combination in cupriferous cyanide solutions, has never been used to my knowledge on a working scale, but in laboratory experiments as high as 90% of the combined cyanogen has been regenerated, and made available as a clean copper free cyanide solution.

Mr. Andrew F. Crosse (*Past-President*): This paper has interested me very much indeed. As some of you may know, some years ago I was engaged in investigations in the cyanide treatment of cupriferous ore near Pilgrim's Rest. During my experiments I found that the solutions were always highly charged with copper, and I was unaware at that time—as many of us often are when doing work of this kind—that anyone else had tumbled to the idea I had, namely, the treatment of cupriferous cyanide solutions with sulphuric acid. I made many experiments. It was difficult at first to find out exactly the right amount of acid required. Then I made a long series of experiments to find out how much gold and how much silver was precipitated. I never obtained 100% of gold by precipitation; I used to get, if I remember rightly, 88% to 97%. I think that nearly all the silver was precipitated, but not quite all the copper. It was a very nice precipitate to handle; it would settle very easily. I treated over a ton of solution in this way. I then regenerated the hydrocyanic acid solution, after decantation, by means of caustic soda.

The second part of Mr. Williamson's paper is certainly an improvement on what I did. I simply took the precipitate, dried it, calcined it and melted it, and I got a little bar containing about 95% of copper, 1% of gold, and 2% of silver; but I certainly think that where sulphuric acid can be obtained fairly cheaply, and where a certain amount of cyanide is decomposed by copper in the ore, that the method is worth considering. Supposing some gold does remain in solution, the solution is simply passed through the zinc box after regenerating with caustic soda. I was also silly enough to give a few pounds to the Government to obtain a provisional patent, but I dropped it directly afterwards.

PRIZES TO STUDENTS.

Mr. Tom Johnson (*Member of Council*): Some time ago this Society offered a prize of ten guineas for the best paper by a student and also a prize of three guineas for the best contribution to the discussion of any paper read before the Society submitted by a student. I have not noticed any rush of students to compete for these prizes; it may be that the students do not care to take the money from the Society, possibly they are rather shy of coming forward, or they cannot pick on a suitable subject. If the latter is the case, I beg to offer the following list in my own branch of work for their consideration:

- Prevention of any specific kind of accidents.
- Rescue apparatus, in relation to fires, gassing, etc.
- First aid or ambulance work.
- Hoisting—Electricity *versus* Steam.
  - Influence of shaft bends in hoisting.
  - Comparisons of hoisting on the Rand and other parts of the world.
  - Single and double stage hoisting in deep shafts.
  - Changing single compound shaft hoisting to double stage hoisting.
- Haulage—Manual, animal, mechanical.
- Method of clearing rock from stopes, drives, etc.
  - Shakers, ground chutes, water chutes, gravity planes, engine planes, conveyors, etc.
- Relative advantages of running incline shafts and drives on the reef or under the reef.
- Leaving pillars *versus* no pillars but other supports.
- Sandfilling.
- Examples of difficult work done.
- General ventilation of mines.
- Rectangular shafts *versus* round shafts.
- Large reciprocating rock drilling machines *versus* small.

Reciprocating drilling machines *versus* hammer machines.
Machines *versus* hammer boys.
Examples of pumping installations.
Relative advantages of overhand, underhand, combined, breast and back stoping.
Relative advantages of different kinds of shaft boxes.
Relative advantages of different systems of payment to miners.

I am sure if our students would only take their courage in both hands, they could amongst them give several good papers on some of the above subjects which would be interesting and informative. There must be many examples of work going on in some parts of the Rand that those in other parts of the Rand are not aware of, but would be glad to know, and short notes or papers, if not by the students, by other members would be welcome. Our students should remember that they should, as a duty, undertake a share of the work of the Society, as it is to them we look for progress and propagation of new ideas. I would personally appeal to them not to be afraid of being criticised, as that is the fate of all of us, and it is the best thing possible to make one endeavour to be quite sure of one's subject.

STANDARDIZATION, ETC.

Mr. H. Stadler (*Associate*) brought forward a scheme for standardising grading of crushed material and proposed that a committee be appointed to look into the matter and bring it to the notice of members. As there was no seconder the matter dropped, after the Chairman had explained to the meeting that Mr. Stadler had already submitted his ideas to the Council and that the latter was considering the matter.

Mr. Stadler also read a letter complaining of the rejection by the Council of the paper entitled "The New Metallurgy and the East Rand Proprietary Mines."

Mr. J. E. Thomas, speaking as a member of the Publications' Committee, stated that Mr. Stadler stipulated that the paper must be published as written, and that the Committee acted in the interests of the Society in recommending the rejection of the paper.

The Chairman: As there is no further discussion, I might mention in connection with Mr. Stadler's grievance that this paper in the usual course was considered by the Publications' Committee and was unanimously rejected by that Committee. Subsequently this Committee reported to the Council, and the Council were unanimous in supporting the action of the Publications' Committee. They considered that the interests of the Society would be best served by not publishing this paper. This paper was subsequently published in the daily Press by Mr. Stadler, and no doubt members had an opportunity of reading it. I do not think that there is any more to say on the subject, and the matter may be considered closed.

THE ACTION OF MINERAL SULPHATES AND ARSENATES ON CYANIDE SOLUTIONS.

By Andrew F. Crosse (*Past-President*).

It has long been known to most of you that both sulphate of magnesia, and sulphate of lime, decompose cyanide solutions; however I think that it will prove interesting to raise a discussion on the subject, as we have had few chemical discussions of late.

The mineral kieserite is not rare, and has the following composition $MgSO_4 + H_2O$, and according to "Comey's Dictionary of Chemical Solubilities" is but slowly dissolved in cold water.

Hydrated sulphate of calcium or gypsum is also common, but it interests us more because sulphate of calcium is produced when the acidity in our tailing and slime is neutralised with lime, and I have also found alumina taken up in cyanide solutions, so that probably some aluminium sulphate or double sulphate was in the ore. The first point I would like to mention is that an ordinary analysis of a mineral or sample of the ore to be treated by the cyanide process is nearly valueless.

For instance take a common example—a certain ore contains iron, sulphur, magnesium, etc., etc., according to the analysis. This analysis is no help to the cyanide manager. What the he wants to know is, what will take place when the finely crushed ore comes in contact with an ordinary cyanide solution? The Mount Morgan ore near Barberton is a good example. I have seen analyses of this ore but as far as I can remember, none of them showed any sulphate of magnesium; magnesia was mentioned and so was sulphur, and as this ore is highly pyritic, I did not think of the presence of sulphate of magnesia, until I began treating the ore. I found on making a preliminary test, that the ore, digested in warm water, gave a neutral reaction. In my first experiments I used an ordinary amount of caustic lime, and found the protective alkali gradually disappearing and eventually free HCy was being given off. Then I digested some of the finely ground ore in warm water, and found magnesium sulphate in the solution. I determined the sulphuric anhydride in solution, but I found out in subsequent experiments that more SO_3 was taken up when using carbonate of soda solution, or cyanide solution. I consider the best method to determine the amount of soluble sulphates present in an ore, is to take 100 gm. of finely ground ore, and shake it up with 500 cc. of warm water containing say 2 gm. of carbonate of sodium of known percentage of Na_2CO_3. I prefer to leave the mixture all night and shake it up again next morning and then titrate the Na_2CO_3 left, taking 50 cc. of the filtered solution, and the amount of carbonate of sodium used per ton can be easily calculated. In the case of the Mount Morgan ore it was 10 lb. per ton of ore, but the same ore after treating with warm water only required 4·4 lb. of Na_2CO_3 per ton of ore to precipitate the $MgCO_3$ and $CaCO_3$ from the aqueous solution, so that even if the ore were washed, it would still decompose a cyanide solution, therefore carbonate of sodium is necessary. This is quite an important point, for if the sulphates were easily removed by water, this would take place during crushing in the mill.

There is a peculiar banket found at the Machavie mine on the Black Reef—between Potchefstroom and Klerksdorp. The pyrite is in a globular form, and there is a large amount of graphite in this ore (the Mount Morgan ore also contains graphite). Some of the tailing from that mine after washing four times decomposed cyanide of potassium (without using protective alkali) at the rate of 0·64 lb. per ton. The unwashed sand however, under the same treatment, decomposed 3·2 lb. of cyanide of potassium per ton of ore. This ore contained sulphate of magnesium.

I made a long series of experiments with various proportions of CaO in solution (lime water) added to a mixture of equal parts of decinormal pure cyanide of potassium solution, and decinormal sulphate of magnesia. This seemed very nice and simple, and I thought that I should get some definite results. There was a time not so very long ago, when it was imagined that all chemical reactions could be nicely expressed in equations, but unfortunately the more we probe into the mysteries of the physical world, the more we find that nature does not arrange things to suit our preconceived ideas. A nice equation is the following:—

$$2KCy + MgSO_4 + 2H_2O = 2HCy + MgH_2O_2 + K_2SO_4.$$

When equal portions of the decinormal solutions of the above salts are mixed, a precipitate of magnesium hydrate is formed, and free hydrocyanic acid is given off, but I doubt whether the reaction is exactly as complete as the equation makes out. I thought that if I added lime water, gradually increasing the amount till no more hydrocyanic acid was liberated (when tested by a drop of nitrate of silver solution placed on the lower side of a glass cover on a beaker) I should arrive at some definite result, but as sulphate of calcium was formed, which also dissociates cyanide of sodium or potassium, it made things complicated. Again as more CaO solution was added, weakening the cyanide of potassium solution, hydrolysis set up. I found however that all the magnesium became precipitated as a hydrate. Then I also found that free hydro cyanic acid was given off notwithstanding that using the ordinary test, protective alkalinity as CaO was present. Then again I am not satisfied in my own mind, that when a cyanide solution containing sulphates of lime or magnesia is tested by a nitrate of silver solution, that one obtains a correct result. The solutions I used were pure; but of course in practice one has to do with solutions containing zinc, which complicates matters still further. The ordinary practical man will shrug his shoulders at all this, but after all, the question is how to get over the decomposing action of sulphate of magnesium in the cheapest way. Carbonate of sodium might be found useful. No caustic soda would be formed, as carbonate of magnesium and sulphate of sodium would be produced. Perhaps some of the older students in our South African School of Mines might make an investigation into some of these reactions, using first pure chemicals, and then working with cyanide solutions obtained from the mines.

Scorodite or ferric arsenate is not a very rare mineral. It occurs often in conjunction with limonite, a hydrated ferric oxide. Probably these minerals when found together, are the result of the oxidation of arsenical pyrite. Scorodite has the formula $FeAsO_4 + 2H_2O$. According to Comey's "Dictionary of Chemical Solubilities," it is insoluble in water, but soluble in weak hydrochloric acid. SO_2 in solution also acts on it. When a finely ground mineral containing scorodite is left in contact with a weak caustic soda solution, arsenate of sodium is slowly formed. Cyanide of sodium or potassium acts in the same way, this accounts for the slow and continuous decomposition of cyanide solutions when treating ores containing arsenates. The solution becomes charged with arsenates which are reduced in the zinc box, and the precipitated gold carries arsenic, which is not a very pleasant thing to have in the sulphuric acid treatment, as arseniuretted hydrogen a very poisonous gas is evolved.

I have made many experiments in order to discover a method of overcoming the injurious action of arsenates on cyanide solutions, and I have found out how to counteract the solution of arsenate in a cyanide solution, by the following method:—A solution of arsenate of sodium and calcium hydrate give the following reaction

$$2Na_3AsO_4 + 3CaO + 3H_2O = Ca_3(AsO_4)_2 + 6NaHO.$$

The calcium arsenate is insoluble, and the sodium hydrate is regenerated.

I ascertained by a series of experiments, that using an excess of lime and a small quantity of sodium hydrate, all the arsenate that was possible to dissolve was taken up by the sodium hydrate, then precipitated as calcium arsenate, no arsenic being left in the solution. The cyanide of sodium was unaffected by this reaction. The method I used in order to determine the amount of lime required, was as follows:—200 grams of finely powdered ore were shaken up every now and then, for seven or eight hours with 200 c.c. of 1% NaHO solution, the amount of NaHO used up was determined in the usual way, and for every pound of NaHO required per ton, 0·7 of a pound of CaO is required in actual practice, but of course some sodium hydrate must be present in addition to act as a solvent or carrier.

For one sample of ore I treated, which contained scorodite, 13 pounds of sodium hydrate were required per ton of ore; by this simple method I prevented the abnormal decomposition of the cyanide, and also prevented arsenic going into solution; and regulating the proper proportion of lime, enough was left in solution to cause perfect settlement of the slime.

It will be evident that ores containing soluble sulphates or arsenates may require a large amount of lime per ton in order to settle the slime; in the case of arsenates if there is no sodium hydrate added, the sodium in the cyanide will effect solution of arsenate, and if only a small amount of lime is used, every trace of lime will be precipitated as an arsenate of calcium, and the cyanide of sodium will be decomposed.

Roasted concentrate originally containing $MgSO_4$ and $CaSO_4$ still contains these salts after roasting, and an arsenical concentrate after roasting will possibly contain some ferric arsenate.

An interesting matter for investigation would be whether if there were organic matter present in a sulphate of lime solution, sulphuretted hydrogen might be formed and coat the gold with a thin layer of sulphide; this is merely however an idea.

I hope that these remarks will cause a certain amount of discussion, which will prove valuable, more especially to some of our members in the outside districts.

Mr. H. A. White (*Vice-President*): I should like to move a very hearty vote of thanks to Mr. Crosse for this very interesting paper. As illustrating one point that he makes I would like to mention that at the Geduld Mine, in the dolomite area, although the water pumped from the mine is alkaline, yet a large amount of lime—much more than is usual on most mines on the Rand—has to be added in order to cause settlement of the collected slime. This is due to the fact that the water carries a small amount of magnesium salts in solution (a saturated solution of magnesium carbonate contains 0·022% of $MgCO_3$) and the lime has no effect in increasing the alkalinity until all the magnesia is first precipitated as hydroxide.

Another interesting point is raised by Mr. Crosse in reference to the fact that our working solutions, particularly from the slime plant, contain large amounts of calcium sulphate. It is obvious that if these solutions were neutral to phenol-phthalein we might yet have serious loss of cyanide strength owing to the effect Mr. Crosse refers to. The interesting point, which requires clearing up, is what amount of protective alkali is required to neutralise the acidic effect of calcium sulphate when present. As far as may be deduced from leaching effects, when the value in gold of solution leaving a vat is traced from hour to hour, it appears to me that evidence of re-precipitation of gold is obtained in some instances when the solution is allowed to fall below an alkalinity of 0·007% NaHO. It may of course depend on the concentration of $CaSO_4$, but I have never yet observed any evil effects when the alkalinity is as high as 0·008% NaHO.

THE "LUMINATOR" PURIFICATION TREATMENT OF WATER FOR STEAM BOILERS.

(*Read at March Meeting, 1912.*)

By William Cullen (Past-President).

DISCUSSION.

Mr. Jas. Gray, F.I.C. (*Member of Council*): This paper brings before us a problem which has caused much discussion in other countries, and although many of us may feel somewhat sceptical regarding effecting an improvement in boiler-feed waters by passage over corrugated aluminium plates, yet there are so many well-authenticated cases where an improvement has been observed through this procedure that it must be something more than charlatanism.

Much information regarding the "luminator" is given in the papers referred to by Mr. Cullen, but I would point out that his references are wrong. Duggan's paper appeared in vol. 30 and Lake's paper in vol. 31 of the "*Journal of the*

Society of Chemical Industry." The generally accepted opinion as far as I can gather from these papers and the discussion on them is that in the passage of the water over aluminium plates an insignificant quantity of aluminium is removed in a colloidal form and acting as a series of nuclei causes the insoluble products to separate as colloids and remain in suspension in the boiler instead of forming a scale as usual. Much the same effect is obtained on adding zinc slugs to boilers and the action is possibly similar. The author's experiments are unsatisfactory and no results of any value could reasonably be expected from them, as rate of evaporation and transference would tend to complicate matters. However, we are grateful to him for giving us an opportunity to discuss this matter, although it may be considered by many that Mr. White has said all there is to be said on the subject.

Mr. F. W. Watson *(Member of Council)*: The author has raised a subject which I am afraid has not had the attention it deserves from the members of this Society, I refer to the treatment of boiler feed water. The "Luminator" treatment is one of these mysteries which both the author and Mr. H. A. White have aptly compared to the Divining Rod and no explanation is given of the resulting changes. The author has referred to the "*Journal of the Society of Chemical Industry*" which contains several articles on the question, and the latest I have seen is by Mr. H. B. Lake in January of this year.[*] His conclusions are that "the process seems to be of little value unless the resulting acid gases and sludge can be removed which appear to cause corrosion and overheating of the plates with subsequent 'pitting' and bulging where the sludge is excessive."

The analytical figures quoted by the author, cannot be classed as numerous and as he himself states are incomplete. It would have been interesting if he had given us a series of complete analyses of the water at the following stages: (1) Before passing over the luminator; (2) After passing over the luminator; (3) From the hotwell; (4) From the feed pumps; (5) From the boiler in use; (6) The blowdown water. Further, he might have given us complete analyses of the various scales. It seems to me that in all these tests too much reliance has been placed on soap tests and temporary and permanent hardness and not enough data given on the alkali salts in solution, chlorides and sulphates of soda or potash, and the possibilities of inter-reaction between these bodies and the lime and magnesia salts. It would also be interesting if the author could tell us whether the luminator has a favourable effect in preventing the dissolved CO_2 in the water from causing blistering of the plates as this gas often causes pitting at places where the circulation is slow such as at the ends of the tubes in the 'header.'

These are only a few suggestions which perhaps may be enlarged on in the author's reply to the discussion; but in dealing with a new phenomenon like this aluminium effect seems to be, one must make all the *material* tests one possibly can before practically attributing the result to the supernatural.

ACCIDENTS IN TRANSVAAL MINES.

(With Special Reference to Falls of Ground.)

(*Read at November Meeting, 1911.*)

By James Chilton (Member).

DISCUSSION.

Dr. J. L. Aymard (*Associate*): Most people are so lacking in imagination that they can only dimly realize the most striking facts. A disaster like that happening to the *Titanic* gets home to all of us. Even distant Johannesburg helps to raise relief funds to mitigate its physical misery; and the millionaire actors in the tragedy give place in the boats to the poorest women emigrants, and die like men and gentlemen. How many of us here realise that, every year, roughly speaking, as many unfortunate human beings meet unnecessary and untimely death in the mines of the Witwatersrand, deaths just as painful, and bringing as much misery to others as those on the doomed *Titanic*? *What do we care about it?* As Mr. Malan pointed out in Parliament the day before yesterday—we take next door to no notice of it at all. *What do we do about it?*

Mine accidents are so varied, that classification is difficult. The heading, "Illness," as a contributory cause, should find a place. Those with experience know that natives often work in a condition nearing collapse, and that in the last stages of pneumonia, they struggle to their work, to die the same night or next day. Natives in such a condition are a source of danger to themselves and others.

In answer to the question "What is to be done?" the reply probably will be, "See what we do already. We provide hot coffee, cocoa, drying houses" and so forth. Doubtless these are excellent things in their way. The real question is the extent of real and effective care and supervision. Special circumstances require special remedies. Yet the medical attendance on the Witwatersrand natives, worked out on the basis of a medical man's "whole-time" services, is one doctor to about seven or eight thousand natives, and delicate ones at that. One man, giving his whole time, could not effectually supervise the health of two thousand natives, and I believe the present day medical arrangements are quite inadequate. A question arises as to how far a white man in the later stages of miners' phthisis contributes to the accident list. One knows that he is physically unfit, and likely to delegate important work to his "boys,' to the common danger of all employed.

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[*] *Journal of the Society of Chemical Industry*, Vol. XXXI. No. 2, Jan. 31, 1912.

The method of preventing accident or disease by making someone else pay appears to be finding favour. On what grounds it is difficult to say. The intention of the Government is plain—to make the industry pay. It will then of its own accord stop miners' phthisis. But the unexpected often happens. Judging from the past, it is open to question whether the industry has sufficient business instinct for such a course. It is more likely that having estimated its liability, it will try to place the burden elsewhere, and leave preventive measures to the energies of bodies known as "commissions," with the usual results.

Miners' phthisis is a striking example of national calamities needing desperate reforms. When the Government denies previous knowledge it merely displays its ignorance. The duty of seeing that the mines are put in order and keeping them up to the mark is that of the State acting through the Mines Department, but unfortunately the Government takes any amount of trouble about ostriches and cattle and hogs, and very little about the human animals who work the principal source of State Revenue.

Mr. Tom Johnson (*Member of Council*): I have spoken several times on this paper and have twice made the statement that the eight hours shift in Great Britain was not accountable for an increased death rate. I am pleased to state that despite the existence of the eight hours shift in 1911, the lowest number of deaths per thousand yet recorded for any year were registered, viz., 1·18 per thousand, a reduction of 30% on the figures for 1910, which were 1·69. I do not claim that the eight hours shift is reponsible for this satisfactory state of affairs.

It will be noticed also that the Transvaal figures for last year were the lowest recorded for many years, despite all the alleged hustling that is taking place. It may be thought that I am too insistent on this eight hours question, but this accident rate of ours is too grave a matter to be dismissed as being caused by this or that, without full adequate examination. It may be that as a result of the reading and discussion of this paper legislation may follow, and it behoves anyone to dissent from any of the opinions expressed if it is considered to be wrong. It is on this account and not from any carping spirit of criticism that I have referred to this eight hours shift again, and that I dissent from the opinions expressed or inferred by most of the contributors that the causes of the high rate of accidents are hustle, the introduction of new men, or to change in management, etc. These things play a small part when considering white labour, as I take it that the whites are those thought of by the different contributors in discussing this subject. To get down to bed rock, the matter has to be very thoroughly investigated. There always will be accidents in mines as in other places, despite every precaution, as there is always the unforeseen to be dealt with, but there is no reason why we all should not do our best to prevent those accidents that are preventable. It is, however, necessary to learn the causes of accidents, or one is likely to find more harm done than good achieved.

Mr. J. Moyle Phillips (*Member*): Much stress has been laid on the incompetency of our miners and the scarcity of practical men. I attribute this incompetence to the following causes:

First.—One class of our men is recruited from a purely agricultural and pastoral race who are devoid of any instinct of mining or even discipline. These, in the great majority of cases, are driven to work on mines by force of circumstances. Their intention is not to become miners, but to earn enough money—no matter in what capacity—to enable them to purchase a few breeding cattle or a span of oxen and a wagon, when they will either retire to their "farms" or go transport riding. No more will be seen of them until their cattle or oxen die, when they will again return to work in the mines with the same object in view as before.

Second.—Another class comprises those who are ex-army and ex-policemen or who have been retrenched from the civil service, and these from necessity—not choice—drift into the mines. Their hearts are not in the work and they are out of their element, and with the exception of a few—and fortunately they are generally the pick of them as far as mining work is concerned—they quit on the first offer of work in more congenial surroundings.

The third reason has not so much to do with the men themselves as with those under whose supervision they are placed and who are expected to teach them their work. The teacher is supposedly a practical man, but practical men never came to our mines to teach others their profession, but to earn as big cheques as they can, and so find they have no time to devote to their understudies. If it were their own sons or the son of a friend then they could and would find time, but for a total stranger, no! Thus the poor learner has to *pick up* what mining knowledge he can by himself until such time as he gets a blasting certificate, when of course—in his own opinion—he is a full blown miner, and consequently quits the mine on which he *learnt* mining to go off in search of a miner's job elsewhere. Thus we find him a week here, and maybe a month there, until he strikes a job where his incapability as a miner is no hindrance to him and he remains until—for some reason or other—he gets the sack, when the same shifting about occurs, and thus we get our present class of "miners."

Now it is no use baulking the fact that these men *are* as we made them, and until some different scheme from that at present in vogue is adopted to teach them, it is useless to expect to make them practical miners, and we still have to look overseas for our practical men. But what inducement have we to offer practical men? A good practical man—such as we are greatly in need of here—can earn a good wage in his own country, so unless something very enticing is offered him he will not leave home. Formerly there was the "bait" of a big cheque, but since it is now known what price he has to pay for it he will not bite, and even if he does, it is not in order to stop, and what is most particularly wanted is that the practical man should stop here and make it his home. At the present time I do not see what inducement there is for him to come; our conditions underground must be made more healthy and the terms of employment must be more permanent.

I would like to state here that I notice a vast improvement in our underground conditions, both as regards ventilation and allaying of dust and sanitary arrangements, and I see no reason why in the near future—say another two years—our miners from a health point of view should not compare favourably with the most healthy miners elsewhere. We have at present several methods for allaying the dust whilst drilling and shovelling, but there is still the question of arresting that which is created whilst blasting. Now I do not know if there is any system in vogue here at present, but it has struck me that a system of water screens or curtains across our drives and cross-cuts would greatly assist in, if not entirely, arresting the dust thus created. This system would be simple and inexpensive, the only thing necessary being an adequate supply of water—which we already have—and an inch pipe connected up with the water pipe in the drive and carried across the top of the drive; this pipe to be perforated as in the style of street watering carts. At blasting time the developer or stoper could—as he is passing along the drive after lighting up—turn on the water and thus create a very effective dust arrester. These screens could be shut off again at the commencement of the following shift. Thus for about three hours there would not only be a dust arrester at work which requires no—or an infinitesimal amount of—supervision, but an air cooler as well.

As to the chance of catching the rising generation of our miners here and making them in turn miners, I fear such is hopeless, as no mother will allow her son to follow the trade or profession which killed his father in eight years from phthisis. As has already been remarked by other members we have not enough old hands left; those who are still alive are but as a drop in a bucket of water, and nearly eaten up with phthisis. I only know of two men here at the present time —I am referring to miners—who were here in 1888, and the men whom I knew prior to the war and are alive and *working* to-day could be counted on the fingers of one hand.

What constitutes a practical miner? Is a good machine or a good hand stoper a *practical* miner? We have our underground work specialised so much now-a-days that it is hard to say who is a miner and who is not, much more who is a practical miner. My definition of such a man is "one who can do any practical work demanded from the collar of the shaft to the bottom." If this is the same definition as that of the members, then I say the only—or at least the principal—positions these men should occupy are those of mine captain or shift boss. No person can supervise work better than the person who has done the same kind of work himself. It is useless to be in a position to *tell* a man to do certain work unless one can also *show* him how it should be done. It will not be done, or at least will not be done in an efficient manner. Whilst on the subject of the miner and before we thoroughly condemn him as incompetent and responsible for a great percentage of our accidents, it is best to make ourselves acquainted with the conditions under which he has to work and the circumstances outside of his practical work which he has to contend with. As regards his practical work, he may be a new man and is given a gang of new boys; he consequently cannot come up to the standard of his predecessor in the same stope and consequently may be deemed incompetent and get the sack on that account. He may be a practical man working under an impractical shift boss. His holes may not be placed to suit the shift boss, or he might be using more dynamite than the shift boss calculates he should, this is quite sufficient for him to be deemed incompetent. He might have an unruly gang of boys; may be the previous ganger was "slack" with his boys, allowing them to run him instead of his running them. The new man might be a strict disciplinarian, consequently he'll have trouble in licking them into shape. One or two of the more unruly will require something more solid than the tongue to be applied to them before they begin to realise that the white man is boss. The consequence is that these boys complain to the compound boss, who complains to the manager, who tells the mine captain to tell the shift boss to notify the man that his services will be no longer required, as he is incompetent to boss boys, notwithstanding the fact that he had already complained to the compound boss about the boys! Thus we have three reasons whereby a man may be and has been deemed an incompetent miner.

Further, consider the number of officials with whom he has to come in almost daily contact:—1. Underground manager; 2. Mine captain; 3. Shift boss; 4. Trammer shift boss; 5. Roof inspector; 6. Machine boss; 7. Sampler; 8. Surveyor; 9. Compound manager; 10. Timekeeper. Each and perhaps every one of these will have something to say or some order to give, and one can see the great possibilities of contradictory orders to the bewilderment of the miner. Now where in the whole mining world has a miner so many officials to deal with in the course of his day's work, and is it any wonder that some times a man gets disgusted and quits for this reason alone?

I will now take another class of underground employee, or rather official, which is being blamed for the accident rate, viz.: the shift bosses.

If one was not personally acquainted with this class of official and judged from the remarks passed on them at our late meetings, one would come to the conclusion that they are the most incompetent underground men of all. I do not think for one moment that this is the construction which it was intended to convey, and probably if one *heard* the remarks instead of seeing them in cold print, one would find that such was not intended, but the vast majority of our members do not or cannot attend the meetings and can therefore only judge the speakers' remarks by what they see and read in their *Journals*, and are thereby apt to arrive at a very different conclusion from that which the speakers intended.

Speaking as one who knows a great many of our shift bosses I can say that as a class they are hard to beat in any part of the world. Certainly there are incompetent men among them as well as among the miners, but in what part of the world will any trade or profession be found in which every official is a practical and thoroughly competent man?

There are shift bosses here to whom a little more experience would be beneficial, not only to themselves but to the men under them, but these inexperienced men as a rule have been placed in their present position by the powers that be. They no doubt in a great number of cases have had the position definitely promised them when they first commenced their mining studies. Consequently their aim was not so much to learn the practical part of mining, but to get hold of sufficient theory to pass their examinations the practical part having to be learnt after, and that I think is what is happening to-day. The men have passed their theoretical examination and are now learning the practical part of their mining in the capacity of shift bosses. But I do not think this class of shift boss is in the majority and so it would be wrong to judge the whole class for the sake of a "few incompetents." We have a lot of young men here now who in spite of their lack of practical knowledge make very good shift bosses, and if they would use a little more "tact" in dealing with the men under them this would not be so apparent, and they would find that their relations with the men would be more comfortable and the men themselves would give of their best in return, and the present friction between the two would soon break down. I think it was Napoleon who—after one of his great battles in which a high percentage of "non-coms." were killed—said, it is easy to make a corporal but one cannot always make a soldier; and so with mining, a shift boss is easily made but not so a practical miner. One reason perhaps why there are not more practical men as shift bosses is that the pay is not remunerative enough. A practical man can earn more money by "working" than he can as a shift boss and his work is not so arduous or the responsibility so great, and he has not to "eat" more dust.

I cannot see the necessity of another official underground in the shape of a roof inspector; the inspection of the roof should form part of the duties of the shift boss. If he is incompetent to perform that duty then he is not fit to be a shift boss, and if his section is at present too large to allow him time to attend to this, this should be changed so that he can easily embrace this among his other duties. Possibly it may necessitate an extra shift boss, but it would mean doing away with three or four roof inspectors. A shift boss is supposed to visit each working face and travelling road in his section once in each shift; surely by so doing he can make himself conversant with the state of the roof. The same thing applies to tramming shift bosses. Here we have three officials traversing the same section and visiting the same places each day to do the work which the shift boss alone can and should perform. This duplication of underground officials does not tend to the greater efficiency of the men and often breeds discontent, as there is always the danger of the orders issued by the different bosses being conflicting.

Resueing has been alluded to as likely to become a lost art on these fields. I do not see why it should be so as it is not necessary for this method to be confined to ore bodies of only 6 and 12 in. wide; it can be in a great many cases applied to bodies far wider, and with great advantage both economically and on the score of safety. We have so got into the habit of blasting both our waste and ore down together that the art of resueing has been overlooked or (what is more likely) not understood until it was prominently brought to our notice by its being put into practice at the Jumpers mine. There are several other methods which are being overlooked at present simply because they are *old* or that their uses are not sufficiently understood.

I cannot agree with Mr. Johnson that the fact of running one, two or three machines has so much to do with the accident rate as he seems to imply. I take it that he contends that a man who is running two or more machines has no time to examine the roof, etc., I agree with him; but what more time has the man who is running only one machine? If the one man machine principle was adopted, as Mr. Johnson suggests, would we not expect that man to drill more holes per machine than when running two or more and would we not expect him to stop at the handle himself and not leave his Kafir to turn it? In fact to make the practice economically better the man would only have one "boy" to assist him and so what time would he have to examine his roof? In fact the man with three machines has more spare time on his hands because he has two boys at each machine, and it is not necessary for the man to be continually at the handle. I am sorry to hear from Mr. Johnson that it is the miner of some years standing who is responsible for more accidents than the younger hands. If this is true, why such an outcry about the lack of experienced miners? Surely a miner of some years standing can lay claim to being a practical man. The two remarks, one complaining of the lack of practical men and the other blaming them for the cause of accidents, seem contradictory. It is not the mere question of the great amount of rock which is being broken and hoisted that has to do with the speeding up underground, but rather the reverse; it is the *lack* of sufficient ore to keep the mines going that is the cause of it. There is not sufficient "ore" being broken, consequently there is always the continual hurry and scurry to try and find sufficient, and that is also the reason why so much waste rock is sent to the mill which otherwise should and in all probability would be left in the stopes. I also endorse Mr. Vaughan's statement when he says that some of the outside mines are more dangerous than those of the Rand, and I would refer Mr. Johnson to some of the mines in and around Pilgrims Rest.

Mr. Cullen's impression that blasting certificates are so easily obtained is correct. Any man who can buy a copy of the Mining Regulations and is capable of reading and understanding those relating to blasting and handling explosives can easily obtain at least a provisional blasting certificate. The introduction of the blasting certificate has not raised the efficiency of the miner one iota, and I venture to say that at the present time there is a greater percentage of incompetent men allowed to do blasting than there was at any time prior to the issuing of certificates. The same remark can be applied to the other numerous certificates which this Government compels us to have. The possession of any kind of certificate granted in this country is not a guarantee that the holder thereof is a capable and competent man, in fact I have known certificates to be obtained by theoretical knowledge only.

I notice with considerable surprise that many of the reasons advanced in this discussion are the same as were put forward at the time Mr. Richardson's paper on "Mine Subsidence" was read, which was in March 1907 or five years ago. Now this strikes me as rather a peculiar thing to happen on a place like the Rand where we are credited with having more brains per square inch than any other place on the globe. At the time the remarks were made—I am referring more particularly to the leaving of more pillars and waste rock in the stopes—no adverse criticisms were made on them, in fact they were practically assented to, and yet we find the same remarks being made again to-day. What is the reason? Were they not practicable or applicable then? A great portion of the first contribution which I sent in to this paper was composed of notes made at the time of Mr. Richardson's paper, but which I forgot to send in before the discussion closed. When I received an advance copy of Mr. Chilton's paper I turned up these old notes and found the majority of them applicable to the present time, so I sent them in, and since then I have found that several other members have corroborated them. I have come to the conclusion that as far as safety measures underground are concerned we have not progressed at all in five years.

The points already referred to certainly contribute, in a greater or less extent, to the accident rate, but I certainly do not think that they are accountable to such an extent as the contributors seem to imply. I call them minor ones as compared to what in my opinion is the main cause, viz., our present method of mining. Although our present method has not been seriously questioned, yet as far back as March 1909 (Vol. vii., No. 9) when Mr. Richardson read his paper on "Mine Subsidence," the policy of packing stopes with the waste broken underground and the leaving of more pillars was advocated. At the present time we are working on what I might say are exactly the same lines as when our reefs were first opened, and this in spite of our increased depth and change of dip. Our method may be described as one analogous to the "long wall" method of coal mining, our "walls" extending from level to level. The aim of "long wall" working is the total extraction of the ore body. No pillars are left except those which are absolutely necessary for the protection of the shafts and main hauling or tramming roads. The three most essential things—to ensure its most complete success—are narrow beds, a roof which will subside gradually and evenly, and sufficient waste adequately to pack the excavations; the absence of either one of these three essentials is quite sufficient to mar the method, and it has to be perforce modified, and for whatever feature is absent some other means has to be substituted. On the Rand as a whole we have not these three essential elements to aid us, either the one or the other is lacking, and sometimes the whole three, so that we have had to modify it by leaving behind us supports in the shape of mining props, packs or pillars of solid ore. But—as other contributors have already said—the pillars left have been and are too small in size and too few in number, and the waste which is being broken along with the ore has been and is being sent to the surface instead of being packed in the workings. So that although we have somewhat modified the original method, yet it is worked in such a way as to render nugatory the real aim of the modification. It has already been pointed out that not only are the pillars too small and too few, but that often they have been left standing when they are a menace to, instead of an aid to safety. Mr. Richardson in "Mine Subsidence" advocated the leaving of pillars every 50 ft., the pillars to have a minimum diameter of 10 ft. I have—in a former contribution—pointed out that chiefly on account of weathering and shearing 10 ft. pillars are in my opinion too small. I am no believer in small pillars, because a pillar of 10 ft. or smaller, if cut by machines, can be better replaced by a 10 ft. pack on the score of cost and utility. Therefore I would enlarge on Mr. Richardson's scheme and leave my pillars of a minimum diameter of 20 ft. with a minimum distance of 100 ft. from the centre to centre of pillars, but I would not stop at this as I would supplement it wherever possible by strongly built packs of not less than 10 ft. in diameter to be further supplemented by mining props of not less that 9 in. in diameter at the small end. The smaller the number of packs—through failure to obtain necessary waste—the greater the number of mining props and *vice versa*. In any case on the top of each "prop" should be a head board of softer wood than that of the "prop" itself to allow of it—the prop—setting snugger and also as an indicator of the weight coming from the roof. One can quickly and easily detect the pressure by the depth to which the "prop" has penetrated the head board. This method is simply an enlargement of the one at present in vogue, only it approaches the "pillar and wall" method of coal mining rather than that of the "long wall." Now from what I have seen and also from the remarks of others, I have come to the conclusion that the "pillar and stall" method is the one which is being aimed at at present, only that the necessity of keeping the mills going prevents the leaving of adequate pillars, both in size and number, to render it satisfactory. I venture to say we have come to the stage now where we are, or should be, compelled under our present method of mining to leave larger and more pillars if we really intend to reach the greatest depth possible for economical mining. This "pillar and stall" method will be condemned no doubt on the score of having to leave too great a percentage of ore behind in the shape of pillars, in fact it has already been condemned on that account, but I think the loss of ore by this method is more imaginary than real when one takes into consideration the fact that a great amount of it can be recovered almost daily by the weathering and shearing of the pillars, and also by the fact that if the pillars are left large enough in the first instance, when once the boundary has been reached, a start can be made to withdraw them and leave the roof to subside behind. But even within the next 5 years I predict that even this method will have to be discarded or modified, and that we shall have to push our development faces to the boundary and then work back home on the long wall method, leaving everything to ' come " behind us. This would be reverting to the original method, but that we should be working to, instead of from "home." No pillars, packs, or other supports, with the exception of mining props—which could be withdrawn—to keep the working face open would be necessary. Perhaps this is the system that Mr. Johnson had in mind when saying "for both on the score of safety and economy it is better to work without pillars and pack."

We have been labouring under the delusion that we were working under a sound roof, and consequently no adequate supports have been left to secure it, with the result that it was not until we had a series of accidents through falls of rock that the condition of our roof was seriously questioned. It has been in the apparent safety of the roof that our danger has lain, and it is only lately that this has been brought home to us. In my opinion no portion of the roof can be considered safe, even if it sounds so, and mining props should be kept up to within at least 25 ft. of the working faces of stopes, if only as a precautionary measure, to be followed by well built packs not farther than 50 ft. away from the face. If this method were followed not so many cases of injury by the fall of isolated slabs of rocks would occur. In the building of pigstyes the corner props should either be tapered at one end or else a head board of softer wood than the prop itself put on top of the prop. However well packed pigstyes may be, the weight from the roof is bound to come on the props first, and unless there is some way in which they can "give" to that weight, until it has settled firmly on the filling, there is the danger of the props breaking and the whole pack collapsing. Besides, what is the use of filling in between the four props if the filling is not allowed to take the weight? One might just as well leave the props standing bare. In the case of hand stopers I would make them put in their own stope props and be responsible for their roof and within 30 ft. of their working faces. They would then take more interest in the state of their workings and not be dependent on another person to keep them and their "boys" safe. "Everybody's work is nobody's work," and so the miner, knowing that it is some one elses work at present to look after the roof, neglects even to take the most ordinary precautions to ensure his safety and that of the workers under his supervision. I think the fact of our underground work being sub-divided, as at present, is to a great extent responsible for at least such minor accidents as falls of isolated slabs and injuries to boys lashing and tramming. In the earlier days of the Rand every stoper had to superintend the lashing, and sometimes the tramming boys, in addition to the hammer gang, and the putting in of his stope props, and there is no reason why he cannot do so to-day providing he has not more than 30 hammer boys and one (1) stope to look after. I do not believe in one man having more than one (1) stope to supervise, and if he keeps to this he can then easily do his own timbering and look after his shovel boys, thus doing away with the "lashing" bosses.

SOME OBSERVATIONS ON ANCIENT MINE WORKINGS IN THE TRANSVAAL.

(*Read at January Meeting, 1912.*)

By T. G. Trevor, A.R.S.M., Acting Inspector of Mines, Pretoria (Member).

DISCUSSION

Mr. T. Lane Carter (*Member*) (contributed): I read this paper with great interest, as it deals with a subject that has attracted a good deal of my attention in different parts of the world. Just as the artist likes to go back to old times and study the pictures of the period, so does the engineer take a lively interest in investigating the beginnings of mining. While I have seen little of the ancient mine workings of the Transvaal, it was my good fortune to study a number of ancient mine workings in Northern Rhodesia and the southern part of the Congo Free State in the year 1904. In reading Mr. Trevor's account of the Transvaal workings, I was struck with the similarity between his experience and mine.

Much has been written about the ancient workings on the gold veins in Rhodesia, but not enough has been said about the old copper and iron relics. To me, one of the most interesting ancient workings I ever saw in Africa was that of the Bwana Mkubwa Copper Mine near the Congo Free State. In some places along the vein, which could be traced for over one half a mile, the excavations were from 20 to 30 ft. wide and extended to a depth of about 100 ft. Although a large tonnage of high grade carbonate copper ore had been taken out of this mine, the strange thing is that no trace of slag could be found anywhere. The ore, therefore, must have been carried away either to be smelted elsewhere or to be used for some other prupose.

In one part of his paper the author suggests that a probable explanation of the low copper contents of a certain old working which he inspected in the Transvaal is due to the long infiltration that has gone on for many years. I saw a striking confirmation of this theory in one of the old workings at the Bwana Mkubwa Mine. At one place they had taken out what must have been a rich pocket of carbonate ore, leaving a big cavity in the rock. The constant leaching of the vein above this cavity had brought the copper down to the old hole, and the walls were one mass of green. When the prospector first saw this he thought that he had found a copper bonanza, but a few strokes of our picks showed that it was simply an incrustation of copper from the vein above.

I am glad that the author has mentioned the old workings on iron deposits in the Transvaal.

I noticed similar workings and old slag heaps from iron smelters in Central Africa. Ever since my first visit to that wonderful mining country Mexico, it has been one of my hobbies to compare the ancient mine workings of Africa with the old holes one sees all over Mexico. If one is inclined to analyse the past he will find a difference between the ancient mine workings of Africa and those of Mexico that will explain the events which have transformed the history of the old Aztec's country. Scattered over Central and South Africa one finds remnants of iron mining and metallurgy, which show that the natives were well acquainted with the use of this metal. History also proves how well they could make implements and weapons out of iron. But the ancient Mexicans were entirely ignorant of iron and therefore their weapons seemed very feeble to the early Spanish adventurers. One cannot help speculating on what a difference there would have been in Mexican history had the warriors of Montezuma been armed with assegais instead of bows and arrows and clubs. The intrepid adventurers under Cortes would have been annihilated if the Mexican warriors had known the use of iron. Thus has the whole history of the world been formed and changed by the lack of knowledge of certain metals.

There is a practical side to the author's paper which I wish to bring out, and that is the use and abuse of ancient mine workings in present day mining. It seems to be a tendency of human nature to make these workings as old as possible. For instance, in Rhodesia many people were never satisfied until they proved that the Queen of Sheba got out the gold from the old pits to present to King Solomon and others; in Mexico, there are numerous old workings which were excavated years after the Aztecs, and yet mining promoters and romancers are not content until they prove to their own satisfaction that these were secret places from which Montezuma got his gold. I have found that too much importance is frequently placed on these old workings, called antiguas, in Mexico. The fervid imagination of some people sees in every hole in the ground the makings of a great mine, and many thousands of dollars have been extracted from the pockets of impressionable investors in Europe and America by oratorically inclined promoters who waxed eloquent over the treasures left at the bottom of these ancient workings by the Queen of Sheba and the Emperor of Mexico. Before concluding that every ancient working is a big mine in embryo, there are many things to take into consideration. In the first place, we do not give the old workers sufficient credit for their skill. The chances are that all this labour was done by slaves, and they had no strong unions in those days to insist upon an eight hour working day. Then it is probable that the metal extracted was, in the opinion of the old workers, far more valuable than it is with us to-day.

For instance, take the case of silver in Mexico. Up to the year 1870 silver was worth about $1.20 to $1.30 United States Currency per ounce. Since then it has sold as low as 49c. per ounce. Here we have a depreciation of more than 100% in the value of the metal. Of course, we have the great advantage to-day of improved methods of working, but I am of the opinion that the average value of the ore mined from these ancient workings is far lower than the romancers would make us believe. There is another point to remember. We have not found the balance sheets of these old miners and therefore cannot audit their books. We do not know how much disappointment these old holes represent. No doubt they did a great deal of prospecting and much of the work carried on was probably run at a loss. The "Golden Age" of mining was certainly not in the past. They undoubtedly had their trials and tribulations as much as we have. I have heard a lot of nonsense talked about these old workings in different parts of the world. My plea is that when we come to consider them we make full use of good common sense and do not let the romances of the past blind us. These old holes in the ground may lead us to payable mines and they may not. Let us investigate them as economically as possible as do not claim to have a mine until we actually get ore in situ, and past the prospecting stage.

Let common sense prevail in this as in all other mining problems.

PRACTICAL APPLICATION OF THE SPECIFIC GRAVITY OR MOISTURE FLASK.

(*Read at November Meeting, 1911.*)

By H. Stadler (Associate.)

REPLY TO DISCUSSION.

Mr. H. Stadler (*Associate*): It is frequently mentioned that the merits of a paper are to be judged by the amount of discussion it attracts. This may be true for subjects of general interest, or a controversial nature, but for a paper which only attempts to draw attention to the consequences of indisputable physical laws, I would rather apply the saying that the best women are those least talked about.

I am indebted to Mr. F. J. Pooler for pointing out a slip in time to correct the original text. From Mr. Pooler's point of view as a laboratory scientist, he is also right in what he says about the accuracy of different operating methods, but in practical work refined scientific methods must give way to quick and handy ones. It has been said that a complete want of practical training manifests itself in a show of worthless figures and such are certainly the decimals in moisture values in mining work. A punctilious reduction manager once stated in this *Journal* that he found that 40% of moisture in a tubemill gave better results than 38%. If he tried to get out how far this correct percentage could really be maintained in practice, he would not need a flask with a perforated stopper to see that the moisture easily shows variations either way of 10%. I am afraid that on many of our mines even a wide-mouthed flask is too refined an instrument for detecting rregularities which are so promptly and visibly shown by the familiar sight of the shower baths falling from overflowing launders on to roofs and machinery.

Mr. Pooler asks why, throughout the paper, the term "density" is applied in the same sense to dry solid material, as "specific gravity" when referring to pulps or mixtures of water and solids. This distinction in terms and symbols was made to avoid confusion and to facilitate the reading of formulae. The *density* of a substance is its mass per unit volume. The unit of mass, in scientific work, is the gram, which is defined as the one-thousandth part of the mass of a standard piece of platinum kept at Paris. This standard was intended to be equal to the mass of 1,000 cc. of water at its maximum density, and although this is not quite correct, the weight of one cc. of water may for practical purposes be taken as one gram. The term "specific gravity" has in practice become familiar to mean the ratio of the density of a body to that of water (or air). Both terms are therefore practically identical, and as the definition of "density" is unequivocal and clearly defined it seems to me that no questions remain to be settled by a Standardisation Committee for metallurgical and mining terms.

Some ambiguity may arise in speaking of the density of powder or porous materials. We may mean the density of the substance, of which it is composed, or we might—and in practice we do—mean the density of the bulk which we can see and handle. Wood, for instance, as matter, has a density of about 1·2, whilst taken in bulk it is lighter than water. Here again we need no Standardisation Committee, we have only to avoid sloppy writing by saying exactly what we mean.

Having explicitly stated that the correct measurements of the specific gravity of slime pulps by hydrometers require that "all solids be freely suspended," it is not I who should be asked why accurate results cannot be obtained by this method, from a pulp which has passed through a screen of 560 holes per square inch. The surface friction of falling bodies increases so rapidly with the decrease of their size, that the falling velocity of the finest particles practically vanishes. The suspension of a given amount of particles of greater density in a liquid increases its weight per unit volume and consequently also the pressure on the hydrometer. A slime pulp in which finely divided solids move freely, without affecting the fluidity of the liquid by friction amongst themselves, has therefore physical properties similar to those of a liquid of greater specific gravity.

For controlling pulp in agitators, large scale hydrometers, like that suggested by Mr. C. F. Spaulding (*Mining and Scientific Press*, September, 1911), may be advantageously used, not so much for getting the absolute value of the moisture, but as a comparative indicator to control possible fluctuations in the consistency of the pulp.

Outside this Society I have often heard the objection raised, that the specific gravity method is less accurate than the method by drying. This generalisation is, of course, absurd, for both methods are bound to give exactly the same results for an ore of a given density. The accuracy is not conditioned by the method but by the efficiency of the instruments used to carry out the measurements. In any case the specific gravity method is much more convenient, for with the one single operation of weighing, the result is known at once, whilst the method by drying is so slow and troublesome that even such an expert experimentalist as our friend Mr. Crosse seems to have a great dislike to it. At least I conclude this from the readiness with which he welcomed the model of a specially arranged balance allowing direct reading of the moisture percentage on a scale on the lower part of the beam.

NOTES ON ABSORPTION OF GOLD BY AMALGAMATED COPPER PLATES, CAST IRON RETORTS AND AMALGAM TRAYS.

(*Read at December Meeting, 1911.*)

By Prof. G. H. Stanley, A.R.S.M., F.I.C. and M. Thornton Murray, M.Sc.

DISCUSSION.

Mr. G. A. Robertson (*Member*): I take it that the piece of copper plate experimented upon by the authors contained traces of amalgam or mercury perceptable to the naked eye. If so, such amalgam can, I feel confident, be recovered by "dry scraping" which is a more simple method for the final clean up of amalgamated plates than many methods advised in text books, provided the plate is flat or can be flattened so that the scraper can come in contact with the surface of the copper. The dry scraping method is employed on some mines for the monthly clean up of the amalgamated plates, and though this process has been tried on several mines and condemned, I must say that I prefer it to steaming. I do not say that the dry scraping process would be efficient for recovering three or four months accumulation of "steam amalgam." In these days however when amalgamated plates are kept under "double lock," it is difficult to see the reason for accumulating amalgam on the plates, more than is necessary for maximum extraction by amalgamation. The dry scraping process is simply the application of an ordinary scraper to the amalgamated plate, the scraper having the cutting edge rounded (Fig. 1) and used on the plate in the direction shown (Fig. 2). Before making a final clean up the plate should be allowed to get as dry as possible, that is free from loose mercury or soft amalgam, to prevent smearing. Dry scraping, to a new hand, is a heartless job, but as it seems so easy to the experienced man, the new comer, though disgusted with the change from the steaming or scouring process, does not give it up when he sees his neighbours making headway. An experienced hand can leave a transparent film of amalgam on the plate, which when dressed without being scoured, maintains a bright appearance, or he can show a clean sheet of copper, which, after mercury has been added and rubbed over with a wet brush, shows up no specks of amalgam through the mercury catching on. Obviously in a hard or final clean up the pressed amalgam contains a proportion of copper. I have not seen any tests made to ascertain the amount of gold still remaining on a plate after a final clean up by this method, but if the authors recovered over 99% by shaving 0·02 in. from the plate, I should say that an equally efficient clean up can be made by the method described.

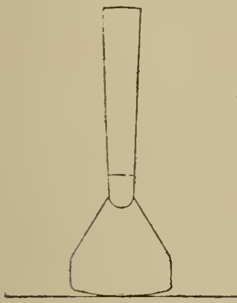

Fig. I.

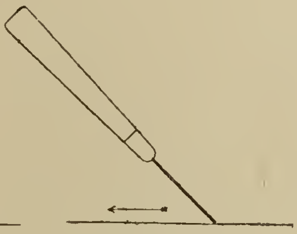

Fig. II.

Mr. A. Whitby (*Member of Council*): Some remarks made by Mr. White on this paper have encouraged me to bring forward a slight contribution to the discussion, mainly relating to absorption in tube mills. Of course one recognises that the tube mill using rock to crush rock is somewhat of a homeopath, and the points I wish to make do not fit in. Making some laboratory experiments with a tube mill taking a charge of 25 lb. of ore crushed to the 20 mesh, and normally amenable to amalgamation, I have found that the mill and balls were capable of absorbing almost the whole of the free gold. A pyritic ore going in 13·8 dwt. came out after three hours only 9 dwt., whilst an oxidized ore of 17·4 dwt. came down in the same period to 7·8. I may say that the pebbles used were really the reduced balls of the Giesecke mill and a close examination under the lens showed the gold actually beaten in. Running the tube mill with cyanide solution alone, I recovered several pennyweights of gold and could still see gold on the balls. There is one other point I wish to make. The authors have endeavoured to prove that old retorts do not show gold on the outside. The specimen I have here seems to prove the generally accepted contrary experience, that it is often more to be found on the outside than the inside, or perhaps, to be correct, more patent.

There are factors influencing this transfusion of gold in retorts such as the quality of metal and the heat used in retorting. It is comprehensible that some retorts are more porous than others, and certainly amongst those we are accustomed to use for small lots of amalgam up to 300 to 400 ounces, there are some in which the metal is so inferior that it melts at a red heat.

The meeting was declared closed.

Obituary.

The death is recorded with much regret of Mr. Eugen E. A. Blume, A.I.M.M., a member of the Society. Mr. Blume came to South Africa soon after the war from the Elmore Oil Concentration Co., was elected a member of the Society in February, 1903. He subsequently joined the Rand Mines staff, and acted for some time as assayer at the Nourse Deep, Ltd. Ill-health necessitated a change from the Rand and after some time spent at Ookiep, Namaqualand and at Bulawayo, he eventually left for England about 12 months ago. He died in London on the 1st March last. The sympathies of his many friends on the Rand are with his mother and sisters in their sad bereavement.

Reviews.

"A Manual of Fire Assaying," by C. H. Fulton. Second Edition, 1911, pp. 219. (Mc Graw-Hill Book Company).

The second edition of Dr. Fulton's book on fire assaying will not be a disappointment to those who are familiar with the first issue, or with the author's other works. Not only has the book been re-written and revised, but numerous relevant facts and data, which have only been made public within the last four years, have been included in the more modern work, while the whole book is written from a very much more scientific standpoint than any previous publication on the subject.

The pages dealing with the composition and nature of assay slags, and the methods of calculating them, and the tables and curves showing cupellation losses at different temperatures, are of especial interest, while the chapter entitled: "Special Methods of Assay" will appeal to assayers unaccustomed to testing refractory ores.

In discussing the calculation of assay charges the author is inclined to give the impression that no excess of slag is necessary, over the calculated amount—an impression which one feels sure Dr. Fulton did not intend to convey. It would be unfair, however, to subject a text book which is otherwise so highly to be recommended to meticulous criticism. Indeed the second edition of "A Manual of Fire Assaying," containing, as it does, such an amount of useful information for assayers and others, can only be unreservedly welcomed.

Besides the pages dealing with the assay of gold and silver ores, there is a chapter on the dry assays of ores of tin, mercury, lead, bismuth, and antimony, the author pointing out that in many cases these methods of assay—somewhat inaccurate though they may be—are still the basis for commercial transactions. (M.T.M.)

"West Australian Mining Practice," by E. Davenport Cleland, cloth 4to., 270 p., with a large number of plates and illustrations. Kalgoorlie: The Chamber of Mines. Price 25/-

This book is a companion to *West Australian Metallurgical Practice*, published a few years ago. It is a valuable work containing a complete description of the geology of the three important gold mining districts, Kalgoorlie (the "Golden Mile"), Leonora and Day Dawn, as well as an exhaustive description of the methods of mining employed. The great bulk of the matter contained in the book has already appeared in a series of articles in the *Journal of the Chamber of Mines of W. Australia*.

Owing to the influx of Mining Engineers into Western Australia from all parts of the world, the methods of mining adopted there are certainly not rule of thumb, which makes this book of unique value. The arrangement is excellent, and besides being a good text book it should also become a valuable book of reference.

The geology of each of the "lode deposits" of the Kalgoorlie district, of the lenticular type of ore shoot of the Sons of Gwalia mine at Leonora, and of the various types of quartz reefs of the Day Dawn neighbourhood has been thoroughly described in Chapter 1, the work of the Government geologists having been largely drawn upon.

Chapters 2 and 3 are devoted to shaft sinking and timbering. Excellent descriptions of the methods of sinking by the "stope cut" and "centre cut" systems are given, together with the quantities of explosives consumed. A large number of illustrations of various types of penthouses, and of different methods of shaft timbering, both for vertical and inclined shafts, are given, and tables of analysed costs of sinking.

A valuable collection of plans of various kinds of head frames of different structural materials is shown in Chapter 4, together with most useful notes on their design. In the same chapter are descriptions of various winding engines and compressors used, while much useful information on various kinds of winding ropes is added. Part of the chapter is devoted to rock drilling machines and tool sharpening, with a drawing of a coke-fired drill-heating forge.

The excavation of shaft stations is detailed in Chapter 5, in which are drawings of double and single stations serving double lodes.

Development work is treated very exhaustively in Chapters 6 and 7. Diagrams showing the positions of drill holes and the orders of their firing in different classes of ground are given; also the consumption of explosives. Details of various development costs are given in segregated and tabular form. Timbering of levels by framed sets, stull sets with pole lagging, stull and post sets, saddleback and rafter methods are well illustrated with photographs.

The constructions of various kinds of ore passes and chutes, including the "Chinaman" chute is described and illustrated in Chapter 7.

A most useful chapter is No. 8. It describes the various methods of breaking ore followed, including systems of timbering, residue filling and "measuring up." The "flat back," shrinkage and rilling methods of stoping are mainly in use in Western Australia, and the advantages and disadvantages of each are thoroughly discussed. Comparative statements of costs, which are analyzed, of various methods of stoping, are given.

The underground transport of ore is described in Chapter 9. Dimensioned drawings of various kinds of trucks are given, as well as details of tracks and the construction of station ore bins

The constructions of various kinds of cages, skips and safety appliances are illustrated by a number of drawings and photographs in Chapter 10.

Other chapters of the book describe the various signalling systems in use, methods of sampling and weighing ore, diamond drilling and mine drainage. A large amount of information is given in a chapter devoted to the composition and behaviour of the various kinds of explosives and safety fuse used in the State, and a summary of the Government regulations in connection with the use of explosives; also a summary of the findings of Royal Commission recently appointed to look into the ventilation and sanitation of mines.

The description of the Coolgardie water supply, by means of which several million gallons of water are pumped daily a distance of 352 miles from a catchment area near the coast to Kalgoorlie and neighbouring districts, would be of great interest to civil and mechanical engineers. The manufacture on the spot and setting up of the "locking bar" pipes, used in the water scheme is described in detail.

The book is written in a very clear style and the illustrations and drawings are all excellent and would be of profit to the trained man as well as to the student. (R. A.)

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Estimating Alkalinity in Lime.—"At the meeting of the American Institute of Mining Engineers in San Francisco, Assistant Professor Luther W. Bahney, of Stanford University, California, presented a paper entitled 'Rapid Estimation of Available Calcium Oxide in Lime Used in the Cyanide Process,' from which the following is abstracted:—

Lime is the alkali that is most universally added to solutions in the cyanide process of gold and silver extraction, for maintaining the so-called 'protective alkalinity.' It is produced by calcining limestone, and therefore varies in the percentage of calcium oxide which it contains according to the purity of the limestone. The value of lime as an alkali depends upon the purity of the limestone, the temperature and period of calcining, the length of time it has been stoed, and whether it has been subjected to dampners. Good quicklime can be purchased in the Unisted States from reliable producers, but in Mexico and Central America, where it is apt to be poorly made, the quality is quite variable.

In consideration of the foregoing, it is apparent that there is need for a rapid method of the valuation of lime to be used in a cyanide plant. The determination of calcium by the gravimetric method requires too much time and is too complicated for an isolated plant. The calculation of the calcium so found to calcium oxide, although sometimes done, is inaccurate.

Several methods of titration by means of a standard acid have been described, and no doubt give results sufficiently accurate for a technical method, but the objections to these methods are that they involve the preparation of a standard solution of some acid, usually decinormal hydrochloric acid, which cannot be weighed out, but must be standardised with some other standard solution. Solutions of the following acids have been used by different operators for standardisation: sulphuric, nitric, hydrochloric and oxalic. Oxalic acid is perhaps the most favourable for this purpose, because a standard solution can be prepared by weighing the solid acid and dissolving in water. The use of the solution employed to determine the alkalinity of the cyanide solutions has also been suggested.

While the method of standardisation with oxalic acid is open to the objection that the hydration of the acid may vary somewhat, yet it yields a solution sufficiently accurate for technical work, but so far as I am aware its use has not been suggested.

For the purpose of determining the feasibility of using oxalic acid, the crystals were dissolved in distilled water, and a decinormal solution made. A decinormal solution of pure hydrochloric acid with distilled water was also made, and both were standardised with a solution of chemically pure sodium carbonate.

Pure calcium oxide was prepared by grinding pure white crystals of calcite in an agate mortar and igniting the fine material in a platinum crucible over a strong blast until constant weight resulted.

This oxide, cooled in a desiccator, was ground in an agate mortar to pass 200 mesh, and the percentage of calcium oxide determined gravimetrically: the result was 99·98, as compared with the theoretical 100%.

The calcium oxide so prepared was used as a standard throughout the succeeding tests. Similar weighed portions were titrated with decinormal hydrochloric acid and oxalic acid, using phenolphthalein as an indicator, requiring 44·2 cub. cm. of hydrochloric acid or 44·6 cub. cm. oxalic acid to complete the reaction.

The solution of oxalic acid used in the subsequent experiments was made by dissolving 14·6068 gm. in enough distilled water to make a litre, this strength being recently suggested for determining the protective alkalinity of cyanide solutions.

The first experiments were made upon small amounts of 140 mgm., to which was added 100 cub. cm. of water before titration, the idea being to have just enough lime present to be theoretically soluble in that amount of water.

This quantity is somewhat small to handle conveniently, and the published method* of weighing out 14 gm., making 1,000 cub. cm., emulsion, removing 100 cubic centimetres, and again diluting to 1,000 cub. cm. and removing 100 cub. cm. for titration, did not give results which checked upon low-grade limes; moreover, this latter method is open to the objection of extra manipulation. A larger amount was then tried, introduced directly to a flask in which the determination was to be made.

The weight of lime to be taken was calculated so that each cubic centimetre of oxalic acid solution would represent 1% of calcium oxide, as given in the formula:

$$\begin{array}{cccc} \text{Lime} & \text{Lime} & \text{Oxalic} & \text{Oxalic} \\ 56{\cdot}09\ : & x\ :\ : & 126{\cdot}048\ : & 14{\cdot}6068, \end{array}$$ in which $x = 650$

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* Treadwell and Hall, Vol. ii., p. 453.

This weight, 650 mgm., was used in all the tests, and Table I. shows the results, which are sufficiently satisfactory for a technical method.

The titrations were made in the cold by introducing 650 mgm. of the sample into a 300 cub. cm. Erlenmeyer flask containing 50 cub. cm. of distilled water, using phenolphthalein as an indicator.

Table I.—Results of Titration for Calcium Oxide, using Oxalic Acid.

Calcium Carbonate Present.	Calcium Oxide Present.	Calcium Oxide Determined.	Calcium Carbonate Present.	Calcium Oxide Present.	Calcium Oxide Determined.
Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
95	5	5·2	45	55	54·5
90	10	10·3	40	60	59·9
85	15	15·3	35	65	64·8
80	20	20·3	30	70	69·6
75	25	25·0	25	75	74·5
70	30	30·2	20	80	80·2
65	35	35·0	15	85	84·8
60	40	40·0	10	90	90·0
55	45	45·0	5	95	94·7
50	50	49·8	0	100	100·0

The results given in Table I. indicate that calcium oxide in presence of calcium carbonate can be determined by this method with a fair degree of accuracy.

Silica, present in most limes, does not interfere. Magnesia, also present in most limes in greater or less amount, is very slightly soluble in water,[†] and shows a faint reaction with the indicator; but it is of no value as an alkali in cyanide work and should not be shown in a determination of the available alkali in lime to be used for that purpose.

Fortunately, the point where the alkalinity due to calcium oxide stops is readily recognised after a little practice, for the colour is a vivid pink, while that of magnesium oxide is faint. Moreover, the colour in the titration of magnesium oxide disappears with the addition of only ·1 or ·2 cub. cm. of oxalic and solution, and returns very slowly and feebly, while that of lime is rapid and sharp. This is illustrated by the fact that a titration of pure calcium oxide requires only 5 minutes, while the same amount of magnesium oxide requires 3·5 hours.

In order to test the oxalic acid titration in the presence of magnesia, two samples of limestones containing magnesia were ground to 200 mesh, ignited in a platinum crucible to constant weight and titrated. The calcium oxide in each sample was determined by the gravimetric method, since there was no silica present, and only a trace of iron. The following results were obtained:

	Amount of CaO by Gravimetric Method.	Amount of CaO by Oxalic Acid Method.
	Per cent.	Per cent.
Sample No. 1 ...	57·6	57·6
Sample No. 2 ...	50·4	51·0

These results indicate that the magnesia does not interfere. Its presence can be judged by the behaviour of the titration, and the approximate amount can be quite accurately estimated by continuing the titration, if one has the time needed.

Iron oxide in considerable amount is sometimes present in impure limes, and it obscures or masks the colour of the indicator, but if the precipitate be allowed to subside, the titration may be carried out to within 1% of the correct result.

The determination of the amount of carbonate present in an imperfectly burned lime may be carried on as follows: Grind the sample to pass 200 mesh, weigh out 650 mgm. and make the titration in the usual manner; call this result No. 1, 'available calcium oxide.' Ignite 650 mgm. of the finely ground sample in a muffle or over a blast lamp, and make a second determination; call this result No. 2. Substract No. 1 from No. 2, divide by 1·78, and the result will be the amount of carbonate present.

Details of the Method.—The sample must be ground to pass through a 200 mesh screen. Into a 300 cub. cm. Erlenmeyer flask place 50 cub. cm. of distilled water; then add the 650 mgm. of the finely ground sample, stopper the flask, and shake vigorously for 10 seconds; add two drops of solution of phenolphthalein, and then run in the standard solution of oxalic acid until the pink colour is discharged; then replace the stopper and again shake. When the colour returns, if it is due to lime it will be a bright, vivid pink, and the addition of perhaps ·5 cub. cm. of solution will be necessary to discharge this colour, but if the flask is again shaken and the colour is faint, weak pink returning slowly, this is the end point for the lime, and indicates that the magnesia is asserting itself.

At all times during the addition of the oxalic acid solution the flask should be violently shaken, being careful not to allow any of the solution to splash out, so the calcium oxide will pass into solution. In nearly every instance of titration of a high-grade lime, the pink colour remained vivid nearly to the finish, which shows that the calcium oxide is rapidly soluble.

If a complete titration is allowed to stand for from 15 to 30 minutes the pink colour will return and show as brightly as in the beginning.

The reading of the burette is in percentage of calcium oxide.

The solutions necessary are: Oxalic acid, 14·6068 gm. pure crystals dissolved in enough water to make a litre of solution. Phenolphthalein, ·5 gm. dissolved in 50 cub. cm. of alcohol and 50 cub. cm. of water."—*Mines and Minerals*, Jan., 1912, p. 368. (A. R.)

DETERIORATION AND SPONTANEOUS HEATING OF COAL IN STORAGE.—"An account of an investigation carried out by the U.S. Bureau of Mines on the deterioration of coal on storage. With New River coal, largely used by the U.S. Navy, there was a loss of less than 1% in calorific value (calculated to "coal substance free of moisture, sulphur, and corrected ash") when a 300 lb. sample was exposed in the open for 1 year. There was practically no loss in samples submerged in sea water or in fresh water. Pocahontas run of mine coal, a steaming and coking coal, which is the principal fuel used in the Panama Canal work, lost less than 0·4% in calorific value by exposure for 1 year. Pittsburg coal, a rich coking and gas coal, suffered no loss in calorific value by exposure for six months. Wyoming coal, a sub-bituminous coal or black "lignite," lost 5·3% of its calorific value in 2¾ years and as much as 3·5% during the first three months' exposure. Storage under water prevents loss of calorific value and as it also prevents spontaneous combustion, may be advantageously adopted in some cases, but there is the great disadvantage that some kinds of coal retain a large proportion of water after draining, and fuel would be consumed in the evaporation of this water. The authors also discuss the question of the spontaneous combustion of coal (compare Threfall, *J.S.C.I.*, 1909, 759) and the following recommendations are put forward: (1) Coal should not be piled more than 12 ft. deep, nor so that any point in the interior will be more than 10 ft. from an air-cooled surface. (2) If possible, only lumps should be stored. (3) Handling of the coal should be reduced to a minimum in order to avoid production of dust. (4) Lumps and fine should be distributed on the pile as evenly as possible, care being taken that lumps do not roll down and form air passages at the bottom. (5) The coal should be re-handled and screened after 2 months. (6) External sources of heat should not be allowed near the pile. (7) The coal, after mining, should be "seasoned" for 6 weeks before storing in piles. (8) Alternate wetting and drying of the coal should be avoided. (9) Admission of air to the interior of the pile through interstices or porous foundations should be avoided. (10) Ventilation by pipes should not be attempted."—H. C. Porter and F. K. Ovitz, *J. Ind. Eng. Chem.*, 1912, IV., 5-8.—*Journal of the Society of Chemical Industry*, vol. xxxi., p. 112—3. (J. G.)

[†] Comey's Dictionary of Solubilities.

Dimethylglyoxime as a Sensitive Reagent for Ferrous Salts.—"When an alcoholic solution of dimethylglyoxime is added to a solution of a ferrous salt, in the presence of tartaric acid, and the mixture is saturated with ammonia, an intense red colouration is at once produced. Ferric salts are not affected by the reagent."—P. Slawik, *Chem. Zeit.*, 1912, XXXVI., 54.—*Journal of the Society of Chemical Industry*, vol. xxxi., p. 126. (J. G.)

Determination of Chromium and its Separation from Vanadium in Steels.—"After discussing defects in the usual methods for the determination of chromium in chrome-vanadium steels, the author proposes a method based on the fact that chromium (and vanadium) can be completely precipitated in a few minutes by boiling the nearly neutral (ferrous) solution with barium carbonate (or cadmium carbonate, zinc oxide, or magnesia), the chromium and vanadium being subsequently separated by precipitating the former as lead chromate (see Noyce and Bray, *Technolg. Quart.*, 1908, 21, 14). A quantity of drillings of the steel, corresponding to not more than 0·06—0·07 gm. of chromium, is dissolved in hydrochloric acid (10 cc. of acid of sp. gr. 1·2 per gm. of steel) in a covered 300 cc. Erlenmeyer flask, the solution is diluted to 100—150 cc. with hot water, nearly neutralised with a saturated sodium carbonate solution, and a slight excess of a barium carbonate "emulsion" added. The flask is covered and the contents boiled vigorously for 10—15 min., with further small additions of barium carbonate at intervals, but not using more than one to two gm. excess of the latter. After allowing to settle, the precipitate is filtered off, washed twice with hot water, the paper and precipitate incinerated, and then fused for 20 min. with 2 gm. of sodium carbonate and 0·25 gm. of potassium nitrate. The fusion is removed from the crucible by means of boiling water, the solution filtered, treated with 1—2 cc. of hydrogen peroxide, and boiled for 5—10 mins. After cooling, the solution is well shaken with a slight excess of nitric acid (1:1) in a separation funnel, the carbon dioxide being allowed to escape by inverting the funnel and opening the stop-cock. The solution is then transferred to a flask, just neutralised with sodium hydroxide, and treated with 2 cc. of nitric acid (1 : 1) for each 100 cc. of solution. The chromium is now precipitated by addition of 20 cc. of a 20% solution of lead nitrate, the precipitate being collected on asbestos, washed three or four times with cold water, decomposed with hot hydrochloric acid (1 : 4), and the chromium determined by titration with $N/10$ ferrous sulphate solution. The results of check analyses are given showing the accuracy of the method (see also *J.S.C.I.*, 1911, 958)."—J. R. Cain, *J. Ind. Eng. Chem.*, 1912, IV., 17-19.—*Journal of the Society of Chemical Industry*, vol. xxxi., p. 132. (J. G.)

Determination of Small Quantities of Methyl Alcohol.—"The colourimetric test of Denigès may be quantitatively applied as follows:—The alcoholic mixture, after purification by the method of Thorpe and Holmes (*J.S.C.I.*, 1903, 232), or otherwise is diluted with water or strengthened with ethyl alcohol until it contains 10% of alcohol by volume. A mixture of 5 cc. of this liquid, 2·5 cc. of permanganate solution (2 gm. per 100 cc.) and 0·2 cc. of strong sulphuric acid is allowed to stand for 3 min., after which 0·5 cc. of oxalic acid solution (9·6 gm. per 100 cc.) is added. The liquid is then shaken and treated with 1 cc. of strong sulphuric acid, and finally tested with 5 cc. of Schiff's reagent. A violet coloration, developing in the course of a few minutes to 20 or 30 min., according to the quantity of formaldehyde produced, gives an approximate idea of the amount of methyl alcohol; whilst the amount of sulphuric acid prevents any development of colour with any acetaldehyde produced. Further tests are now made with the liquid suitably diluted so as to contain from 0·001 to 0·004 gm. of methyl alcohol in 5 cc., and the colourations are compared in Nessler tubes with those given by standards containing known quantities of methyl alcohol in 10% ethyl alcohol. In this way it is possible to detect 0·0003 gm. of methyl alcohol in the 5 cc. of liquid. The method of Thorpe and Holmes (*J.S.C.I.*, 1904, 208) is not applicable to the determination of small quantities of methyl alcohol."—C. Simmonds, *Analyst*, 1912, XXXVII., 16-18.—*Journal of the Society of Chemical Industry*, vol. xxxi., p. 150. (J. G.)

Two New and very Delicate Tests by use of the Reagent "Tetramethyl Base." [Gold and Ammonia].—"The new tests relate to the detection of gold and ammonia by means of tetramethyldiaminodiphenylmethane ("tetramethyl base"), the compound originally employed by Trillat in testing for traces of lead and manganese and since used by other investigators for the detection of ozone. The author has found that a citric acid solution of the reagent—prepared by dissolving 2·5 gm. of the latter in a solution of 10 gm. of citric acid in 10 cc. of water and diluting to 500 cc.—is more satisfactory than the acetic acid solution originally recommended.

The test for gold depends upon the oxidising action of auric chloride upon the reagent, a beautiful purple colour—much more distinct, it is stated, than "purple of Cassius"—being produced in cold, very dilute solutions: the colour soon changes to blue and the liquid finally becomes colourless, but the blue colour is restored by warming the solution. The test is not influenced by the presence of platinum, palladium or other elements; but since free mineral acids interfere with the reaction, the gold solution, previously freed from chlorine by evaporation, must be neutralised and slightly acidified with acetic or citric acid before the reagent is added. The delicacy of the test in a solution containing nothing but gold was found to be 0·01 mgm. of gold per 50 cc.

The test for ammonia depends upon the fact that hydrogen peroxide has no action upon a solution of manganous salts unless an alkali is present, a trace of the latter being sufficient to effect the oxidation, by the peroxide, of the manganese to a brown higher oxide. The solution supposed to contain the ammonium salt is boiled with sodium hydroxide in a flask provided with a rubber stopper and a delivery tube, the vapour issuing from the latter being made to impinge upon a piece of filter paper moistened with a solution prepared by dissolving 2 gm. of manganous sulphate in 200 cc. of water and adding 5 cc. of hydrogen peroxide solution. The presence of ammonia is indicated by a brown stain on the paper, which assumes a deep purple colour when moistened with a drop of the organic reagent ; the purple colour is produced even when the quantity of ammonia present is so small that no brown stain is apparent. Steam has no action upon manganese sulphate, but produces a slight brown stain with manganese acetate when no ammonia is present. It is stated that by means of this test the presence of 0·02 mgm. of ammonia may be easily detected and that probably half this amount would be indicated by careful working." R. J. Carney, *J. Amer. Soc.*, 1912, XXXIV., 32-35. —*Journal of the Society of Chemical Industry*, vol. xxxi., p. 153-4. (J. G.)

Revised Atomic Weights, 1912.—" The International Committee on Atomic Weights has issued its annual report on the atomic weights to be used in 1912. The committee has investigated researches that have been made on the atomic weights of nitrogen, sulphur, chlorine, iodine, sodium, calcium, mercury, vanadium, tantalum, tellurium, iron, uranium, scandium, neodymium, erbium, argon, and niton, the last previously known as radium emanation. Many of the researches simply confirm existing atomic weights. In the cases of calcium, mercury, vanadium, tantalum, iron, and erbium the researches have been so positive that the committee has deemed it advisable to make changes, which, however, except in the cases of mercury and vanadium, are only slight.

The changes that have been made are based on the following researches. Calcium. From analyses of the chloride and bromide (*Journal Amer. Chem. Soc.*, 32 : 1577, 1910 and 33 : 28, 1911). Richards and Honigschmid obtained the figures 40·07 and 40·074, respectively. The figure 40·07 has been adopted. Iron. From analysis of ferrous bromide (*Jour Amer. Chem. Soc.*, 33 ; 319, 1911), Baxter, Thorvaldson, and Cobb obtained the figure 55·838, while Baxter, Thorvaldson (the same, p. 337), starting from meteoric iron, found the value to be 55·836. The figure 55·84 has been adopted. Mercury. As suggested as likely in the review of last year's report, Eastley, from further researches on the analysis of chloride (*Jour. Amer. Chem. Soc.*, 32 : 1117, 1910), confirms his previous conclusion that the atomic weight of mercury was altogether too low. He finds that it should be 200·63, instead of the old figure, 200. The committee has adopted the value 200·6. Vanadium. By heating sodium vanadate in a stream of hydrochloric-acid gas and converting the sodium into chloride (*Jour. Amer. Chem. Soc.*, 32 : 1603, 1910), McAdam obtained 51 as the atomic weight of vanadium. This figure has been adopted. Tantalum. By hydrolysis of the penta chloride (*Jour. Chem. Amer. Soc.*, 32 : 1127, 1910), Balke has determined the ratio $2TaCl_5 : Ta_2O_5$ and obtained the figure 181·52 as the atomic weight of tantalum. The rounded off value of 181·5 has been adopted. Erbium. From analysis and synthesis of the sulphate of neoerbium (*Berichte*, 43 : 2635, 1910) Hoffmann obtained the figure 167·68 as the atomic weight of erbium. The value of 167·7 has been adopted. Niton. By the aid of a microbalance (*Proc. Roy. Soc.*, A, 84 : 536, 1910), Gray and Ramsay have determined the density of the emanation from radium, which they have named niton. Niton, with an atomic value of 220·4, is the only new element to find a place in the table this year.

To recapitulate, then, the new atomic weights adopted by the International Committee are : Calcium, 40·07 ; erbium, 167·7 ; iron, 55·84 ; mercury, 200·6 ; niton, 222·4 ; tantalum, 181·5 ; and vanadium, 51·0."—*Mining and Scientific Press*, Dec. 16, 1911, p. 778. (H. A. W.)

The Electrolytic Determination of Lead in Large and Small Amounts, using a Gauze Cylinder Anode.—" Among the articles published by various workers on electrolytic methods for the determination of lead, there seems to exist a lack of description of exact conditions necessary for the rapid deposition of PbO_2 on a gauze cylinder anode with a rotating cathode.

Prof. E. F. Smith in his book on Electro-Analysis recommends the use of a sand-blasted platinum dish to secure an adherent deposit, but in a technical laboratory the use of a dish is not practicable.

J. G. Fairchild describes in the *Journal* of Ind. and Eng. Chemistry conditions that must be observed to get a good deposit on a smooth cylinder with stationary electrodes. This is of merit, although the amount of PbO_2 that can be obtained in an adherent condition is limited to 0·3 gm.

The conditions that are good in these cases are not satisfactory when different forms of electrodes are used.

At present, most of the metallurgical laboratories that are using electrolytic methods are furnished with gauze cylinders and rotating propeller-shaped anodes. The description of conditions that have been found necessary in this laboratory to obtain a good deposit of PbO_2 on a gauze cylinder will, no doubt, be of some value to those who are using this device.

The method, as it is used for lead base babbitts, is as follows : The tin and antimony are separated from the lead and copper by means of sodium sulphide. The sulphides of lead, and copper when it is present, thus obtained, are dissolved in a known amount of HNO_3 (10 cc.), filtered, the sulphur and filter paper burnt, the residue dissolved in a little HNO_3 and added to the main solution, which should be in a tall electrolytic beaker of 200 cc. capacity. Sufficient HNO_3 is added to make a total of 45 cc. of strong HNO_3 in 150 cc. of solution.

The electrolyte is heated till it starts to boil and electrolyzed with 3 amp. for a minute or two, or till the deposit is brown in colour, then with 5 amp for 20 to 30 min.

The deposit must be dried at the usual temperature for PbO_2, viz. : 230° C. for 20 to 30 min., depending on the amount of deposit.

The conversion factor is used as given by Prof. Smith, 0·8643, instead of the theoretical 0·8660.

The deposit will be so adherent that it will not come off when rubbed with the finger ; a finger nail may scratch it off, however, so the electrode must be handled with care while disconnecting, drying, and dessicating. It is a good idea to have in the drying oven an adjustment for hanging the electrode. A binding post fastened to the wall of the oven will answer the purpose very well.

When other metals, especially copper, are present in large amounts, and Pb does not exceed 10-12%, the amount of HNO_3 may be considerably reduced, and this is usually the case with brasses. In this case to the filtrate from metastannic acid sufficient HNO_3 is added to have 10 cc. in a total volume of 150 cc. The solution is heated to boiling and electrolyzed with 5 to 6 amp. for 15 min. The deposit will range in colour from golden brown to dark brown and may be dried completely at the usual temperature of the drying oven, viz., about 105° C.

If copper is to be determined after lead it will be found better to have more HNO_3, say about 20 cc. in 150 cc. of solution. Then less Cu will be deposited on the cathode, which must be dissolved off before neutralising and subsequent acidifying the solution for Cu deposition.

At present this laboratory uses a magnetic rotating apparatus called the Frary solenoid, a description of which may be found in "Electro-Analysis," by Smith, and in "Technical Analysis of Brass," by Price and Meade. It has proved to be the best apparatus adopted for the deposition of PbO_2 especially when lead is present in very large amounts. The conditions are identical with those described for a rotating anode device. With this apparatus a number of depositions of PbO_2 are made daily, the amount of PbO_2 deposited ranging from 0·0005 gm. to 2·00 gm., without any difficulty.

In conclusion, therefore, the author feels justified in highly recommending the solenoid, as it gives the most satisfactory deposits, not only of PbO_2 but of all the metals which can be electrolytically determined." — BRUNC WOICIECHOWSKI, *Metallurgical and Chemical Engineering*, February 1912, p. 108. (F. W. W.)

PERMANENCE AND DELICACY OF THE FERRIC CHLORIDE REACTION FOR SALICYLIC ACID AND ATTEMPTS TO DETERMINE THE FREE ACID IN ASPIRIN AND OTHER TABLETS.—" When the dilution of salicylic acid in aqueous solution is greater than 1 in 50,000, the violet blue colouration produced by ferric chloride fades into a reddish violet colour in about two hours, finally turning brownish-yellow. The ferric chloride solution is prepared by diluting the *Liquor Ferri sesquichlorati* of the German Pharmacopœia with 25 volumes of water, a single drop being used in each test. With stronger solutions of salicylic acid, the colour is more or less permanent. If 25 cc. of the solution of salicylic acid are put into a test-tube 20 mm. wide, the acid can be detected in a dilution of 1 in 900,000 by a distinct violet colouration after the addition of one drop of the dilute ferric chloride solution. The amount of free salicylic acid in tablets of aspirin or acetylsalicyclic acid can be roughly determined by means of this reaction. The tablet is ground up with 25 cc. of water, a drop of the ferric chloride solution added, the whole shaken and filtered into a test tube 20 mm. in width. The colour is then compared with that produced by solutions of salicylic acid of dilutions of from 1 : 50,000 to 1 : 200,000, tested in the same way. The amount of free salicylic acid found in several samples of aspirin tablets varied between 0·125 and 0·360 mgm." —H. LINKE, *Apoth.-Zeit.*, 1911, *26*, 1083-1085.—*Journal of the Society of Chemical Industry*, Vol. 31. p. 93. (J. G.)

METALLURGY.

PROPOSED TREATMENT OF SULPHIDE ORES WITH NITRIC ACID, OR SO-CALLED RANKIN PROCESS.— " The difficulties attending the economic treatment of ores containing the sulphide minerals of several metals have resulted in classifying such ores as refractory. This term is generally taken to mean that these ores do not yield readily to known methods of treatment, and consequently the newer developments in metallurgy are heralded as means of successfully treating refractory ores.

The proposal to treat mixed sulphide ores with nitric acid is an endeavour to solve this problem in an economical way. The recovery of the nitric acid is a vital feature of the process. In its present state of development it represents the original idea of Mr. H. D. Rankin as modified and improved by his coworkers.

In the following description of the process reference will be had to the treatment of an ore containing some form of the sulphides of iron, copper, lead and zinc, with accompanying gold and silver, either native or combined. The statements given represent the claims for the process as outlined to the writer by Mr. E. H. Westling, metallurgist for the company controlling the process. Experimental work has been done at Pittsburgh, Pa., and Lake City, Colo., and it is announced by the company that a commercial plant is to be built and operated immediately at Chloride, Ariz.

With the exception of gold, the products of the process are not metals ready for refining, but salts of the metals equivalent to high-grade ores suitable for reduction by present processes of smelting. In some instances the precipitated salts may find commercial use as such, as in the case of iron and copper, but for the most part they must be reduced to metals. Briefly, the primary products of the process are: Gold in the form of amalgam, silver as silver chloride, iron as hydroxide, copper as basic carbonate, zinc as hydroxide and lead as sulphate. Thus the process is designed to effect a chemical separation and concentration of the valuable constituents of an ore in a more economical manner than might be done in other ways.

A number of objections and queries will at once arise in the mind of the investigator as to the feasibility of the process. The complete precipitation and washing of salts in large quantities: the economical regeneration and recovery of nitric acid; the completeness of the various reactions; the operation of a process with a gas under pressure; the economical operation of a process combining many separate operations; and the recovery of unfinished products which must stand further cost of reduction and refining—all these are points which call for proof by commercial operation on a large scale. The sponsors of the process, however, feel sufficiently assured on these points and claim that the method can be operated profitably.

Description of Plant.—The process naturally divides itself into three parts: The decomposition of the ore, recovery of the metallic salts and recovery of the nitric acid.

The decomposition of the ore is effected in a digester of cast iron lined with an acid-resisting mixture. This piece of apparatus is egg-shaped and supported in a vertical position. Within is an air lift agitator made of aluminium, connected by aluminium pipe with an air compressor outside. The proportional dimensions of air lift and digester are about the same as in a Pachuca vat. Above the digester, and tightly connected with it, is the ore bin, which can be tightly sealed. In the connection between digester and bin is a revolving cup feeder. In a suitable position above the digester is a receptacle for nitric acid.

The recovery of the metallic salts is accomplished in a series of precipitating and settling tanks, with such filtering and washing apparatus as may be needed. Ordinary amalgamating devices are provided for gold recovery, and a concentrating table is used for the separation of lead sulphate from the gangue.

The acid recovery system consists of a series of condensers and water towers. Aluminium pipe is used in all connections and cooling coils, and the tanks and towers are protected with acid-resisting mixtures wherever exposed to the acid gases. The water towers are constructed of paraffined redwood or cypress and filled with clean quartz. The first of the three towers is half full of water and the others are two-thirds full. At suitable places in the aluminium pipe are two indicators to show the condition of the gases at those points. These indicators are glass sleeves surrounding openings in the pipe, and the first is placed ahead of the first water tower and the second beyond the last tower.

Treatment of Ore.—Although the treatment of ore and recovery of acid are simultaneous operations, they will be described separately for the sake of clearness.

The process is worked intermittently on successive charges of ore and acid. Six tons of ore per charge is a convenient unit of capacity, and the time of treatment of each charge is between two and three hours.

The ore should be ground to about 30 mesh size, and contain at least 2·5% to 3% sulphur. If the sulphur content is below that figure the ore should be concentrated before treatment. At least 5% sulphur has been found necessary if the reaction is to proceed without the application of heat. With 5% or more sulphur the reaction with nitric acid generates sufficient heat for complete decomposition of the ore, but with less than 5% it is necessary to heat the mixture. This is usually done by agitating with live steam instead of air for five or ten minutes at the beginning and end of the treatment.

The acid is run into the digester in quantity proportional to the amount of sulphur in the ore. It is claimed, however, that if pyrite is present in considerable quantity, 5% less than the theoretical amount of acid can be used, as the ferric salts act as catalyzers in oxidising the sulphides to sulphates. It is customary to use acid of 1·15 to 1·42 sp. gr. The recovered acid has a sp. gr. of 1·25 to 1·35 and is used at that strength.

With the necessary charges of ore and acid in the sealed ore bin and digester respectively, the feeder is started which slowly delivers the ore in small quantities into the acid. At the same time the air agitator is started and the feeding and agitation continue until the sulphide minerals have been converted into sulphates, all of which are soluble excepting that of lead. Some sulphuric acid may be used with the nitric if the ore contains lime, or to prevent the formation of basic iron sulphate. It is claimed that the conversion of sulphides to sulphates is practically complete, only about 2% of the nitric acid remaining as nitrates of the metals. The following equation is typical of the reaction on which the process is based.

$$3PbS + 8HNO_3 = 3PbSO_4 + 8NO + 4H_2O$$

Throughout the treatment of a charge of ore the digester and acid recovery system are under about 25 to 30 lb. pressure. When the reaction is completed the digested mass is drawn into a tank where the solution is separated from the gangue by filtration or decantation. The residue is washed and passed over an amalgamated plate and Pierce amalgamator for the recovery or the gold. The lead sulphate is then separated from the gangue by concentration on a Wilfley table, the sulphate flowing off as a slime and collecting in settling launders. Naturally it will be contaminated with the fine gangue formed in crushing the ore, but it forms a high grade lead product for smelting.

The metals in solution are precipitated successively in a system of tanks, and the precipitated salts are washed and dried. Silver first is precipitated with salt, giving silver chloride which must be reduced to metallic silver. Iron next is precipitated with pulverised limestone (30 mesh) giving a fine granular precipitate of ferric hydroxide which settles rapidly. This precipitation is carried out at a temperature of 55° C., which is about the normal temperature of the solution at this stage of the process.

A remarkable claim is made in connection with the precipitation of iron and the subsequent precipitation of copper. Ground limestone is used as the precipitant in each case, but the temperature is 20° C. higher for copper than for iron. It is claimed that at 55° C. limestone will precipitate iron without copper, and that upon raising the temperature of the solution to 75° C. the same precipitant will throw down copper as basic copper carbonate. Wherefore, after the precipitation and separation of iron the solution is heated by passing live steam through it until a temperature of 75° C. is obtained, when copper is precipitated.

The iron precipitate has no value unless circumstances should warrant its use as a paint pigment. A market might be found for the copper precipitate as such, but in all probability if any considerable quantity were produced it would have to be reduced to copper.

With iron and copper removed the zinc is precipitated with milk of lime, giving zinc hydroxide which is to be calcined and smelted. The zinc precipitate will be free from iron but may contain some manganese. It would seem that, inasmuch as lime in some form is the precipitant for iron, copper and zinc sulphates, certain amounts of calcium sulphate must contaminate the precipitates. Calcium sulphate is soluble at ordinary temperatures in about 500 parts of water, the solubility increasing slightly until a temperature of 100° C. is reached, when it decreases rapidly. Hence the salt will precipitate unless enormous volumes of solution are used.

The precipitating tanks for these operations are fitted with pipes for air agitation, bronze pipe being used in all precipitation where copper is still in solution. The entire precipitation system is preferably arranged in duplicate to facilitate cleaning up.

Nitric Acid Recovery.—The principle upon which the recovery of nitric acid is based is shown in the following cycle of reactions. As shown before, the treatment of a sulphide with nitric acid gives nitric oxide as one of the products. Nitric oxide and oxygen unite to form nitrogen peroxide, which is soluble in water at ordinary temperatures, forming nitric acid and nitric oxide. The latter then combines with more oxygen and the cycle is repeated.

$$3CuS + 8HNO_3 = 3CuSO_4 + 8NO + 4H_2O$$

$$NO + O = NO_2$$

$$3NO_2 + H_2O = 2HNO_3 + NO$$

The oxygen necessary for the reaction is supplied by the air used in agitating the charge of ore and acid. The nitric oxide and oxygen are constantly being forced out of the digester through the acid recovery system, the oxygen being supplied in sufficient quantity so that the gases escaping into the air at the end of the system will show an excess of at least 2% O. It is claimed that 95% of the acid used is recovered.

The gases first enter a large narrow cylinder having about two-thirds the volume of the digester. The function of this so-called mixer is to allow time for thorough mixing of the gases and formation of nitrogen peroxide. Some nitric acid is formed here by reason of the fact that when the reaction in the digester is well under way steam will issue with the other gases, giving the right conditions for the formation of some nitric acid. It is claimed that about 10% of the acid is recovered here.

The gases are next cooled in an aluminium condenser or cooling coil, where another 10% recovery is made. Following the cooler is the first water tower, through which the gases bubble and where the greatest part of the recovery is made, amounting to 60%. Between the cooler and first tower is placed an indicator, where the gases should show reddish brown in colour. Following the first tower is another cooler where about 3% of the recovery is made. Finally, the second and third towers make about 12% recovery. The acid drawn from the mixer and coolers is 1·25 to 1·35 sp. gr., while that in the towers gradually builds up to 1·35 to 1·40 sp. gr. before it is drawn for use. The first tower builds up in strength most rapidly and when its contents are withdrawn they are replaced by those of the second tower, and so on. The gases issuing from the last water tower pass through a second indicator, where they should appear colourless if the system has been working properly.

The cost of chemical treatment is determined by the quantity of sulphur in the ore. The acid loss is claimed to be only 5%, and the precipitants used are inexpensive In addition there will be the usual cost for preparation of the ore, pumping and agitating solutions, filtering, drying and marketing the products.

The results of the first attempt to operate the process on a commercial scale will be awaited with interest." — FROM EDITORIAL CORRESPONDENCE, *Metallurgical and Chemical Engineering*, Nov. 1911, p. 571-573. (G. F. A)

EFFECT OF MOISTURE ON VOLUME OF SAND.— " A quantity of dry Michigan City sand was measured, 10% of water by volume added to it, the mass mixed over by hand, and an attempt made to put it back in the same measure from which it had been poured. It was found that 40% could not be returned to the measure. Experiments were then made by taking measured amounts of sand and adding first 1% and then 2% of water, and so on up to 20% or more. It was found that at first each addition of water increased the volume of sand until about 10% had been added, and that beyond this the volume began to decrease once more, until the volume of water was so great that the sand could settle out of it, when the volume of sand became very nearly the same as that for dry sand.

The explanation is simple. In dry sand the grains slide past each other until the maximum number of bearing points are in contact. When there is just sufficient water present to form a film over each grain of sand, two grains coming in contact adhere and will not move further without the application of force. Fairly hard ramming even will not drive the sand back to the former volume.

The practical application of this is that if the men in the core room are allowed to take sand as it comes and measure it by volume they will be getting constantly varying ratios between the sand and the binder. If a given volume of dry sand is taken it will contain a certain number of grains and there will be a certain ratio to the binder. If the same volume of wet sand is taken there will be a very much smaller number of grains and hence a larger proportion of binder or smaller binding ratio, which will result in a stronger core, providing the mixing is properly done.

Table 4.—Tests with New England Bank Sand from Waterbury, Conn.

Sand, by Volume.	Water, by Volume.	Water by Weight, per cent.	Sand Water Added, Volume.	Increase in Volume, per cent.
500	5	0·77	590	18
500	10	1·54	680	36
500	15	2·31	750	50
500	30	4·62	790	58
500	45	6·93	830	66
500	60	9·24	810	62
500	75	11·55	810	62
500	90	13·86	800	60

Table 5.—Tests with Rockaway Beach Sand, Rockaway Beach.

500	5	0·68	700	40
500	10	1·36	740	48
500	15	2·04	730	46
500	30	4·08	700	40
500	45	6·12	810	62
500	60	8·16	810	62
500	1000	Flooded	470	—

—H. W. LANE, *Page's Weekly*, Jan. 19, 1912, vol. xx., p. 110. (J.E.T.)

TUBE CONCENTRATOR.—" The plant of 119 tube concentrators, built as an auxiliary to the Morning mill of the Federal Mining & Smelting Co., at Mullan, Idaho, has been in operation since May 15th, 1911, and the results have been so satisfactory that the company is negotiating for the building of another unit of equal capacity to that now in operation. The original plant, as designed by O. B. Hoffstrand of the American Direct Concentrator Co., had an estimated capacity of 50 tons per day, but this has been exceeded. During the month of October the average was 61 tons per day, the month's output having been 200 tons of concentrate carrying 48% to 49% zinc. According to tests made in operating the mill it has been demonstrated that by keeping the flow surface in the tubes crowded, or fully covered, the grade of concentrate produced is higher than that obtained while running with lighter feed. The original practice was for some of the tubes to re-treat the middling of the other tubes, and this no doubt will be eventually the accepted practice after the present equipment has been doubled ; but during the past three months all tubes have been given direct feed, with no re-treating of the middling, and this has resulted in raising the capacity of the present plant to 95 tons per day, giving a concentrate product of close to 300 tons per month that is estimated to sample 49% zinc. The object of this temporary practice is to recover as large a tonnage of zinc as possible at a sacrifice of the higher saving effected by the aid of reconcentration. The material treated consists of the middling and tailing from Wilfley tables and the tailing from vanners, containing zincblende, siderite, and silica, also a small percentage of galena that is mechanically associated with the blende. This flotation process effects a separation of the zinc sulphide from the siderite, the former floating on the surface of the water, the latter sinking and passing out with the gangue. The use of sulphuric acid in the process is found to be an aid, as it serves to cleanse the surfaces of the sulphide particles, making them more amenable to flotation ; but the quantity of acid used does not exceed 2c. per ton treated. The operation of the plant, especially while some of the tubes were used in reconcentrating the middling of the others, has given results equivalent to those indicated by the original tests made by the General Engineering Co. before the mill was built. Mr. Hoffstrand, who designed the plant, has had general charge of its operations for the Federal company."—W. A. Scott, *Mining and Scientific Press*, Dec. 23, 1911, p. 809. (H. A. W.)

Some Notes on the Action of Heat on Fireclay.—"In studying the changes which occur when firebricks are heated, it is interesting to start with the clay itself. Fireclay may be conveniently regarded as a mixture of clayite and kaolinite, which for convenience is here called clayite, along with quartz, colloidal silicic acid, and fluxes—mica and felspar. The clayite largely predominates. We need not here consider the organic matter, and a few other minerals which occur in small quantities.

When a sample of dried (110°) fireclay is heated on a gradually rising temperature, with a thermo-couple immersed in the clay and connected with a recording pyrometer, the heating curve shows a terrace just below 200°. This corresponds with the decomposition of the colloidal silicic acid. If the clay contains no colloidal silicic acid (and one or two hydrated alumino silicates) there will be no terrace.

No special change occurs in the heating curve until just over 500°, when the clayite decomposes into free silica, free alumina, and water. The reaction can be symbolised :—

$$Al_2O_3\ 2SiO_2\ 2H_2O \rightarrow Al_2O_3 + 2SiO_2 + 2H_2O$$

This change produces another terrace in the heating curve. Both changes are attended by an absorption of heat, and the terraces are produced by a retardation in the rate of heating.

Just over 800°, the alumina which was formed by the decomposition of the clayite undergoes some kind of change, which is accompanied by an evolution of heat. The result is a kink in the heating curve. The heating curve of a typical fireclay thus shows three well-defined changes :—

1. Dehydration of silicic acid and silicates.
2. Decomposition of clayite.
3. Polymerisation of alumina.

If the clay be heated still further, the alkalis begin to volatilise, and under suitable conditions of temperature, the free alumina recombines with some of the silica, forming an aluminium silicate Al_2O_3, SiO_2 which appears in the fired clay in the form of needle-like crystals of sillimanite. These crystals form between 1,200° and 1,300°. At still higher temperatures the clay begins to sinter to a more or less stone-like mass, softens at, say, 1,670°, and melts at just over 1,700° to a brown coloured viscid fluid.

I might summarise these statements thus :—

110°	Loss hygroscopic moisture, decomposition, silicic acid, and other colloids.
500°	Clayite decomposes into free alumina, free silica, and water.
800°	Alumina polymerises.
1,200°	Alumina and silica recombine to form sillimanite.
1,500°	Sinters to a stone-like mass.
1,650°	Clay softens and loses its shape.
1,700°	Brown viscid fluid.

You may consider some of these statements require proof. The statement regarding the dehydration of silicic acid is based on the development of a terrace just below 200° when silicic acid alone and some alumino silicates (not kaolinite and clayite) are heated ; and also on the action of dilute solutions of sodium carbonate on such clays.

The theory that clayite and kaolinite decomposes at 500° into free silica, free alumina, and water, and not into an aluminum disilicate and water :—

$$Al_2O_3\ 2SiO_2\ 2H_2O \rightarrow Al_2O_3\ 2SiO_2 + H_2O$$

is mainly based on the following facts :—

(*a*) Clay dehydrated at 500° can be partially rehydrated—this is probably a rehydration of alumina and silica.

(*b*) The changes in specific gravity, hydroscopicity, and solubility in acids of clayite calcined at temperatures between 500° and 1,000°, resemble the changes in alumina derived from calcined aluminum nitrate.

(*c*) The acceleration in the heating curve just over 800° is also produced by alumina derived from calcined aluminum nitrate.

Numerous statements to the effect that clay decomposes into Al_2O_3. SiO_2 ; Al_2O_3. $2SiO_2$; $Al_2O_3.3SiO_2$, etc., occur in clay literature, but I have not found one to be based on any experimental evidence, and the statements are therefore of no more value than mere assertions.

It is also interesting to note that this decomposition of alumina silicates into free alumina and silica is not confined to clays, because it is well known that many native silicates which are very slowly attacked by acid break down quickly if the silicate be first calcined at dull redness. The explanation I have just indicated furnishes a key to the most probable interpretation of the phenomenon. Similarly, the so-called polymerization of many oxides on calcination, whereby they become very resistant to the action of solvents, is well known to analysts.

The formation of sillimanite has been demonstrated by the optical properties of the crystals, and by chemical analysis. The crystals have been isolated. If, therefore, we reject the view that clayite decomposes as I have stated, we still have to assume a decomposition of the hypothetical Al_2O_3, $2SiO_2$ just over 1,200°, in order to account for the formation of these crystals, Al_2O_3, SiO_2. The hypothesis I have indicated is the simpler interpretation, and fits all the facts known to me.

The contraction which takes place when a fireclay is heated is the joint effect of several changes. Crystalline quartz, for instance, expands about 14% ; crystalline felspar on fusing expands about 6% ; alumina contracts nearly 40% ; the silica undergoes very little change. Hence, the observed contraction is the resultant effect of a number of changes—some expansions, some contractions.

When air dried clay is gradually heated, there is a slight expansion, just over 110° ; then a slight contraction, which continues up to 500° ; after that there is a slight expansion, and this is followed by a steady contraction. The two expansions here noted appear to correspond with a slight disintegration of the clay when the hygroscopic moisture is driven off at about 110°, and when the clay decomposes just over 500°. If a piece of clay were to be heated too rapidly while these changes are going on, the piece might rupture owing to the development of the so-called 'fire-cracks.'

The main contraction of a firebrick above about 900° is due to a change analogus with the universal tendency of liquids to assume a shape which presents the smallest possible surface area—namely, a sphere. As soon as the fluxes in the clay have softened, they commence to draw the more refractory grains into closer and closer contact. The whole piece is trying, so to speak, to assume the form of a sphere. Contraction only ceases when the grains have been pulled close together by the fluxes, so that the whole forms a compact non-porous mass. When the clay is high in fluxes, the body will appear like a vitreous stoneware.

All the time the fluxes are doing this work, they are dissolving the more refractory clay and quartz. If the clay be low in fluxes, the fluxes will soon become saturated with clay and quartz, and form a solution which is practically infusible at the given temperature. The brick must then be heated to a higher temperature before the solution can melt and the fluxes continue their work (1) dissolving the more refractory materials, and (2) pulling the grains into closer contact.

There is a curious difference in the contraction changes which occur during the repeated firing of the two types of fireclay, or rather of firebricks, etc., made from the two types of fireclay. The vitrifiable fireclays when subjected to the gas engineers' test give a rather high contraction, and the manufacturer has more difficulty in keeping his goods up to specification than when the non-vitrifiable fireclays are used.

The contraction follows a definite law. If $n-1$ represents the number of times the brick has been fired, and the time-temperature curve of each firing is the same, the contraction on the nth firing is, according to my measurements,

$$\text{Contraction} = \frac{an}{1+bn}$$

where the constants a and b depend upon the time temperature curve, and on the nature of the mixture used in making the firebricks.

During the past year I have investigated the value of these constants. I worked with typical fireclays, and varied the amount and grain-size of the grog. I found that the contraction of the fireclays low in fluxes soon reached a maximum when the firing conditions were constant. After that, the fired bricks began to expand. If the firing temperature be then raised, contraction again sets in, reaches a maximum, and is followed by a slight expansion. This expansion puzzled me for some time. So far as I have been able to make out, the fluxes are saturated when the maximum contraction occurs, and the fluxes are no longer mobile enough to do their work pulling their grains together. The brick has then attained its maximum crushing strength. Any further heating is followed by a reduction in the crushing strength. In other words, the expansion is produced by a disintegration of the brick. The brick also loses a small amount of alkali each firing, as indicated above.

Several important deductions can be drawn from these observations, but, before yielding to the obvious temptation, I prefer at present to wait until the ideas here put forward have been subjected to more searching inquiry. Most of the facts and views indicated above have been described in detail in papers in the Transactions English Ceramic Society.

It will be noted that I have used the terms vitrifiable and non-vitrifiable fireclays. These terms, or something equivalent, are necessary because the majority of fireclays can be grouped into one or other of two classes. By a vitrifiable fireclay, or a fire-clay high in fluxes, I understand a fireclay which vitrifies to a stone-like mass at a comparatively low temperature, say, cone 18, and yet retains its shape above, say, cone 28. The non-vitrifiable fireclays are comparatively low in fluxes, and exhibit the phenomenon of maximum contraction indicated above. I am well acquainted with both types of fireclay. It is a great mistake to suppose that because a clay is comparatively high—say, 5% or 6% fluxes—it will not be refractory when tested according to the gas engineers' specification. Some of the Yorkshire fireclays, and some of the Dorset ball clays give remarkable results. For instance, I know a Dorset ball clay which vitrifies at cone 10 to a stone-like mass, and yet cones made from the clay retain their shape at cone 33.

If the vitrifiable fireclays exhibit no abnormal tendency to split or dunt on cooling, they make valuable bricks, crucibles, etc., resisting the action of molten fluxes at high temperatures. But these clays show to best advantage when blended with other less vitrifiable clays. What I called the vitrifiable fireclays do not bloat at high temperatures, whereas many or rather most vitrifiable clays develop a spongy structure owing to development of gases or vapours when heated a little above their temperature of vitrification.

To summarise these remarks on the volume changes which occur on heating fireclays :—

110°	Slight expansion.
110°-500°	Slight contraction.
500°	Slight expansion.
600°	Contraction of alumina sets in.
800°	Expansion of quartz commences.
1000°	Fusion and expansion of felspar and main contraction commences.

The subsequent behaviour of the fireclay depends on whether a vitrifiable or non-vitrifiable fireclay is in question.

Vitrifiable fireclays contract to a maximum below, say, cone 18, forming a stone-like, non-porous mass, which does not bloat below the softening temperature, say, cone 28 upwards.

Non-vitrifiable fireclays exhibit a maximum contraction which varies with the temperature when the maximum contraction has been attained ; subsequent heating at *that* temperature is attended by a slight expansion, and a slight disintegration of the brick. Subsequent heating at a *higher* temperature is followed by a further contraction which attains a maximum as before.

About six years ago I put forward the view that the decrease in the crushing strength of certain refractory goods on repeated firing is due to two conditions :—(1) The gradual loss of alkalies by volatilisation. (2) The strains set up owing to the growth of local patches by volatilisation. A substance in a crystal condition occupies a less volume than the same substance in a vitreous condition. If therefore local patches of crystals are produced in a vitreous body it follows that local contractions must be produced, and local strains leading to rupture must also be produced.

Zoellner, a couple of years ago, put forward the view that the crystals make the body tougher and less liable to fracture. At first sight it does look as if the anastomosed crystals would make the body tougher, but when the comparatively great contraction strains set up during the passage from the vitreous to the crystalline condition are taken into consideration it seems to me that Zoellner's view is not sufficiently comprehensive, nor does it fit the facts.'—J. W. MELLOR, *The Birmingham Engineering and Mining Journal*, July 1911, pp. 459-464. (F. W. W.)

HEATING CYANIDE SOLUTIONS "In the issue of January 6, under heading of 'Heating Solutions,' Alfred James in his 'Progress in Cyanidation of Gold and Silver Ores During 1911,' expressed a desire for data on benefit of heat on extraction. The following tables of comparison, showing extraction on Belmont ore, with different degrees of temperature, may be of interest. The benefit shown has been borne out to our entire satisfaction in practice.

I. Ore Au 0·05 oz., Ag 18·2 oz., Total Value $10·15

Agitator.	Agitation, Hours.	Temp.	Residue.		
			Au. oz.	Ag, oz.	Val.
Belmont ...	60	Cold 60°	0·015	3·1	$1·86
,, ...	60	Warm 90°	0·01	1·7	1·06
Pachuca ...	60	Cold 60°	0·02	3·8	2·32
,, ...	60	Warm 90°	0·015	2·1	1·36
Bottle	60	Cold 64°	0·02	3·4	2·12
,, ...	60	87°	0·01	2·5	1·49
,, ...	60	109°	0·01	1·7	1·06

II. Ore Au 0·05 oz., Ag 20·6 oz., Total Value $11·30

Agitator.	Agitation, Hours.	Temp.	Residue.		
			Au. oz.	Ag. oz.	Val.
Belmont ...	48	+90°	No sample taken.		
Pachuca ...	48	+90°	0·015	2·6	$1·61
,, ...	48	60°	0·02	4·5	2·67
Belmont ...	69	+90°	0·01	1·7	1·06
Pachuca ...	69	+90°	0·015	2·1	1·36
,, ...	69	60°	0·02	3·8	2·32

Material agitated in each of the above tests was taken from collectors and contained the gold and silver remaining after concentration and that dissolved in crushing after conveying at head of mill.

To maintain a heat of 90°F. in the agitators, boilers are run two 8-hr. shifts during summer and three 8-hr. shifts during winter, being fired with crude oil at a cost of $2·05 per barrel at the mill, making cost per ton of ore treated $0·161 for past nine months. Degrees of heat are marked +90 to show they were kept as near 90° as possible. varying from 90° to 95°."—A. H. JONES, *Mining and Scientific Press*, Jan. 27, 1912, p. 176. (H. A. W.)

ADVANTAGES OF METALLURGICAL SPECIALISATION.—"Referring to the growth of the mineral industry, he said that the world's production of gold between the year 1861 and the year 1890 averaged less than 5½ million ounces per annum; but in 1891 the cyanide process was introduced on the Rand, and since that date the world's annual production of gold had enormously increased, amounting for the year 1910 to about 23,000,000 ounces. Some credit for this increased production belonged, no doubt, to the mining engineer, who had applied himself so successfully to effecting economies in the cost of mining, raising and transporting ore; but the larger credit was due to the metallurgical engineer who had so adapted the cyanide process to special conditions, that low grade deposits, which could not otherwise have been profitably exploited even subsequently to the introduction of that process on the Rand, had been worked to great advantage.

In the growing tendency throughout all trades and professions to specialise, the members of their profession, and more particularly those belonging to their institution, had been among the foremost, with the result that they had contributed appreciably to the elevation of their professional status. For more years than he could remember, not only was a mining engineer supposed to know all about mineralogy, geology, and the working of mines (for which it was necessary to have a practical knowledge of mechanical and electrical engineering), but metallurgy in all its branches was included among his essential requirements. Even at the present day, there were no important universities of 'metallurgy,' although they had their school of 'mines,' and the 'Mining' academy at Freiberg, the fact was overlooked that the metallurgist was a specialist, whom they, as mining engineers, gladly recognised as being no less important to the success of mining enterprise than themselves."—EDWARD HOOPER, President, Inst. of Mining and Metallurgy.—Inaugural address at Annual Meeting of Inst. of Mining and Metallurgy, on March 21st, 1912. Also p. 280 of *South African Mining Journal*, April 20th, 1912. (W.R.D.)

VACUUM FILTRATION ON THE RAND.—"The following notes are taken from a report by Mr. F. L. Bosqui, consulting metallurgist to the Central Mining and Investment Corporation, Ltd., upon the operations of the Butters filter plant at the Bantjes Consolidated, New Modderfontein and Crown Reef mines. The details are fully given, because they are of unusual interest not only with regard to the working of the Butters process, but generally with reference to the possibilities of slime treatment upon the Rand. Costs, it will be observed, range from 3·13d. to 4·07d. per ton of slime treated, and extractions between 90·1% and 95%. There are, of course, various savings which are not dealt with in this report, as for instance that arising from the smaller proportion of solution per ton of dry slime which it is necessary to precipitate, and that consequent upon the more rapid carrying out of the slime cyaniding process, as compared with, say, the ordinary decantation process:—

Bantjes Consolidated Mines.

Total cost of slime treatment (including Butters filter and Brown agitators).

	May and June, 1911. Without Filter.	Aug. and Sept., 1911. With Filter.
Slime treated (tons)	16,640	17,890
Cost per ton of slime treated	1/5·34d.	2/0·17d.

Cost of operating Butters filter (i.e., additional cost incurred over old system).

	August, 1911.	September, 1911.
Native labour	£4 6 10	£7 10 0
Chemicals	28 13 0	19 9 4
Electrical power	41 6 9	41 14 6
Air power	11 2 4	10 14 10
Assaying, etc.	47 4 5	32 5 3
General stores	—	8 1 6
	£132 13 4	£119 15 5
Cost per ton slime treated	3·425d.	3·347d.

Extraction Results.

1911	Tons Slime Treated.	Original Value. Dwt.	Residue Total Dwt.	Residue Washed Dwt.	Percentage Extraction.
May	8,625	2·45	·617	·2	74·8%*
June	8,015	2·67	·915	·22	65·7%*
July	Changing over from A.U. to Butters.				
August	9,300	2·50	·134	·10	94·6%†
September	8,590	2·77	·147	·11	94·7%†

Notes.—The increase in slime costs in August and September over June and July is not due altogether to the filter and will be considerably reduced during the next few months. Assaying is an extraordinary item, due to a large number of experimental assays made on Brown and Butters pulp and solutions in the progress of trials to determine proper duration of agitation, time required for washing in filter, etc. Zinc costs were considerably increased due to putting more zinc boxes into commission, and the cyanide consumption was increased, due to starting up Brown vats and experimenting with varying strengths of solution. On the present small slime tonnage (about 9,000 per month) the average total cost per ton of slimes treated is not likely to exceed 1s. 8d. No white labour is included in Butters costs, because, owing to the plant having excess capacity, no additional white labour is required.

New Modderfontein G.M. Co.

Total cost of slime treatment (including Butters filter).

	Tons Slime Treated.	Cost per Ton of Slime.
August	30,053	1/1·437d.
September	27,830	1/1·113d.

Separate cost of operating Butters filter.

	August.			September.		
White wages ..	£64	7	0	£77	19	6
Coloured labour ...	51	7	6	48	3	11
Stores and materials	140	1	3	68	15	10
Workshops ..	79	19	2	92	15	2
Power ...	146	15	4	137	9	5
Air power ...	25	0	0	25	0	0
	£507	10	8	£450	3	10
Cost per ton slime treated	4·053d.			3·882d.		

Extraction Results.

1911.	Tons Slime Treated.	Original Value. Dwt.	Residue Total Dwt.	Residue Washed. Dwt.	Extraction.
May ...	19,714	1·538	·265	·082	82·7%‡
June ...	23,658	1·628	·234	·091	85·6%‡
July ...	Changing over to Butters filter.				
August ...	30,053	1·744	·086	·077	95·0%§
September ...	37,830	1·807	·093	·082	94·8%§

Notes.—The sums charged to stores and materials and to workshops are unduly high, owing chiefly to repair on leaves. The system of dry discharge of filter cakes, peculiar to this plan, and not repeated in later installations, has been found to require a much stronger leaf than the one supplied; and until the whole set of leaves has been properly reinforced, the above items are likely to remain high. Under normal conditions, on the present slime tonnage of about 30,000 per month, the Butters cost should be about 3d. per ton.

Crown Reef Plant (of Crown Mines, Ltd.)

Total cost of slime treatment (including Butters filter and modified Brown agitators.)

	Tons Slime Treated.	Cost per Ton of Slime.
July	19,269	1/1·4d.
August	19,942	1/1·2d.
September	18,323	1/1·9d.

Separate cost of operating Butters filter.

	July.			August.			September.		
White labour	£86	7	6	£77	12	6	£81	0	0
Coloured labour	9	6	2	9	13	3	12	18	0
Coloured labour sundries ...	5	11	0	5	6	7	5	4	3
Sundry stores	91	11	1	56	13	9	97	5	8
Workshops ...	6	9	8	9	13	11	27	8	7
Total power ...	89	0	0	101	0	0	87	0	0
	£288	5	5	£260	0	0	£310	16	6
Cost per ton slime treated	3·59d.			3·13d.			4·07d.		

Extraction Results.

1911.	Tons Slime Treated.	Original Value. Dwt.	Residue. Total Dwt.	Residue. Washed. Dwt.	Extraction.
July ..	19,269	2·038	·20*	·132	90·1%
August ...	19,942	2·031	·20*	·159	90·1%
September ...	18,323	2·023	·18*	·151	91·1%

Notes.—This plant has been handicapped from the start by the inclusion of accumulated slime, very unsuitable for treatment, owing to presence of organic matter, etc., and to large percentage of coarse sand, improperly separated. This has rendered normal operation of stock tanks impossible, and has seriously affected the valves—so that repairs and renewals have been continuous and very troublesome. With the exhaustion of the supply of old slime, the complete renewal of defective valves (now under way) and the enlargement of the plant to adapt it to the tonnage from the new Crown Reef mill, a very much better record of costs and extraction may be expected.

Summary.—It will be seen that while all these filter plants have proved profitable to the group wherever adopted, they are at present not doing the best work that may be expected of them. At the end of another six months a considerable reduction of working costs may confidently be looked for, as well as improved filtering efficiency. Interesting figures are forthcoming from the Robinson and Modder B. filters, which are not handicapped by any unusual conditions, and where, in design, and construction, advantage has been taken of experience gained in the earlier plants."—*S.A. Mining Journal*, Jan. 6, 1912, p 635. (A. R.)

PREMATURE PRECIPITATION OF GOLD IN THE CYANIDE PROCESS.—"The author considers that losses of gold stated to be due to premature precipitation ore often due to other causes, *e.g.* de-aeration of the cyanide solution. Experiments on an ore containing graphite showed that the latter does not precipitate gold from a cyanide solution. The action of charcoal as a precipitant is considered to be due to occluded gases perhaps aided by osmotic pressure. Some experiments on precipitated gold showed that even after removal of zinc, the black amorphous gold is much less easily soluble in cyanide solution than yellow gold."—R. S. BOWNE.—Transactions American Metallurgical Society, *through Mining and Engineering World*, Dec. 2. 1911, 1108-1111.—*Journal Society of Chemical Industry*, Vol. 31, p. 29. (J.G.)

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* Adair-Usher system.
† Butters and Brown agitators.
‡ Decantation.
§ With Butters filter.

* Before the installation of the Butters filter the total slime residue at this plant ranged between ·35 and ·45 dwt.

Zinc Dust.—"A new ruling of the Interstate Commerce Commission (U.S.A.) lists zinc dust and zinc flue dust as 'inflammable solids' and directs that they must bear a yellow shipping label, as prescribed in paragraph 1834, which reads: 'Packages must be tight and strong, and the interior packages must be so cushioned and secured that no rupture of either package can result from the ordinary shocks incident to transportation.' Zinc dust is commonly shipped in wooden casks containing 1,500 lb. It has been for a long time recognised as a dangerously inflammable substance and there has been a strong suspicion that it is also explosive, although the causes of explosions in which it has figured have not, perhaps, been conclusively traced to the zinc dust." —*Engineering and Mining Journal*, 1911, *92*, 1173.—*Journal Society of Chemical Industry*, Vol. 31. p. 30. (J. G.)

MINING.

Single Shift on the Rand.—"The following are a few points for and against the single-shift system, as they have occurred to me, on reading articles on that subject previously published in your magazine. The conditions existing at the moment are:

(1) Shortage of labour.
(2) New mines being developed.
(3) Mines in full going order.
(4) Old mines being worked out.
(5) Introduction of mechanical ventilation.
(6) The adoption of colliery methods, such as mechanical haulage underground.

Let us take the effect of the conditions mentioned under the foregoing headings and see how they affect the single or double-shift system.

(1) *Shortage of Labour.*—The output of the mines must be maintained, in other words, the same profit at least must be made whether labour be deficient or not. This may be taken as an axiom. Let it be taken for granted that with narrow lodes and consequent narrow stoping-widths, hand labour is the cheapest and best method of breaking ground, and let us assume that the average tonnage broken per boy per shift is 1 ton, as a basis on which to compare the various types of machine drills. The labour force on the mine is continually decreasing, but the tonnage must be maintained and the grade also. The obvious thing is to use more machine-drills, which may be of three types:

(*a*) Large reciprocating machines (about) $3\frac{1}{4}$ in. diam. cylinder.
(*b*) Small reciprocating machines (about) $2\frac{1}{4}$ in. diam. cylinder.
(*c*) Hammer-drills.

Machines of the first two types require two boys each: the hammer-drill requires only one boy. The average number of tons broken per machine per shift, for the various types of machines are:

(*a*) Large reciprocating machines 11 tons.
(*b*) Small reciprocating machines 5 tons.
(*c*) Hammer-drills 4 tons.

Compared on a tonnage basis, not taking up-keep of machines into account, the relative advantages of the different types of machines, from a labour-saving point of view, over hammer boys are:

(1) Large reciprocating machines $5\frac{1}{2}$ tons per boy per shift.
(2) Hammer-drills 4·0 tons per boy per shift.
(3) Small reciprocating machines $2\frac{1}{2}$ tons per boy per shift.

To overcome labour shortage the obvious way is to run more machine-drills. A case can thus easily be imagined, by which, in changing from hammer-boys to machines, more ground could be broken on single-shift with machines than could be broken on double-shift with hammer-boys.

(2) *New Mines being Developed.*—Again we face an axiom, namely, development must be rushed as fast as possible, at any rate for some time. Under this category shaft-sinking is included. To push development as fast as possible and work on single-shift is out of the question; either two or three shifts are worked in 24 hours, the limiting factors being the air-compressor capacity and the difficulty in clearing away broken ore from the development faces.

(3) *Mines in Full Going Order.*—These are at their maximum efficiency. At this stage a mine lends itself best to the adoption of the single-shift system; if the output can be maintained by this system, so much the better, as undoubtedly increased efficiency and safety will result. If the capacity of shaft-bins, sorting-floors and hoisting-plant allow, it may be possible to handle the required tonnage entirely in one shift, otherwise although sufficient ore can be broken in one shift, yet shovelling, tramming, and hoisting will have to be done on double-shift. In addition, development and stope faces must be cleared on night-shift ready for the next day's work. Thus in a mine that has reached a stage when conditions seem most favourable to the adoption of a single-shift system of working, in most cases only the actual breaking of the ground will be done solely on day-shift.

(4) *Old Mines Becoming Worked Out.*—As the ground becomes worked out the number of stope-faces is continually decreasing. To maintain the tonnage the stope-faces are changed from single to double-shift, and tramming and shovelling are put on double-shift also.

(5) *The Introduction of Mechanical Ventilation.*—By this means smoke and noxious fumes will be quickly expelled from the mines, which will favourably affect the introduction of double or treble shifts per 24 hours, as the second or third shift will be able to go to the working face almost immediately after blasting has taken place.

(6) *The Adoption of Colliery Methods* underground will affect an economy in labour and with proper organisation will facilitate the handling of broken ore, making it possible to do all the necessary tramming and shovelling on the day-shift, while repairs, landing of stores, jumpers, and timber will still be done on the night-shift.

In conclusion, by working a single-shift entirely, increased safety and efficiency underground is the result and, in addition, economy and time-saving for repairs, both in respect to the surface plant, such as air-compressors, and also in respect of the underground plant. Generally speaking the single-shift system of working is advantageous, but it is often practically impossible to work a mine entirely on single-shift, by reason of many of the causes mentioned. Mining conditions vary continually and considerably on every mine throughout the life of the mine. Thus although it may be possible to work a mine at one stage of its life to the best advantage on single-shift, yet at a later stage it may be quite impossible to maintain the profits without at least some modification of the single-shift system."—G. Hildick Smith—*Mining Magazine*, Jan. 1912, p. 55. (A. R.)

Small Stope Drills.—"Of recent years the breaking of ore by rock drills has been carried out far more extensively than in the past, when smaller milling equipments were at work and a lesser tonnage had in consequence to be sent to surface.

Shortage of native labour has further increased the necessity for supplanting hammer boys by machine drills wherever practicable. One of the great endeavours of the mining industry of recent years has been to obtain an efficient and economical stoping drill. The history of the development of the small machine on the Witwatersrand has been recorded at length in the pages of this journal. It would be folly to pretend that perfection or finality has been secured, but there can be no denial of the fact that a very great deal more of the economics of machine stoping is known to-day than was contained in the philosophies of Main Reef mine managers a year or two ago. Moreover, machine drills are now accounting for an increasing percentage of the ore milled, and the policy of breaking ore in stopes by machines is not now attended with such serious disadvantages in the breaking of abnormally large widths of reef as obtained a few years ago. In a recent issue we dealt with a new system of drilling in stopes which was being practised at the Jupiter mine, where the results secured were so satisfactory that it was queried with some justification, 'Is a solution of the labour shortage on the mines in sight?' We learn that the Ingersoll 'Butterfly' air feed hammer drill continues to do good work on that property, and the method of stoping adopted there is said to be showing such good results that many mining men entertain a high opinion of the labour-saving possibilities of the method.

Experiments at the Geldenhuis Deep.—Following on the interesting work that Mr. Henry is conducting at the Jupiter with these hammer drills putting in down holes with hollow steel, we understand that the Geldenhuis Deep have been successful in drilling down holes to a depth of 57 inches with the hammer drill, using solid steel. The trials are in the experimental stage, and fuller particulars will be given later. If the trials prove successful, the disadvantage of using hollow steel will be overcome. As at the Jupiter, Ingersoll 'Butterfly' drills are being utilised for this work. So satisfactory have been the results obtained from this drill, using the Henry and Hardach bars, that we understand that one mine on the Central Witwatersrand has about forty of these machines at work. The further progress of hammer drill stoping at the Jupiter and Geldenhuis Deep will be watched with keen interest. We would here also direct attention to the very promising work which is being achieved at the Knights Deep by Flottmann drills."—*S. A. Mining Journal*, Feb. 3, 1910, p. 784. (A. R.)

The Rate of Burning of Fuse.—"Either retardation or acceleration in the rate of burning of fuse can readily become the cause of accident, and accordingly the study of the conditions under which the rate of burning of fuse is so changed as to cause the fuse to burn more rapidly or more slowly becomes of particular importance.

Miners have frequently noticed that fuse, when surrounded by very dense stemming material, as in a hole that has been firmly tamped, burns faster than it should. In this case the dense and impermeable stemming prevents the free escape of the gases from the powder, and so compresses them in their attempt to escape as to bring about a decided quickening in the rate of burning. As the use of firm stemming is nearly always desirable, one can readily see that information as to the degree of quickening in the rate of burning of fuse under these conditions is essential in order that the miner may in such cases know what additional length of fuse is necessary to provide for the increased rate of burning.

Retardation in the rate of burning of fuse is a serious fault. Cases are known, though fortunately very few, in which a piece of fuse has seemingly stopped burning, and yet after an interval of one or more hours has brought about the explosion of the charge to which it was attached. Cases of this sort never happen with fuse that is in good condition, but may sometimes happen when fuse is of poor manufacture or has been subjected to unfavourable conditions of moisture or temperature.

The conditions that are believed to be most active in bringing about either a retardation or an acceleration in the rate of burning of time fuse have been investigated experimentally, and the results that have been obtained are classified under effects due to: (a) Pressure; (b) temperature; (c) moisture; (d) mechanical injury.

Under ordinary conditions nearly all types of fuse show great uniformity in their rate of burning. Practically all types of fuse examined in connection with the preparation of this report had a total variation in their rate of burning under normal conditions of less than 20% and all would have been passed under the allowance of 'no variation greater than 10% above or 10% below the average rate of burning.'

Under the influence of pressure practically all types of fuse are subject to wide variation in their rate of burning. Such pressure as can readily be produced by the confinement of the gases evolved by the burning fuse itself is sufficient to increase the normal rate of burning from 92⅔ seconds per meter (28·2 seconds per foot) to 21 seconds per meter (6·4 seconds per foot). Thus, even confinement will cause fuse to burn from three to four times as rapidly as its normal rate. In experiments made with fuse confined by stemming of various kinds, wide variations in the rate of burning were noted, and whenever lengths of fuse are confined by stemming or other materials impervious to gas, a sufficient length of the fuse should be used to allow for the increased rate of burning due to the pressure produced by the evolved gases.

High temperature causes a marked retardation in the rate of burning of fuse, and storage for even a short period of time near boilers, or wherever the temperature may be high, is sufficient either to cause 'misfires' or to retard the rate of burning of the fuse so much as to greatly increase the liability to 'hold-backs,' delayed shots, etc. It is probable that many of the difficulties that are sometimes encountered in regard to fuse burning too slowly and causing delayed shots are due, in part at least, to such fuse having been kept in too warm a place. Fuse that is not intended for use in wet places (cotton fuse, etc.), does not suffer marked change in its normal rate of burning by reason of the effect of high temperatures, whereas the more completely waterproofed types of fuse show increasingly great effects from heat. Even exposure to comparatively low temperatures for considerable lengths of time causes marked retardation in the rate of burning of such fuse, and exposure to a fairly high temperature for even a short length of time may cause certain types of fuse to burn from three to five times as slowly as their usual rate. To insure the best results, fuse should always be protected from extremes of temperature.

Climatic conditions affect to a considerable extent the rate of burning of the less waterproof types of fuse. Damp fuse burns more slowly than normal fuse, and fuse that has been wet and then thoroughly dried tends to burn at a rather slow rate, and may even cause delayed shots by smouldering for a considerable time. Fuse containing several wrappings of tape saturated with tar or asphalt resists moisture to a considerable extent, and may be used for firing shots under water, provided the fuse is not allowed to remain too long a time in contact with water before the shot is fired.

It is, of course, evident that when these waterproofed types of fuse do become wet, whether through storage for a long time in a damp place or through exposure to water after the protecting layers of asphalt or gutta-percha are mechanically abraded or injured in any other way, they are more difficult to dry out than are other types of fuse, and are more liable to burn at a rate slower than the normal rate.

Fuse that has been subjected to actual mechanical injury, particularly to hammering or pounding, or the blows of falling rock, etc., has a greatly increased rate of burning, and sometimes burns so rapidly as to be almost instantaneous in its action. The mere bending, coiling and twisting of fuse, such as would be brought about by forcibly placing within a bore hole a length of fuse considerably greater than the depth of the bore hole, does not produce any marked change in the rate of burning, but pounding or direct abrasion of fuse greatly increases that rate. Fuse that has been injured by severe abrasion or by too great pressure from any cause should not be used in any work where adjustment of the rate of burning is desired.

As a final summary it may be stated that ordinary fuse may under some conditions burn as fast as three seconds per metre (one second per foot), and under other conditions it may burn as slowly as 745 seconds per metre (227 seconds per foot). The former rate is more than 200 times as fast as the latter, and each is widely removed from the normal rate of burning of similar brands of fuse. Hence, the condition and past history of any roll of fuse is an important matter, and in mining and blasting operations the safety of the miner demands that only fuse that has been carefully stored and kept from unfavourable conditions shall be used."—W. O. SNELLING AND W. C. COPE, *Mining Science*, Feb. 15, 1912, p. 169. (J. M.)

DECREASE OF VALUE IN ORE SHOOTS WITH DEPTH.—"It is but proper to state at the outset that payable ore deposits do not usually persist in great depth, in fact in a geologic sense they are essentially superficial phenomena. It is true mineralized fissures are known to extend to depths of 6,000 ft. and probably much more, but it seems likely they will seldom be sufficiently rich to pay.

As a general fact, openings or fractures in the earth's crust decrease in size as depth is attained, and a fissure that may have a width of many feet in its upper part, if followed to sufficient depth is likely to end in a series of small cracks whose width is measured in inches rather than feet. Fissures thus defined may not be necessarily ore-bearing, or they may be too lean to pay in whole or part. A fissure is not necessarily a vein, but a vein assumes the pre-existence of a crack or fissure.

In this dissertation, only such veins are contemplated as have had their mineral contents deposited by filling or replacement along fissures through which the metal-bearing waters and vapours have supposedly circulated. Bedded deposits, such as the Lake Superior copper and the South African gold-bearing conglomerates are in a class by themselves, one not contemplated in this discussion, although as is well known the workings on them are about the deepest in the world.

It is essential to bear in mind that many veins as we now find them, may be mere roots of what they once were, erosion having removed hundreds or even thousands of feet from their original upper portions. According to Lindgren, in some of the Australian deposits, such as those at Bendigo, although carrying payable ore as deep as 4,600 ft., the gold quartz was probably deposited therein when they were at least 7,000 ft. below the original surface. Similarly he believes the gold ores of the Mother Lode, California, to have been formed at depths not less than 6,000 ft. from the surface. In the gold deposits of the southern Appalachians we have perhaps as good examples of vein roots as are to be found. The amount of material eroded from their croppings since the time of formation Lindgren believes to be greatly in excess of those figures given for Australia and California. When we consider that although usually lean these gold ores have been found and followed in the southern Appalachians to depths of over 1,000 ft., their original outcrops must have had a vertical range exceeding 7,000 or 8,000 ft. According to Ransome, the original surface at Cripple Creek, Colo., was probably from 2,000 to 5,000 ft. above the present one. In the gold districts of Nova Scotia, Faribault has estimated that the erosion of some of the sharpest and highest folds has had the effect of truncating them to a depth, as far as known, of over 8 miles. The presumption here is that this vast amount of erosion has removed what was probably the best part of these gold lodes, since the mere stumps that remain have never yet proved of much value. Basing his views on the extreme narrowness of the fissures in the Cobalt mining district, Emmons concludes that the veins there were formed under the pressure of an enormous weight of superincumbent rock at a great depth below the then existing surface; hence before present conditions were reached, the greater part of the original veins had been eroded away, leaving us to-day mere remnants of their original metal treasures. Whether we accept or not this unpromising and unwelcome view, it appears to be a fact that the change from rich to poor in the Cobalt veins is abrupt, and that this alteration takes place at depths not exceeding 300 ft.; in other words, the productive zone is seemingly shallow. Since these observations were made by Emmons a continuation of payable ore shoots from the Huronian into the Keewatin is reported, which, if correct, rather dispels the pessimistic view held by many that these deposits would not persist below the Huronian, that is below 300 or 400 ft. in vertical depth. In eastern North America it is certain that the type of veins found close to the surface, that is, those cutting volcanic rocks, recent flows, the upper surface of which at the time of vein formation can be determined with more or less accuracy, is exceedingly rare. On the other hand, older veins which may intersect the intrusive rocks like diorite and granite, rocks which we know have surely been consolidated at great depths, and which were probably once covered by heavy masses of just such flows as above indicated, are comparatively common. It is doubtless assuming too much to conclude that a district which has been profoundly eroded like some of those mentioned, is unlikely to contain veins payable and persistent in depth, it is impossible to know how deep the original fissures may have been; it is only possible to collect observations from a comparatively few widely scattered districts all over the world, where deep mining has been attempted. These records are ever open to suspicion, for we may never be sure that a vein supposedly bottomed in lean ore or no ore at all, has not simply temporarily pinched out, to be followed lower down by another shoot as good as any found higher up. Undoubtedly many, possibly most, veins in which such conditions have been encountered and finished as far as profitable working is concerned, are hopeless. The probabilities are, however, that in the majority of cases veins or lodes do decrease in richness below 1,000 to 1,500 ft. This cannot always be safely accepted as a rule, save it be one with many exceptions.

Suess has pointed out that the land surfaces of Africa are far older than either North America or Europe. We are reminded in this connection that Africa has as yet produced no great vein or lode deposits of the first rank, the gold "banket" of the Transvaal not belonging to the class under discussion. The lodes of Rhodesia have been on the whole disappointing, while none of the modern gold mines of Egypt have ever paid a dividend and most of the concessions taken up in 1900 and 1901 have been abandoned.

Instances of the discovery of new deep-seated ore shoots are not uncommon in well established metal-producing districts where systematic exploration and development are being carried on. A number of such cases might be cited, including the famous Comstock and some of the old mines on the Mother Lode in California. At Guanajuato, Mexico, the lower reaches of the Valenciana lode on the "Veta Madre" are to be tapped at a vertical depth of 2,000 ft. below the collar of the Nueva Luz shaft, and about 3,300 ft. on incline of vein from top of the old Valenciana shaft, by new workings, with a reasonable assurance of finding large bodies of payable ore.

In Australia, the Victoria Reef quartz mine at Bendigo has attained a vertical depth of over 4,600 ft., but as the ore body at this level is too lean to pay it has been decided to sink to a depth of 5,254 ft., the Government bearing half the cost. Between the 4,254 ft. and the 4,154 ft. levels considerable coarse gold was found, some pieces weighing a half ounce after passing through the stamps. The average of the ore was about $17 per ton. The gold itself has not decreased in purity nor was it in unusually fine particles as had been expected. This at present is the deepest gold mine in the world, and is situated centrally on the famous New Chum lode from which many millions of dollars have been obtained.

H. Kilburn Scott recently had occasion to visit the Morro Velho mine in Brazil, belonging to the St. John del Rey Gold Mining Co., and found it almost 5,000 ft. in vertical depth; the ore has an approximate dip of 45°, so that the depth along the slope is about 7,000 ft.

According to H. C. Hoover, at Kalgoorlie, West Australia, there are several properties upon which ore bodies have been discovered in depth where none or but small ones existed at the surface. In other instances in this district a marked increase of value has been noted with depth. This paper of Hoover's was written eight years ago, whether this last characteristic has been maintained I do not know; the latest reliable information obtainable is to the effect that several of the deeper mines in the Kalgoorlie district are producing abundantly at the 2,600 to 2,800-ft. levels.

Lindgren says these deposits have no definite boundaries, either horizontally or vertically, other than those determined by the decrease of assay value of the rock in any one direction. It would be a mistake to draw any general conclusion from conditions at Kaalgoorlie. They are not unique, though perhaps unusual. There seems to be little reason to assume that depth below the surface has *per se* any direct relation to the extent and richness of the lodes. Recent experience at Tonopah, Nev., tends to show a decrease in value with depth with a compensating tendency to wide lateral extent of pay ore in the veins, limited only by the characteristic block faulting of the district. However, it is not to be inferred from this that a vein or lode or series of veins are necessarily continuous laterally through the full extent of such a block. Evidently such is not the case, for the veins appear to cease laterally as they do in depth even before being cut by a fault or intrusive dike. But the practical point in this instance is that the chances of finding ore are overwhelmingly greater in lateral exploration than in vertical. An interesting classical case may be cited in this connection. At Joachimsthal, Bohemia, the veins have been divided into two classes, first, those rich in the upper levels but which grow poorer with depth, and second, those fissures possessing fairly constant and persistent values as deep as 2,000 ft. A peculiarity of the latter class is that some of the fissures reach the surface, not as mineralised veins, but as narrow barren cracks. A number of years ago W. P. Blake observed that when a lode cuts different foundations we must expect its contents to change with the varying character of the rocks, either downward or horizontally. The change in Cornwall, England, from copper ore to tin ore in passing from "killas" into granite is a good and classic example of this. When lodes traverse stratified rocks parallel with bedding the conditions as respect the chemical constitution of the walls may be assumed to be alike for great distances, and therefore we may look for similar mineral contents as far as these conditions extend. But when the lode traverses a group of beds transversed to bedding, uniformity of mineralisation either in quantity or quality need not be expected.

Among the minor causes of enrichment of ore deposits near the surface is the mingling of solutions from cross fissures. Van Hise attributes the principal cause of ore shoots to them and it is indisputable that at such intersections ore bodies are so common. But it does not appear that bonanzas or local enrichments in deep lodes are necessarily dependent upon such influences, in fact, deep-seated ore shoots are generally characterised by comparative dryness."—F. LYNWOOD GARRISON *Mining and Engineering World*, Feb. 10, 1912, p. 346. (A. R.)

MISCELLANEOUS.

VENTILATION AND THE INFLUENCE OF OZONE ON AIR.—"The good effects which result from efficient ventilation and open-air treatment are generally supposed to be due to the chemical purity of the air. They are due really to the movement, coolness, relative humidity of the air, and to the ceaseless variation of these qualities. The ventilating engineer has hitherto followed a great illusion in thinking that the main object to be attained is chemical purity of the air. The heating engineer has sought after an equally great illusion in striving to give us a uniform summer temperature. The ventilating and heating engineers primarily should aim at giving us air which is cool, of proper relative humidity, and which moves so as to vary the cutaneous state of the body. Our comfort and discomfort in crowded rooms and shut-up places depends, not on the chemical purity of the air, but, to a minor degree, on the influence of the smell of the air on the olfactory sense, and, to a vast degree, on the influence of the temperature, relative humidity, and the variations of these qualities of the air, which act on the great field of cutaneous sensibility.

It is needless to point out that our sense of well-being depends to a very great degree on the comfortable condition of our skin, and yet the ventilating and heating engineers have paid little attention to this. While asserting that the chemical purity is of no account, we make the proviso that the air is only altered by the presence of human beings, and is neither rendered poisonous by the escape of coal gas, or other noxious trade product, nor deoxygenated by the oxidative processes of the soil, as it is in mines.

The chemical purity of the air can be considered from three points of view, the concentration of CO_2, the concentration of O_2, the supposed presence of organic poison exhaled in the breath.

Owing to the fact that a percentage of CO_2 is not legally permissible in factories, which exceeds a very few parts per thousand, it is commonly supposed that any greater excess of CO_2 acts as a poison. The truth of the matter is quite otherwise; for, whatever the percentage of CO_2 in the atmosphere may be, that in the pulmonary air is kept constant at about 5% of an atmosphere by the action of the respiratory centre. It is the concentration of CO_2 which rules the respiratory centre, and to such purpose as to keep the concentration both in the lungs and in the blood uniformly about the same. This fact, which was first firmly established by Dr. John Haldane, is of fundamental importance. It is impossible that any excess of CO_2 should enter into our bodies when we breathe the air of the worst-ventilated room, in which the percentage of CO_2 assuredly does not rise above 0·5%, or at the outside 1%. The only result from breathing such an excess of CO_2 is a slight and unnoticeable increase in the ventilation of the lungs. The increased ventilation is exactly adjusted so as to keep the concentration of CO_2 in the lungs at the normal 5% of an atmosphere. The very same thing happens when we take gentle exercise, and produce more CO_2 in our bodies; the pulmonary ventilation is then slightly increased, and thus the CO_2 concentration in the blood and lungs is kept at the same uniform level. At each breath we rebreathe into our lungs the air in the nose and large air tubes (the dead space air), and about one-third of the air which is inhaled into the lungs is "dead space" air. Thus, no man breathes in pure outside air into his lungs, but air contaminated perhaps by one-third, or (on deep breathing) by one-tenth with expired air. When a child goes to sleep with its head partly buried under the bed-clothes, and in a cradle with the air confined by curtains, he rebreathes the expired air to a still greater extent, and so with all animals that snuggle together for warmth's sake. Not only the new-born babe sleeping against its mother's breast, but pigs in a stye, young rabbits, rats and mice clustered together in their nests, young chicks under the brooding hen, all alike breathe a far higher percentage than that allowed in our factories by the officials of the Home Office. To rebreathe one's own breath is a natural and inevitable performance, and to breathe some of the air exhaled by another is the common lot of men who, like animals, have to crowd together and husband their heat in fighting the inclemency of the temperate and arctic zones. By a series of observations made on rats confined in cages with small ill-ventilated sleeping chambers, we have found that the temperature and humidity of the air—not the carbonic acid and oxygen concentration of the air—determines whether the animals stay inside the sleeping-room or come outside. When the air is cold, they like to stay inside even when the carbonic acid rises to 4% or 5% of an atmosphere. When the sleeping chamber is made too hot and moist they come outside.

In breweries the men who tend the fermentation vats work for long hours in concentrations of CO_2 of 0·5 to 1·5% (Lehmann). Such men are no less healthy and long-lived than those engaged in other processes of the brewing trade. In the Albion Brewery we analysed on three different days the air of the room where the carbonic acid gas, generated in the vats, is compressed and bottled as liquid carbonic acid. We found 0·4% to over 1% of carbonic acid in the atmosphere of that room. The men engaged therein worked twelve-hour shifts, having their meals in the room. They had followed this employment for eighteen years, and without detriment to their health. It is only when we come to the higher concentrations of CO_2, such as 3% to 4% of an atmosphere, that the respiration is increased so that it is noticeable to the individual himself, and such percentages, of course, diminish the power to do work; for the excess of CO_2 produced by the work adds its effect to the excess in the air, and the limit of panting is soon reached. Thus the power to work is checked. Divers who work in diving dresses, and men who work in compressed-air caissons, constantly work in concentrations of CO_2 higher than 1% of an atmosphere, and so long as the CO_2 is kept below 2% to 3% of an atmosphere, they are capable of carrying out efficient work.

It results, then, from what we have said, that concentrations of CO_2, such as occur in most crowded and worst-ventilated rooms, are of no account. Forced to admit this fact, the hygienist has fallen back on the hypothesis that organic chemical poisons are exhaled in the breath, and that the percentage of CO_2 is a valuable guide as to the concentration of these. It is necessary, he says, to keep the CO_2 below 0·1 per thousand, so that the organic poisons may not collect to a harmful extent.

Before we turn to the discussion concerning the supposed existence of these organic chemical poisons, we will deal with the question of oxygen. The oxygen in the worst-ventilated schoolroom, chapel or theatre, is never lessened by more than 1% of an atmosphere. The ventilation through chink and cranny, chimney, door and window, and the porous brick wall, suffices to prevent a greater diminution of the oxygen concentration. Now, in all the noted health resorts of the Swiss mountains, such as St. Moritz, the concentration of oxygen is lessened considerably more than this. On the high plateaux of the Andes there are great cities: Potosi, with 100,000 inhabitants, is at 4,165 meters (barometric pressure about 450 mm. Hg.); railways and mines have been built even at altitudes of 14,000 to 15,000 ft. Owing to the nature of the chemical combination of oxygen with hæmoglobin, man can adjust himself to very great variations in oxygen concentration. At Potosi girls dance half the night, and toreadors display their skill in the bull-ring. We have watched our students, shut in an air-tight chamber, until they came to breathe air containing only about 16% of oxygen and 3½% of carbonic acid, and seen their puzzled look when they found they were unable to light a match and smoke a cigarette. There was then too low a percentage of oxygen to support combustion, but of this they were quite unaware. All the evidence goes to show that it is only when oxygen is lowered below a pressure of 14% to 15% of an atmosphere, that signs of oxygen want arise. A diminution of 1% of an atmosphere has not the slightest effect on our health or comfort.

We must now discuss the evidence for the existence of organic chemical poison in the exhaled breath. The evil smell of crowded rooms is accepted by most as unequivocal evidence of the existence of such. This smell, however, is only sensed by, and excites disgust in, one who comes to it from the outside air. He who is inside and helps to make the 'fugg' is both wholly unaware of, and unaffected by it. Fluegge points out, with justice, that while we naturally avoid any smell that excites disgust, and puts us off our appetite, yet the offensive quality of the smell does not prove its poisonous nature. For the smell of the food or trade of one man may be horrible and loathsome to another not used to such. The sight of a slaughterer and the smell of dead meat may be loathly to the sensitive poet, but the slaughterer is none the less healthy. The clang and jar of an engineer's workshop may be unendurable to a highly-strung artist or author, but the artificers miss the stoppage of the noisy clatter. The stench of glue-works, fried-fish shops, soap and bone manure works, middens, sewers, become as nothing to those engaged in such, and the lives of the workers are in no wise shortened by the stench they endure. The nose ceases to respond to the uniformity of the impulse, and the stench clearly does not betoken in any of these cases the existence of a chemical organic poison. On descending into a sewer, after the first ten minutes, the nose ceases to smell the stench; the air therein is usually found to be far freer from bacteria than the air in a schoolroom or tenement (Haldane).

If we turn to foodstuffs, we recognise that the smell of alcohol and of Stilton or Camembert cheese is horrible to a child or dog, while the smell of putrid fish—the meal of the Siberian native—excites no less disgust in an epicure, who welcomes the cheese. Among the hardiest and healthiest of men are the North Sea fishermen, who sleep in the cabins of trawlers reeking with fish and oil, and for the sake of warmth shut themselves up until the lamp may go out from want of oxygen. The stench of such surroundings may effectually put the sensitive untrained brain-worker off his appetite, but the robust health of the fisherman proves that this effect is nervous in origin, and not due to a chemical organic poison in the air. The supposed existence of organic chemical poison in the expired air is based upon experiments of Brown-Sequard and d'Arsenval. They injected into guinea-pigs and rabbits either the condensation water obtained from the breath, or water which they used several times over to wash out the trachea of dogs. The water was injected subcutaneously and in large amounts, and produced in their hands signs of illness, collapse and death.

These experiments have been repeated by many others, and with negative results by those whose methods of work demand most respect—Dastre and Loye, Van Hofmann Wellenhof, Lehmann and Jessen, Haldane and Smith, Weir Mitchell and Borgey, etc. A few confirmatory results have been obtained by methods of experiment which are truly absurd in their conception. One to two cubic centimetres of condensation water (obtained by breathing through a cooled flask) have been injected into a mouse weighing 13 gm. or so. This is equivalent to injecting five litres of water into a man weighing 65 kilos. Who would not be made ill by the injection of about nine pints of cold water beneath his skin? It has been shown that injections of pure water alone in doses of over one cubic centimetre may make a mouse ill (Inaba). Such experiments are ridiculous, and deserve not a moment's attention.

In the case of Brown-Sequard and d'Arsonval's experiments they injected from 4 to 44 cubic centimetres into guinea-pigs and rabbits, either directly in the circulation or subcutaneously. Using the washings of a dog's trachea, or the condensation fluid obtained from the breath, they could not fail to inject traces of the proteins of the saliva. A second injection of such into the same animal might produce the well-known anaphylactic shock.

The question before us is—Do men breathe out a substance poisonous to man? If there were anything in the claims of the American authors, we should expect to find rats, which dwell in the same confined cage and breathe each other's breath, sensitive to the injection of a trace of each other's protein. We are informed, by those who study the phenomena of anaphylaxis, that no such sensitivity can be shown.

After studying the literature on this subject we are convinced that there is no positive evidence which demonstrates the poisonous nature of the condensation water obtained from the breath. We go further and say there is at present no trustworthy evidence of the existence of any such poison in the exhaled air. Brown-Sequard sought to substantiate his views by carrying out a different series of experiments.

These experiments have been repeated with the greatest care by Haldane and Lorrain Smith and others. It has been proved conclusively that no harm results so long as a sufficient air current is maintained to keep the carbonic acid below a poisonous amount. The animal in the last cage dies when the CO_2 reaches 10% to 12%. If the CO_2 is kept down the animal in the last cage puts on weight and thrives as well as the animal in the first cage. Of course, it is necessary in such experiments to clean the chambers daily, and supply the animals with suitable food and bedding. Brown-Sequard's result must be ascribed to suffocation arising from failure in experimental method. We accidentally lost four of our animals from suffocation after the experiment had continued for a month, owing to the chinking of a tube and consequent interruption of the air current. Benedict has shown that a man can live many days in a closed chamber in comfort without damage to his health, having not the slightest cognisance of any defect in ventilation, when the ventilation is so reduced that the carbonic acid accumulates in the chamber up to 1%—that is to say, so long as the air in the chamber is kept cool and dry. We have enclosed eight students in a small chamber holding about three cubic metres of air and kept them therein until the CO_2 reached 3% to 4%, and the oxygen had fallen to 17% or 16%. The wet-bulb temperature rose meanwhile to about 85°F., the dry-bulb a degree or two higher. The discomfort was very great, but this was relieved to an astonishing extent by putting on electric fans placed in the roof, whirling the air in the chamber, and so cooling the bodies of the students.

In a crowded room the air confined between the bodies and clothes of the people is almost warmed up to body temperature and saturated with moisture, so that cooling of the body by radiation, convection by evaporation, becomes almost impossible. This leads to sweating, wetness and flushing of the skin, and a rise of skin temperature. The blood is sent to the skin and stagnates there instead of passing in ample volume through the brain and viscera. Hence arise the feelings of discomfort and fatigue. The fans in our chamber whirled away the blanket of stationary wet air round their bodies, and brought to the students the somewhat cooler and drier air in the rest of the chamber, and so relieved the heat stagnation from which they suffered. The relief became far greater when we allowed cold water to circulate through a radiator placed in the chamber, and so cooled the air of the chamber about 10°F.

Messrs. R. A. Rowlands and H. B. Walker have carried out, with us, numerous experiments in this chamber. They have performed measured amounts of work (raising a weight) under various conditions of temperature, moisture, and with varying percentages of O_2 and CO_2 in the chamber.

As a measure of the exhausting effect of the work, we have counted the frequency of the heart-beat in the first, second and third minute after the completion of the work. When the work is done without over-fatigue the frequency of the pulse, which is accelerated by work, quickly returns to normal. Our results show that increased percentages of CO_2, and diminished oxygen percentages of 2% to 3% and even 4% to 5% have little effect in modifying the frequency of the pulse, while the temperature and humidity of the air have a profound effect. The feelings of discomfort depend on the excessive heat and humidity, and are relieved by cooling and whirling the air in the chamber. If we suddenly raised the percentage of CO_2 in the chamber up to 2% we found the subjects inside were quite unaware of this. If we sat outside and breathed through a tube the air in the chamber we felt none of the discomfort which was being experienced by those shut up inside. Similarly, if one of those in the chamber breathed through a tube the pure air outside he was not relieved. Similar experiments were carried out by Paul, and with like results. The cause of the discomfort is thus proved to be due to the excessive heat and humidity, and absence of movement of the air. Studying the ventilation of sleeping-cars, T. B. Crowder finds that in these cars, called "close" or "stuffy," the temperature invariably is high. There has sometimes been an unpleasant odour. A high temperature renders this more noticeable. The most marked offensiveness noticed was in a day coach, where "the air was of such a degree of chemical purity as to indicate ideal ventilation by any standard that has ever been proposed. The car was hot and had many filthy people in it." Perfect comfort has been found associated with the highest chemical impurity in other cars. After healthy and clean students have been shut in our chamber we find their is no offensive smell in the confined air. Ventilation cannot get rid of the source of a smell, while it may easily distribute the evil smell through a house. As Pettenkofer says, if there is a dungheap in a room, it must be removed. It is no good trying to blow away the smell. Houses and people and their clothes and bodies must be made clean, and latrines and kitchens placed on the top of houses, or outside them, and on the windward side. There is yet another aspect of ventilation which we have not yet discussed, viz., bacterial infection. Catarrhal infections are spread by the expulsion of droplets of saliva when speaking, coughing or sneezing. During quiet respiration the exhaled breadth is practically sterile, for the wet mucous surfaces of the respiratory tract catch all the inhaled bacteria, and no "droplets" are exhaled. Can we lesson the "droplet" infection by ventilation? Fluegge concludes, from the results of his admirably contrived experiments, that we cannot. He says a current sufficient to drive out such droplets cannot be borne by the inhabitants of the room. A moderate ventilation current tends to keep the droplets suspended in the air. In a still room they soon fall to the ground, and, clinging to floor and furniture, may be wiped up next morning and removed by the housemaid.

We cannot hope to prevent infection in crowded railway carriages, theatres, chapels, schools, etc. The epidemics of common colds that sweep through the community show this only too well. It is impracticable to isolate all those suffering from colds, but we can teach them at least to cough and sneeze into a handkerchief, and perhaps even talk with a handkerchief held in front of their mouths. Above all, we must seek to keep our immunity and resist infection. The state of all these defensive mechanisms of the respiratory membranes are modified by the temperature and relative humidity of the air. Exposure to over-heated dry air dries up not only the skin but the membranes of the nose and throat, and so lessens immunity.

Exposure to over-heated moist air brings the blood into the skin, lessons the circulation through the viscera, and deprives us of the stimulating effect of cold on the cutaneous nerves, decreases the evaporation from the respiratory tract, diminishes the muscular activity, and so the amount of oxygen breathed in and food eaten, and thus altogether lowers the plane of our existence. Hence arise diminished health, strength, and increased susceptibility to catarrh. Those who habitually expose themselves to cold rarely take cold.

Confinement in overheated, windless air, which too often pervades places of business and amusement, is one of the chief causes of the depressed physical and mental vigour of town dwellers. The evidence of daily life proves that those who take hard exercise in the outside air for some hours a day escape these ill-effects. The health of sailors and country labourers shows us that sleeping in hot, confined and ill-ventilated quarters is of no consequence if the working day is spent in the open air. It is persistent exposure to the uniformity of warm, windless air which is the cause of the mischief. The old English methods of open fire and open window have very much to recommend them. By the open fire air is kept moving and cool air is brought in; the heating is by radiation, and uniformity of the conditions of temperature in the room is prevented. On the other hand, the impulsion of hot air into a room is the most objectionable of all the systems employed. A cool air and radiant heat are the ideal; the hot-air system gives us neither.

11. We are now in a position to examine the uses of ozone in ventilation. Ozone is represented by O_3, and differs from the oxygen molecule, O_2, by the addition of a third atom of oxygen, which is loosely bound and has powerful oxidising properties. It corrodes cork, rubber, and other organic substances, and oxidises, iron, copper, and even silver when moist, and dry mercury, and iodine. Galvanised iron tubes or glass can be used for the conductance of ozonised air. Traces of ozone are found in the atmosphere after thunderstorms, produced by the effect of the

electric discharge on atmospheric oxygen. It is said that traces of ozone exist in sea and mountain air. Ozone is generated by the silent discharge of high-tension currents from one plate of metal to another across a sheet of insulating material. In the Ozonair apparatus demonstrated by us the metal plates are sheets of fine gauze separated by micanite. The discharge takes place from the multitude of points on the gauze, and this equalizes the tension at which the discharge takes place, prevents sparking and the formation of oxides of nitrogen. These oxides contaminate the ozone which is formed when ordinary smooth plates are employed, for the discharge then takes place at a few rough points at a far higher tension and with sparking. It is said to be the sparking discharge in the air which causes the oxides to form. Each complete element of the Ozonair apparatus consists of a thin sheet of micanite, covered on each side by a square of gauze. The air can be made to pass by a series of baffle plates over a series of these elements. There are over 230,000 round projections to the square foot of the gauze, from which discharge takes place. The gauze is made of an aluminium alloy of forty meshes to the inch.

On exposing mammals to the different concentrations of ozone in our chamber, we have found they may be killed by prolonged breathing of concentrations of about twenty parts per million. The cause of death is an acute irritative inflammation of the respiratory tract. After death congestion of the lungs is found. The ozone in such concentrations causes the eyes to water and makes one cough. It is so irritative and unpleasant that no one would think of continuing to breathe it. Thus a man cannot be poisoned unawares by a dangerous concentration of ozone. He will certainly remove himself from its presence. No harm results from breathing such concentrations for a short period. A slight irritation of the respiratory tract and some sneezing and coughing, and perhaps headache, are the only penalties. The danger signals are clearly set by ozone, and anyone must naturally obey them. When ozone is employed in ventilation it is arranged in such a concentration as to produce none of these symptoms of irritation. Certain insects are comparatively immune to the effects of high concentrations of ozone. For example, we have found that fleas stand exposure to about one hundred parts per millon for five hours. This is very unfortunate. Had these parasitic insects only proved highly susceptible we should have found in ozone a clean and ready method of attack.

Ozone can be used in high concentrations for sterilising water; and herein is to be found one of its most valuable uses. The water is not only sterilised but oxygenated, and ozone can be used, therefore, with great advantage to keep the water of aquaria in good condition. Ozone has been successfully applied as the means of purifying the water-supply of several towns, both abroad and in this country. Where the water-supply is contaminated and the source of electric power is cheap, no better or simpler method can be employed. Ozone is a most powerful deodoriser. It takes away all disagreeable smells. Whether it destroys them or prevents the nose smelling them, is of little importance. The psychical effect is the same—the disgust due to the evil smell disappears. Ozone itself has a peculiar smell. It reminds one of the fumes of nitrous acid, and whether the smell is that of ozone or minute traces of oxides of nitrogen, it is difficult to say. The nose may detect what the analytical methods of the chemist fail to show. By the smell we can detect concentrations of far less than one part in a million, and the smell is the safest and easiest guide to a suitable concentration. Ozone should be present in the air for continuous breathing in concentrations not greater than that scarcely perceptible to smell. Concentrations of even one part in the million are too irritative. Very weak concentrations, barely perceptible to the smell, have no ill effects, but destroy the effect of unpleasant smells and give a certain tang or quality to stuffy air which relieves its monotony and uniformity.

It is in this respect that ozone has its use. There exist in modern conditions of life so many trade shops, tube railways, cold meat stores, etc., where the employees are exposed to a persistent uniform and depressing smell. The air in many buildings is made to smell by the heating appliance used. The addition of ozone takes away the smell and relieves the monotony of such air, and, as the Ozonair apparatus can, by the turning of a switch, be put in or out of use, the uniformity of the atmospheric conditions can thus be frequently changed. The ozone, no doubt, exerts its effect both on the cutaneous and respiratory nerves.

We have not been able to obtain any evidence that ozone in weak concentrations influences the respiratory metabolism of the resting man. We have studied the effect of ozone both on animals and on ourselves, estimating the amount of oxygen used up and carbonic acid produced during successive periods of time when air or ozonised air was breathed.

The rate of oxidation of the human body is set by the nervous system, which controls the activity of the body. Breathing pure oxygen in place of air has no effect on the metabolism of the man at rest. It will increase the metabolism of a man doing hard work, but only in the case of his working so hard that he suffers from a shortage of oxygen when breathing air. The body, unlike a fire, cannot be fanned up to burn faster. The living substance works at its own rate independent within wide limits of the changing conditions of the outside world. We should not expect, therefore, that ozone would stimulate the metabolism of the resting man. By altering the unpleasant uniformity of a close atmosphere it may help an employee to work more briskly. In high concentrations, ozone, just as high concentrations of oxygen, depresses the metabolism because it is toxic to the cells of the respiratory tract. The ozone which is breathed into the lungs is caught by the wet mucous membranes and none of it is exhaled. It is, no doubt, used up there in an oxidative process of the tissues. It is excess of such oxidative process which damages the cells lining the respiratory tract, when either too great a concentration of ozone (or oxygen) is breathed. There is no evidence that ozone reaches the blood, or that it has any influence on the body. The effect which ozone in weak concentrations has on the olfactory nerves, and those of the skin and respiratory tract, is the justification for its use in ventilation."—L. Hill, M.B., F.R.S. and M. Flack, M.A., M.B., B.Ch., *Journal of the Royal Society of Arts*, Feb. 9, 1912, pp. 344-351. (J. M.)

Economy in Power Transmission.—"The line-shaft of a system of machinery has been termed the highway for power, and the comparison is apt, for in many ways it bears an analogy to the roadbed of a railway. The ballast of a railroad may be compared to the balance of the rotating members upon a line-shaft; every pulley should be in perfect balance, and, so far as possible, the pull of belts in opposite directions be equalised. Not only as regards balance, but also in connection with support and alignment, may the line-shaft be compared to the railroad line. It must be firmly supported, and thus the hangers and bearings come into play, and its alignment must be correct and self-correcting, otherwise harmful resistances will multiply rapidly, and efficiency fall off accordingly. It should never be forgotten that the final efficiency of any system, such as that of a line-shaft and distribution, consists of the continued product of the several detailed efficiencies, so that any small falling off in the various elements will produce a much greater influence upon the whole. This means that the closest vigilance is necessary upon all details—watchfulness in the shop, in the erection, and in the continual operation. It does not mean, however, that the transmission should be a continual source of anxiety and trouble to the superintendent of motive power, for if the mechanism has been properly designed and erected it will take care of all these questions automatically. If the hangers are of the real ball-and-socket construction they will place the bearings themselves in parallel with the shaft at all times; if the adjustments may be made in all directions, the shafting can be placed in perfect alignment and kept there; while if the various portions are provided with independent friction-clutch connections, every opportunity is given for control over the whole system or any of its parts. Efficiency in manufacture means efficiency in operation, but there is yet a further kind of efficiency—that completeness which is always ready for any emergency. When it is understood that every detail of a power transmission system is kept in stock, it will be realised that there is an efficiency of the highest character, an efficiency which permits a minimum of loss in time in the case of accidents, changes, or extensions."—C. A. TROWBRIDGE, *Mining and Scientific Press*, Dec. 30, 1911, p. 837. (H. A. W.)

Reviews and New Books.

GRAPHICAL SOLUTION OF FAULT PROBLEMS. By C. F. TOLMAN, JR. $5\frac{1}{2} \times 8$, pp. 43, illustrated. $1. *Mining and Scientific press*, San Francisco California.

"Contents: Nomenclature of fault movement; methods of projection; elementary problems in line and plane intersections; fault movements of pure translation—fault plane vertical; fault movement of pure translation—fault plane inclined; fault problems where a rotation about a pole is combined with translatory movement."—*Engineering and Mining Journal*, Jan. 17, 1912. (A. R.)

Adams, Henry. The Mechanics of Building Construction. 8vo., pp. 252. *Longmans.* Net 6s.

Allsop, R. Owen. Engineering Work in Public Buildings: Power, Lighting, Heating, Ventilation, Water-supply. Illust. 4to., pp. 168. *Spon.* Net 12s. 6d.

Andrews, Ewart S. Elementary Principles of Reinforced Concrete Construction. Cr. 8vo., pp. 208. *Scott, Greenwood.* Net 3s. 6d.

Bottone, S. R. Electro-Motors: How Made and How Used. 6th Ed. Cr. 8vo., pp. 190. *Whittaker.* Net 2s. 6d.

Brown, Sir Hanbury. Irrigation: Its Principles and Practices as a Branch of Engineering. 2nd Ed. Revised. 8vo., pp. 318. *Constable.* Net 16s.

Browne, A. J. Jukes. The Student's Handbook of Stratigraphical Geology. 2nd Ed. 8vo., pp. 682. *E. Stanford.* Net 12s.

Brown, William Norman. The Principles and Practice of Dipping, Burnishing and Lacquering and Bronzing Brass Ware. 2nd Ed. Revised and Enlarged. Cr. 8vo., pp. 46. *Scott, Greenwood.* Net 3s.

Chapman, S. The Kinetic Theory of a Gas Constituted of Spherically Symmetrical Molecules. 4to., sd. *Dulau.* Net 2s. 6d.

Chatley, Herbert. Practical Gyrostatic Balancing. Cr. 8vo., pp. 74. *Technical Publishing Co.* Net 7s. 6d.

Chree, C. Studies in Terrestrial Magnetism. 8vo., pp. 218. *Macmillan.* Net 5s.

Cohen, Julius B., and Ruston, A. G. Smoke: A Study of Town Air. 8vo, pp. 96. *E. Arnold.* Net 5s.

Coleman, T. E. The Civil Engineers' Cost Book. 12mo., pp. 304. *Spon.* Net 5s.

Copperthwaite, W. C. Tunnel Shields and the Use of Compressed Air in Subaqueous Works. 2nd Ed. Revised. 4to. *Constable.* Net 31s. 6d.

Darling, Chas. R. Heat for Engineers. A Treatise on Heat. 2nd Ed. Revised. 8vo., pp. 475. *Spon.* Net 12s. 6d.

Dowson, J. Emerson, and Lartner, A. T. Producer Gas. 3rd Ed. 8vo., pp. 336. *Longmans.* Net 10s. 6d.

Faber, Oscar, and Bowie, P. G. Reinforced Concrete Design. 8vo., pp. 352. *E. Arnold.* Net 12s. 6d.

Garcia, A. J. R. V. Dictionary of Engineering Terms in English and Spanish. With Indexes in both languages. 16mo. *Hirschfield.* Net 4s.

Goldingham, A. H. The Gas Engine in Principle and Practice. 8vo. *Spon.* Net 6s. 6d.

Hasfield, W. H. Cast-Iron in the Light of Recent Research. With Frontispiece and 164 Illusts., including many Photomicrographs. 8vo. *C. Griffin.* Net 10s. 6d.

Hool, G. A. The Elements of Structures. 8vo. *Hill Publishing Co.* Net 7s. 6d.

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Kershaw, J. W. Elementary Internal Combustion Engines. Cr. 8vo., pp. 182. *Longmans.* Net 2s. 6d.

Lineham, Wilfrid J. A Text-Book of Mechanical Engineering. 11th Ed. Revised. 8vo., pp. 1,268. *Chapman & Hall.* Net 12s. 6d.

Martin, J. M. Practical Electro-Therapeutics and X-Ray Therapy. 8vo. *Keener.* Net 18s.

McMeen, S. G., and Millar, K. B. Telephony. A Comprehensive and Detailed Exposition of the Theory and Practice of the Telephone Art. Roy. 8vo., 6. *Lockwood.* Net 17s.

Moore, T. Handbook of Practical Smithing and Forging, Engineers' and General Smiths' Book. New Impression. Cr. 8vo. *Spon.* Net 5s.

Parker, G. W. The Elements of Hydrostatics with Numerous Examples. Cr. 8vo. *Longmans.* 3s.

Paterson, G. W. Lummis. Wiring Calculations for Electric Light and Power Installations. Cr. 8vo., pp. 108. *Scott, Greenwood.* Net 5s.

Potts, Harold E. The Chemistry of the Rubber Industry. 8vo., pp. 162. *Constable.* Net 5s.

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Swingle, C. F. Dynamo-Electric Machines and Central Station Equipment. Cr. 8vo. *American Book Supply.* Net 6s.

Taylor, F. Noel. Small Water Supplies. Cr. 8vo., pp. 182. *Batsford.* Net 6s.

Weinschenk, E. Petrographic Methods. 8vo. *Hill Publishing Co.* Net 15s.

Wilson, Harold A. The Electrical Properties of Flames and of Incandescent Solids. 8vo., pp. 126. *Hodder & S.* Net 6s.

Abstracts of Patent Applications.

(C.) 28/12. Willibald Grun (1), Pokorny and Wittekind Maschinenbau A. G. (2). Improvements in the method of and devices for regulating rotary compressors. 13.1.12.

This application relates to a method of regulating rotary compressors in such a manner that the number of revolutions of the compressor is governed by the pressure or quantity of the air.

With reference to the application, it appears that the device is only adaptable to rotary compressors, and that its action is quite different from the governing arrangements in use on ordinary compressors or turbine driven plants for generating power.

(C.) 60/12. Wilheim Hinselmann. Improvements in conveyors of the jigging type. 26.1.12.

The above specification refers to improvements in conveyors of the jigging type and deals particularly with certain improvements in carrying the wheels or rollers and in stiffening up and providing removable wearing parts where the different sections of the troughs connect together.

Reinforcing plates bent to the shape of the trough are fitted to the under-surface thereof where the latter is supported by wheels or rollers. It is claimed that by the use of these reinforcing plates the trough sections can be made of thinner metal than would otherwise be necessary.

The reinforcing plates are bent over on the outer edge so as to form guards for the wheels and are intended to prevent any person from getting his fingers trapped in the wheels. The reinforcing plates also form dishes on which the ends of the various sections of the trough rest.

There are six claims covering the use in a jigging conveyor of the reinforcing plates above mentioned; hangers under the reinforcing plates for keeping the axles of the wheels or rollers in position; extension of the reinforcing plates to form overhanging projections over the rollers; and wedge shape blocks up and down which the rollers run.

(C.) 75/12. W. Bullock. Improvements in stamd mills. 2.2.12.

This application relates to mortar boxes of stamp mills, and the claims are for the idea of separating the various stamp units operating in one mortar box by means of compartments, either permanent or removable.

(C.) 76/12. John Elford Porter (1), William Wallace Thomas (2). Load-carrier. 2.2.12.

This application has reference to the transport, raising and lowering of loads on a stationary cable or similar support for the trolley.

The device consists of a trolley with the usual fall block and terminal trolleys at each end of the line of intended transport, these trolleys being movable and clamped at determined points by means of a hanging rope. By means of a system of levers on the transporting trolley, it is held in position when engaging either of the terminal trolleys, the fall block is released for lowering and again engaged when the hoisting rope raises the block, and the trolley is then again automatically released for traversing to the other terminal. The description is for a means of transporting, raising and lowering a load by means of only one suspending cable and one haulage rope.

(P.) 94/11. William Joss (1), Charles Edward Horne (2). Improvements appertaining to the means for connecting rock drilling machines to their operating fluid supply pipes. 14.2.11.

This application relates to a connection on the air pipe at the point where the hose to serve a rock drill is fastened. This comprises a spud having its bore enlarged to serve as a box or casing for a valve arranged completely therein, in such a way that it is automatically opened by making a connection between the spud and the machine hose.

There is also a scheme which allows, by means of a small by-pass, a certain quantity of air to escape when the machine hose is not connected.

(P.) 102/11. William Cochran Boyd. Improvements in rollers for incline shafts and the like. 17.2.11.

This application relates to improvement in rollers in incline shafts and the like, and consists essentially in using renewable or interchangeable wearing rings for the outside surface of the roller.

(P.) 195/11. James Esson (1), Denis Frances Breslin (2). Improvement in respirators. 6.4.11.

This respirator consists of a series of rings having a mouthpiece of any shape, in which trap plates are arranged. The trap plates are smeared with either viscous, glutenous, oily, or other adhesive substance, and may be either grooved or roughed to retain more of this substance.

The object of this respirator is to provide free ingress and egress of air through a series of angles or curves formed by the trap plates, and the adhesive substance smeared thereon purifies the air from dust and such like matter, which may be cleaned off and the trap plates re-smeared when necessary.

(C.) 219/11. Harry August Stockman. An improved concrete reinforcement and apparatus for producing the same. 15.4.11.

This invention relates to a method of reinforcing concrete in which the reinforcement consists of a mat, bolster or skeleton wire structure comprising longitudinal and transverse tension and compression members. These members are arranged in two sets or series lying in different planes with inclined truss members between the series.

The inventor claims that in all the existing systems of reinforcement the wires have been laid flat or arranged in a common plane and that his arrangement of the wires in two planes constitutes a novelty. In addition he claims for certain specific arrangements of wires by which the above principle is applied in practice.

(P.) 313/11. John Davidson. Improvements in means for use in boring and re-boring the cylinders and valve chests of rock drills and the like. 3.6.11.

This relates to improvements in means for boring and re-boring cylinders and valve chests of rock drills and the like.

It consists of a boring bar fitted with cutters and a pair of bearings fixed at each end of the cylinder in such a manner as to support the boring bar whilst at work. Another claim of the invention is for making these bearings adjustable.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 144/12. William Cullen (1), Thomas Donaldson (2), The British S.A. Explosives Company, Limited (3). Improvements in gelatinous explosives. 8.3.12.

(P.) 146/12. Frederick Walter Bawden. Improvements in explosives. 11.3.12.

(C.) 147/12. Kenneth Lancaster Graham (1), Frederick Augustus Alexander (2). A device to be employed on mechanical haulages to prevent wear on truck wheels, rails, sleepers, and rope, and to throw the rope outside the rails on to rollers or other similar devices. 12.3.12

(P.) 148/12. James Grant Gibson. Improvements in tube mill liners and the like. 12.3.12.

(C.) 149/12. Allison Dalrymple Smith. Improvements in automatic couplings for railway, tramway, and other similar vehicles. 12.3.12.

(P.) 150/12. John Whidborne Watts. Improvements in treating complex or refractory ores, concentrates and the like. 13.3.12.

(P.) 151/12. George Daniel Hook (1), Alexander Purser (2). A new or improved apparatus for lighting fuse and the like. 14.3.12.

(P.) 152/12. John Thomas Kennaugh Maddrell. Improvements in apparatus for greasing ropes and rods. 15.3.12.

(C.) 153/12. David Hamilton Coles (1), Frederick Charavay (2). Improvements in internal combustion engines. 15.3.12.

(P.) 154/12. Friedrich Nellen. Improvements in or relating to mining props. 15.3.12.

(C.) 155/12. Daniel Bosqui. Process and apparatus for precipitating materials from solutions. 15.3.12.

(C.) 156/12. Robert Wishart. Improvements in apparatus for grading and levelling land. 15.3.12.

(P.) 157/12. Henry Smith. Heerigs Phonecap. 15.3.12.

(P.) 159/12. Edward George Clifford-Jones. Fluid for spraying mine workings. 16.3.12.

(P.) 160/12. John Frederick van der Velde. Improved cradle for rock drill-machines. 20.3.12.

(P.) 161/12. Charles William Dowsett. Improvements in regulating the outflow from pulp settling or classifying apparatus. 20.3.12.

(P.) 162/12. Tom Hurst. Improved electric lamp holder. 21.3.12.

(C.) 164/12. Herbert Bullock. Improvements in or connected with regulating or controlling devices applicable among other purposes for controlling dampers for furnaces. 22.3.12.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

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ALLEN, G., *to* 138, Yeo Street, Bellevue East, Johannesburg.
ATKIN, A. J. R., *l/o* Johannesburg; Ulundie G. M. Co., Eureka, Barberton.
BASKETT, E. G., *l/o* Brakpan; City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg.
BEAVER, W., *l/o* Roodepoort: P. O. Box 18, Sabie, Transvaal.
BELL, K. S., *l/o* Bulawayo; P. O. Box 4563, Johannesburg.
BIDLAKE, A. E., *l/o* Rhodesia; c/o H. Hay, Esq., West Coast Amalgamated Mines, Adjah Bippo, Tarquah, via Sekondi, Gold Coast Colony.
BLEWITT, E., *l/o* Selukwe; Box 379, Rhodesia.
BRODIGAN, C. B., *l/o* Brakpan; Old Hall, Halkyn, North Wales.
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HEALEY, J. E., *l/o* Johannesburg; P. O. Box 5, Maraisburg.
HERALD, A. J., *l/o* Germiston: 7, Glover Street, Arbroath, Scotland.
JONES, C. K. D., *to* Golden Kopje Proprietary Mines, Ltd., P. O. Golden Kopje, via Salisbury, Rhodesia.
KAY, H. G., *to* 80, Library Avenue, Premier Mine, Transvaal.
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SANER, C. B., *l/o* Pilgrims Rest; Glynns Lydenburg, Ltd., P. O. Sabie, Transvaal.
SHORT, A., *l/o* Barberton; Sunlight Mine, Kowyns Pass, P. O. Pilgrims Rest.
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THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

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Vol. XII. MAY, 1912. No. 11.

Proceedings
AT
Ordinary General Meeting, May 18th, 1912.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, May 18th, 1912, Mr. W. R. Dowling (Vice-President) in the chair. There were also present :—

37 Members : Messrs. H. A. White, E. J. Laschinger, A. F. Crosse, W Cullen, J. Gray, A. McA. Johnston, J. Littlejohn, A. Richardson, Prof. G. H. Stanley (Members of Council), H. A. Adams, H. W. Adler, S. Beaton, H. S. Candy, J. Cellier, J. Chilton, F. W. Cindel, P. L. Edwards, H. Lea, J. Lea, L. Marks, G. Melvill, M. T. Murray, S. Newton, W. J. R. North, J. M. Phillips, F. D. Phillips, J. F. Pyles, E. Roberts, L. J. Robinson, A. Rogers, C. O. Schmitt, G. H. Smith, R. Stokes, W. A. C. Tayler, A. Thomas, Prof. G. R. Thompson and O. Tonnesen.

28 Associates and Students : Messrs. M. S. Archibald, E. G. Baskett, W. E. Burrows, C. A. Damant, J. Davidson, I. Dicks, W. J. N. Dunnachie, O. A. Gerber, J. Gibson, H. S. Gieser, M. Green, T. D. Hall, J. M. L Henry, R. W. Irwin, V. V. de P. Lefrère, H. Lomberg, F. Millar, W. S. New, G. S. Paterson, F. J. Pooler, H. Rusden, P. Scatterty, H. Stadler, A. M. Thomas, J. Thorlund, W. Waters and J. L. White.

12 Visitors and Fred. Rowland (Secretary).

MINUTES.

The minutes of the April meeting as published in the April *Journal* were confirmed.

Messrs. A. McA. Johnston and E. J. Laschinger were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been elected as follows :—

NEW MEMBERS.

COLLINS, WILLIAM LEONARD, Transvaal Gold Mining Estates, Limited, Pilgrims Rest. Assistant Amalgamator.

CREESE, ALBERT FRANK, Transvaal Gold Mining Estates, Limited, Pilgrims Rest. Amalgamator.

HODGES, ISAAC, Bovey Court, Vereeniging. Mining Engineer.

ROLFE, F. P., New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston.

STUART, ALAN, Groenfontein Tin Mines, Potgietersrust. Mining Engineer.

GENERAL BUSINESS.

ANNUAL GENERAL MEETING.

The Secretary announced that the Annual General Meeting of the Society would take place on the 22nd June—the fourth Saturday in next month instead of the third Saturday as was usual. The nomination form for the election of Officers and Council was attached to the agenda paper, and all nominations should be lodged with him not later than the 23rd instant.

COMPETITION PAPERS.

The Secretary drew the attention of Student Members of the Society to the fact that the competition for the best paper submitted in Chemistry, Metallurgy or Mining would not be closed until the 31st May, and any papers received up to that date would be considered by the Committee and the awards would be notified at the next meeting.

STANDARDIZATION, ETC.

Mr. H. Stadler (*Associate*) asked whether the Council had yet arrived at a decision regarding his standardization scheme ?

The Secretary replied that he had written a letter to Mr. Stadler on Wednesday last on the subject.

The Chairman said the Council had come to the conclusion that they were not prepared to take any further action in the matter.

Mr. Stadler asked if he might know what led the Council to that conclusion and why his scheme had not been brought before the Society?

Mr. A. McArthur Johnston (*Past-President*) said the matter referred to by Mr. Stadler had, he thought, been dealt with by the Council and having been so dealt with it should go no further.

The Chairman said he must now declare this matter closed. It had been brought before the Society in a paper by Mr. Stadler in May 1910,[*] and he really did not think there was now anything more to be said or done regarding it.

ANNUAL DINNER.

Mr. Andrew F. Crosse said he would like to propose a very hearty vote of thanks to the Annual Dinner Committee. He believed there was some difficulty this year in getting the usual number to attend but, nevertheless, in the end they had a very successful dinner and everything went off smoothly. The vote of thanks was agreed to unanimously.

SCRUTINEERS FOR ANNUAL BALLOT.

The Chairman said it was necessary to appoint scrutineers for the annual ballot of officers for the new year. The following names had been suggested to him in this connection:—Messrs. S. Newton, F. D. Phillips, Abe Thomas, George Melvill and R. Lindsay. He asked if there were any other names members would like to suggest. It was necessary for them to appoint scrutineers that evening. The names as submitted above were agreed to.

The Chairman: I would ask those gentlemen to act when called upon.

NOTE ON DUST DETERMINATION BY FILTRATION THROUGH SUGAR.

By A. McArthur Johnston, M.A., M.I.M.M.
(Past President).

The collection of dust in air by drawing the latter through a layer of sugar is mentioned by Soper in "Air and Ventilation in Subways."

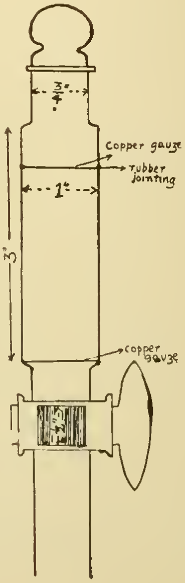
After a number of tests carried out by aspirating both surface air and underground air through sugar, and after comparing the results obtained with those found by using cotton wool, glass wool and water as the filtering media, we concluded that higher and more consistent results were obtained with the sugar, and for the last nine months we have been using this as the filtering medium with which to estimate the quantity of dust in mine air.

The filter bed is obtained by coarsely crushing pure cane lump sugar and retaining that failing to pass a 90 mesh (aperture 0·006 in.) sieve. This is charged into a glass stoppered separator of about 25 cc. capacity. The layer of sugar should be about 2 in. in depth and should be just slightly wetted to cause the dust to adhere to it, though not sufficiently wet to cause caking of the sugar. If found necessary, a small piece of brass or copper wire gauze is placed in the bottom of the separator to prevent the sugar escaping. (See sketch.)

The amount of air to be aspirated through the sugar is dependent on the quantity of dust in the air. Thus, surface air might, after a shower of rain, contain such a small quantity of dust that it would be necessary to pass about 500 litres, or even more, through the sugar to obtain a weighable quantity of dust, whereas one-tenth of this amount would, under present conditions, suffice for ascertaining the amount of dust in the air of most of the mines of the Rand. After taking the sample, the separator is stoppered and placed in a box to avoid contamination.

To estimate the dust, the sugar is dissolved in distilled water and then filtered through tared filter papers of the best quality or through a layer of asbestos in a Gooch crucible. The weight found represents the total dust present. By incinerating this, the amount of mineral dust may be determined, the difference representing the organic matter. It is usual to express the results in grams per cubic metre, though some chemists still adhere to grains per cubic foot.

(Grams per cubic metre × 0·437 = grains per cubic foot).



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[*] See this *Journal*, Vol. X., May 1910, pp. 382-391.

When it is desirable to estimate in mine air only the finest particles of dust, a sieve may be used to eliminate all those larger than 0·003 in. in diameter. The standard 200 mesh per linear inch sieve may be employed, but the precaution of washing this into the sample must be taken, for when the air is moist there is every probability that a portion of the finest dust may be retained by clinging to the wires. I show herewith a sketch of a separator which I am having made for these estimations. Through such an apparatus it should be possible to aspirate about 200 litres of air in half an hour.

I need scarcely mention that every precaution to avoid contamination by laying the glass apparatus on the ground or on ledges of rock should be taken, whilst of course when it is found that the sugar contains any dust, blank determinations should be performed and allowance made for this when reporting the result.

Mr. J. S. Cellier (Member): How do you damp the sugar to the right extent?

Mr. A. McA. Johnston: You have to guess it. The only thing is to strike some happy medium whereby the sugar does not cake.

Mr. H. A. White (*Vice-President*): I would just like to point out that Mr. Johnston does not mention washing the precipitated dust. I suppose that operation has to be performed?

Mr. A. McA. Johnston: Necessarily.

Mr. W. Cullen (*Past-President*): I would like to ask how much of the mine air you have to aspirate?

Mr. A. McA. Johnston: It depends entirely on what part of the mine is being sampled. In some places 50 litres are sufficient; in other places 100 or even 200 litre samples must be taken.

NEW APPARATUS FOR SAMPLING AIR FOR DUST.

By E. J. Laschinger, M.E., B.A.Sc. (Tor.), (Vice-President).

In the determination of dust held in suspension in air, in cases where the quantity of dust is not great and the particles very small, it is essential to draw a large sample of air through the filtering and collecting medium in order to obtain a reliable sample. To obtain an accurate result, it is, of course, also essential that the air sample should be measured accurately.

In order to obtain accurate results more quickly than by existing methods, the author at the request of the Ventilation Sub-committee of the Miners' Phthisis Prevention Committee, designed two methods of collecting samples. One apparatus was constructed locally for immediate use, and another of different construction is on order from oversea.

The sampling apparatus originally used in the committee's work was a plain aspirator tank of about 2 cu. ft. or 50 litres capacity, and owing to the high resistance to the air in travelling through the dust collecting medium, it required about one and a half hours to collect a sample of dust from 2 cu. ft. of air.

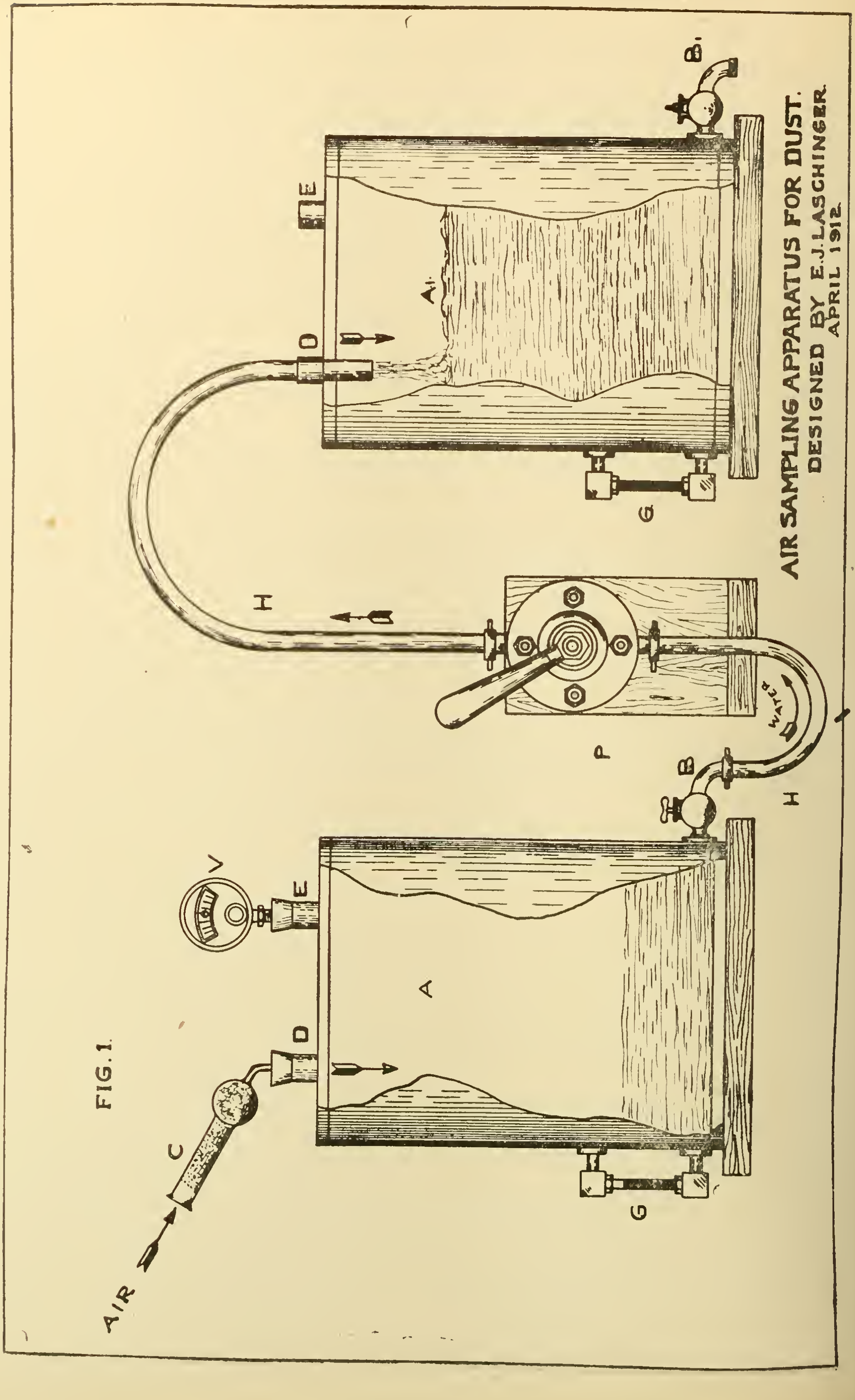
The first mentioned apparatus consists of two aspirator tanks in combination as shown in Fig. 1 herewith. Each of the aspirator tanks (A and A1) is fitted with a half-inch bib-cock (B) with screw hose connections and a gauge glass (G) near the bottom of the tank to indicate water level when the tank is nearly empty. On the top of the tank are two plain ferrules (D and E) into which rubber corks may be inserted to close them. The rest of the apparatus consists of a small hand pump and 20 ft. of half inch hose cut in two, one half being used for the suction and the other for the delivery of the pump. The method of taking a sample is as follows:—

One tank (A) is filled with water up to the top and the other (A1) is partly filled up to a mark on the bottom of the gauge glass. In one ferrule (D) on the top of the tank (A) is inserted a perforated rubber cork carrying the dust sampling tube (C) and in the other ferrule (E) a cork carrying a vacuum gauge (V). The pump is connected by its suction hose (H) to the Bib-cock (B) and the delivery hose is inserted into one ferrule on tank (A1). the other ferrule being uncorked aud the Bib cock (B1) closed. By opening the Bib cock B and operating the pump, the water is transferred from (A) to (A1) until (A1) is full and (A) is emptied to a mark on the gauge glass (G). A strong vacuum is induced in tank (A) which draws the air through the collecting tube. The object of the vacuum gauge (V) is to minimize the time required to complete an operation. After the water has been drawn from the tank (A) it may take a long time for the pressure inside the tank to balance with that of the outside air as the difference in pressure decreases slowly, but by observing the reading of the vacuum gauge and then quickly pulling it out, the volume of air at atmospheric pressure (free air) which has already been drawn through the collecting tube is easily calculated:—

Let v be the volume of water drawn from the tank.

Let B be the reading of the barometer in inches of mercury.

FIG. 1.



AIR SAMPLING APPARATUS FOR DUST.
DESIGNED BY E.J.LASCHINGER.
APRIL 1912.

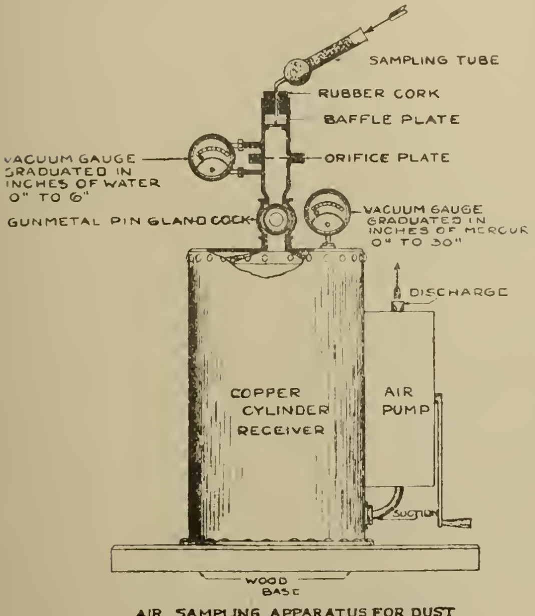
Let V be the reading of the vacuum gauge in inches of mercury at the time it was withdrawn from the tank.

Let Q be the volume of air at atmospheric pressure drawn through the collecting tube

Then $Q = v\frac{B-V}{B}$

The tank (A1) is now full and if the collecting tube and vacuum gauge be transferred to (A1'), the suction hose connected to the cock (B1), and the delivery hose inserted into (E), the water may now be pumped back and an additional amount of air drawn through the collecting tube. By a repetition of these reversals, any desired quantity of air may be sampled. In general, with fine sugar as the collecting medium, one operation will take about 5 min. instead of one and a half hours. This apparatus was hurriedly constructed locally and did not cost more than £7. It has been in successful operation now for a few weeks for dust determination in the streets of Johannesburg and in the mines.

While the above apparatus is satisfactory, it is not so convenient and portable as one could desire for underground work. With a view to having an apparatus which should be convenient in operation so that the work of conducting the tests and working out the results is reduced to a minimum, with weight and bulk as small as possible, another apparatus was designed. This is illustrated in Fig. 2.



AIR SAMPLING APPARATUS FOR DUST
DESIGNED BY E J. LASCHINGER
APRIL 1912.

FIG. 2.

This apparatus consists of a small dry air pump to be worked by hand, a small receiver to equalise the pulsation of air pressure in the suction of the pump and a small orifice through which the air is drawn after passing through the dust collecting tube. This orifice is the air meter and the flow of air is calculated from figures obtained by observing:—

(1) The barometer reading.

(2) The indications of a vacuum gauge connected to the receiver, and

(3) The indications of a gauge which gives the head or difference of pressure, between the two sides of the orifice plate.

The method of making a determination is as follows:—

Fix the dust collecting tube into the mouth at the top of the apparatus by means of a rubber cork and close the cock on top of the receiver; then start the pump until a certain predetermined suction is indicated on the vacuum gauge. Now open the cock and take the time of opening. Then by means of the pump, keep the vacuum gauge and the orifice gauge at constant readings till sufficient air has been drawn through the apparatus and close the cock, noting the time of closing. Also observe the reading of a pocket aneroid barometer. It is also desirable to record the temperature of the air. The rate of flow in cubic feet of air per minute multiplied into the time in minutes of the test will give the total cubic feet of free air passed through the apparatus. The rate of flow in cubic feet of atmospheric air per minute will be given by the formula:

$$Q = \frac{60\ CA}{144 \cdot P}\sqrt{2g\ RT}\ \sqrt{P_1\ h}$$

Where C is the co-efficient of contraction for a sharp edged orifice.

Where A is area of the orifice in square inches.

Where g is acceleration due to gravity in feet per second, per second.

Where R is a constant for air.

Where T is absolute temperature of the air (°F. + 461).

Where P_1 is absolute pressure of the air in receiver (lb. square inch).

Where P is absolute pressure of the outside air (lb. square inch).

Where h is head on orifice in lb. square inch.

Inserting the values for the constants and substituting

For P the reading of the barometer in inches of mercury = B.

For P_1 the reading of vacuum gauge in inches of mercury subtracted from barometer = B − V.

For h the reading of the orifice gauge in inches of water = H.

We have:—

$$Q = 3{\cdot}97 \frac{A}{B} \sqrt{(B-V)\,HT}$$

assuming for mine work on the Rand a mean temperature of 75°F. we get :—

$$Q = 92 \frac{A}{B} \sqrt{(B-V)\,H}$$

or for an orifice of say $\frac{5}{16}$ in. diameter :

$$Q = \frac{7{\cdot}05}{B} \sqrt{(B-V)\,H}$$

If for example a test were conducted for half an hour with a $\frac{5}{16}$ in. diameter orifice, the barometer reading 25 in. of mercury, the vacuum gauge 10 in. of mercury and the orifice gauge 5 in. of water, the amount of air passed through the apparatus would be

$$\frac{30 \times 7{\cdot}05}{25} \sqrt{(25-10)\,5} = 73 \text{ cu. ft.}$$

It might be mentioned there is a high degree of accuracy in this method of air measurement and in a calibrated apparatus there should not be more than 2% of error in the calculated result.

The over-all dimensions of the apparatus as designed are about 24 in. × 24 in. × 15 in. and the weight should not exceed 50 lb.

Mr. James Gray (*Member of Council*) : I have been using the first apparatus described by Mr. Laschinger for some time now and have found it to be very suitable for the purpose even although it is rather cumbersome. With regard to the time necessary to collect the sample I would point out that this should be 40 min., not 20, *i.e.*, 5 min. to transfer the 50 litres from one tank to the other and 5 min. to allow the vacuum to go back to atmospheric pressure, a total of 10 min. or 40 min. for 200 litres. This is a considerable improvement over the old apparatus which took 1½ hr. to collect 50 litres. The experimental error is also considerably reduced. The type of tube illustrated by Mr. Laschinger has been found most effective. But I think that the adoption of the tube suggested by Mr. Johnston would result in the time necessary for the collection of the sample being considerably reduced.

The thanks of those who have anything to do with dust collection are due to Mr. Laschinger for his work in this matter.

A Member : The time required also varies according to type of tube used ?

Mr. E. J. Laschinger : Yes. The tube there shown is most effective. I think the type of tube suggested by Mr. A. McA. Johnston would reduce the time of taking a sample very materially.

Mr. W. Cullen (*Past-President*) : The two papers which we have listened to have been extremely interesting, and the authors deserve our cordial thanks. It has just struck me, however, that an even simpler device could be used than that recommended by Mr. Laschinger. I do not say that it would be so accurate, but, in my opinion, it would be accurate within 5%, which is near enough for all practical purposes. The alkali inspectors in England use a particular type of bellows for sampling flue and chimney gases, and the degree of accuracy which they obtain satisfies them. If Mr. McArthur Johnston's appliances were fitted into the inlet of these bellows and a suitable exhaust valve provided, then all one would have to do would be to work these bellows like an ordinary concertina, and their capacity multiplied by the number of times they were filled, would give the amount of air which had been passed through. The apparatus I have described could be put into the care of an absolutely unskilled man, who would work at playing his "concertina" all day, while a chemist would deal with the dust caught in the sugar by the method prescribed by the author.

Mr. Laschinger's apparatus, I have no doubt, will give all the accuracy which is desired, but it strikes me as being somewhat cumbersome ; however, only those who have actually experimented with it are entitled to speak with any assurance on the point. The apparatus mentioned by both authors involves the principle of evacuation, and at Modderfontein we have adapted this same principle to the sampling of mine gases. The method simply consists of detonating the explosives under practical conditions inside an iron chamber, which is almost air-tight, and immediately after the explosion drawing out the gases by means of a pump which may be situated at any convenient distance ; obviously one cannot do the same on the mines, but as we have found out by practical experience that our imitative conditions produce exactly the same effect as is produced on the mines, the advantage gained is obvious indeed. Reference has been made by more than one speaker to the chance of the dust not having been completely removed from the mine gases, and this is a possibility which cannot be overlooked. I have little doubt myself however, that after experience has been gained the passage of the gases will be so regulated that it will be impossible for any dust to escape. There is, however, a very simple method of telling whether dust is in the gas or not, and I only mention it because of the fact that the Miners' Phthisis Prevention Committee are going into this question very fully just now. If one takes an ordinary piece of iron piping, 2 in. or more in diameter and 5 to 6 ft. long and closes up each end with a piece of ordinary glass, while keeping an electric light burning at the one end, it is possible by looking through the pipe from the other end to detect the slightest traces of dust, of any kind. At the Dynamite Factory we have used this simple test with great success for a number of years, but obviously to make it of any use for the work now being carried on, certain standards would require to be established and certain very definite comparisons made.

I have again much pleasure in proposing a hearty vote of thanks to the two authors for their thought producing papers.

Professor G. H Stanley (*Member of Council*): I would like to second the vote of thanks to Messrs. Laschinger and Johnston. There is just one point that occurs to me in connection with filtration: it is quite possible, I should think, for the air to be sucked through the sugar so rapidly that the filtration would not be efficient. I would like to know whether the authors have satisfied themselves that by this rapid method the results are the same as when working slowly.

Mr. F. J Pooler (*Associate*): In connection with the first example mentioned, I should like to know if the formula is taken for dry air. Should the pressure not be corrected for the maximum vapour pressures of water, as the air enclosed is saturated, I presume? Taking the temperature at about 26°C., this would make a difference of approximately one inch in the pressure, which may introduce an error of considerably more than 1%.

Mr. E. J. Laschinger: There is no correction there for the difference in moisture. Such a refinement is, to my mind, rather ultra-scientific for practical purposes. The difference between the vapour pressure of the air outside and in the apparatus is so small that it is practically negligible. When it comes to matters which effect the result to within less than 1%, I think refinements unnecessary. With regard also to the portability of the apparatus, or its being fit for underground use, the second apparatus was designed for that purpose, that is to say, an apparatus which is so portable and yet still so accurate for practical purposes that it may be used as a kind of standard apparatus for sampling mine air; and I am quite certain that there should be no difficulty in sampling at any point underground, because if the dust were very bad a man could wear a respirator and make a short determination; if the dust is in very small quantity and somewhere near what a prescribed minimum might be, then of course a man would take 15 minutes or half-an-hour to collect the sample.

Mr. G. Hildick Smith (*Member*): I should like to ask Mr. Laschinger if he has tested the efficiency of respirators by means of this apparatus at all?

Mr. E. J. Laschinger: I am sorry not to be able to answer the last question, and also wish to disabuse the minds of any members of this Society of the idea that I am an expert on dust determination. I simply designed this apparatus at the request of the Miners' Phthisis Prevention Committee, because the chemical and other expert members of the Committee were not satisfied with the apparatus that they had. They approached me as a mechanical expert to ascertain whether or not some apparatus could be designed better than the one they had before.

Mr. J. Gray: We have only collected samples of dust so far, but I understand that it has been suggested that the different types of respirators should be tested.

A SYSTEM OF KEEPING UNDERGROUND COSTS AND RECORDS.

By G. Hildick Smith, B.Sc. (Member).

From an economical standpoint and in order to obtain the highest efficiency and best results underground, on any mine, it is obvious that the official in charge of the underground department should know exactly what it is costing, in detail, to do all the various classes of work necessary in his department. In order that underground costs may be kept in such a way that they may be of actual practical use, the ruling prices of all stores and material required, the cost of labour, the cost of upkeep, for example, of the various types of machine-drills, together with their air-consumption, must be known. The figures relating to such items as these must be recorded in as simple and concise a manner as possible. Unless this is so the underground supervising staffs of the mines, cannot be working on the most economical lines, unless by accident. At present, practically no thought is taken by underground officials of the cost of the material used by them. Matters such as these are left to the storekeepers or mechanical engineers. It is now high time that a different state of affairs became the rule. Under the existing conditions it is practically impossible for any accurate estimate to be formed of the cost of doing any piece of work necessary underground, and it is therefore also impossible to choose the cheapest and best method of doing the work.

To improve matters in this res[illegible]he author has introduced the following syst[illegible]he Ferreira Gold Mine by which it is p[illegible] keep in touch with the cost of all [illegible]ng on underground, by means of tabulated cost sheets and graphic records. The idea of the system evolved is to give in sufficient detail, particulars, in regard to underground work solely, which are of actual practical use to the head of the underground department. At present the comparative statistics as published by the various mining groups for the edification of their consulting engineers, managers and others, are of very little, if any, use to the underground officials, as from such comparative statistics only more or less worthless comparisons may be made between the costs of the same classes of work at the various mines, as the costs are given in too general a form. For example, the hand-stoping costs for any mine show that for the total tonnage broken by hand throughout the mine, the cost on an average was 5s. per ton broken. From this it is impossible to see whether good or bad work was being done in any one stope. The same applies to all other classes of work underground to which only rule of thumb methods are at present applied, and often the results depend purely on guess work. The results of the introduction of some such system as is here described would be :—

(1) *Improved underground supervision*—the old style of "hustle" will be gradually replaced by more thoughtful work by shift bosses, mine captains and underground managers.

(2) *A more settled condition of the underground white labour force*—men could be judged from the actual results of their work, and not by the fact that they may be found at any moment apparently doing nothing.

(3) *A maximum of safety and economy* in all underground work.

(4) *A reduction in the accident rate* by the keeping of accident statistics for each shift boss' section, and by the introduction of a bonus scheme, the bonus to be paid to the shift boss obtaining best results, thus encouraging shift bosses to use more care and thought in the prevention of accidents.

(3) *A gradual improvement in the class of miner* on this field.

(6) *The further introduction of well-trained men on the underground supervising staffs* of the mines.

(7) *The speedy eradication of the phthisis evil* by the decrease in the "hustle" evil and the improvement in the general system and supervision of the work underground.

These are some of the results which would undoubtedly be obtained eventually by the general introduction on the mines, of a system of keeping in detail, figures relating to all underground work, its cost, the efficiency obtained and the accidents that occur.

The system in vogue at the Ferreira Gold Mine. Method of obtaining cost of Stores.—In order to arrive at the cost of any detail of the underground work, quickly and simply, all orders for stores, such as gelatine, candles, fuse, timber, or spares, are made out in duplicate, one copy being filed in the office of the underground manager, the other copy for which the stores ordered are issued, being taken to the store. As the shift bosses know best whether or not their men really require certain stores, and as the costs for their sections are an important part of the system here described, all orders, for the usual class of stores required, are issued by each shift boss to his own men. This is an important point, as by this means only, is it possible to effect the strictest economy—this cannot be the case when the detailed ordering of stores is done for example by mine captains, who are not so closely in touch with the men as are the shift bosses. By the ordering of stores on duplicate forms, any alteration of the order form by unscrupulous miners can be at once detected. The general ordering of such materials as timber, rails, sleepers and pipes is done by the head of the underground department. On the order forms as issued by shift bosses are written the miner's name, the stores required, and the class of work for which they are to be used. The duplicate orders are filed in the office in alphabetical order of the names of the miners for whom they are made out. Thus it is a simple matter to obtain the cost or quantity of stores used by any miner over any period.

White and Native Labour Costs.—Particulars as to the cost of white labour are obtained from the white time office. The type of white time ticket used is as shown in Fig. 1 :

WHITE TIME TICKET.

Pay Roll No.......... Date.................................1912

Day or Night Shift.......... .....

DETAILS OF WORK.	HOURS.
....	....
....	....
....	....
....	....
....	....

No. of Natives Employed......................
(Underground Only)

Workman's Signature..... ....

Foreman's Signature......... ....

FIG. I.

The shift bosses see that under 'Details of Work," the correct heading is given. This is done before the shift boss countersigns each of his own men's tickets as correct. Each employee places opposite the words "No. of natives employed," the number of natives he has working with him during the shift. Particulars as to the cost of native labour for each class of work are then arrived at by taking the number of native shifts from the white time ticket and allowing an average rate per shift. This gives a cost for native labour near enough for the purpose required. An exact figure can be obtained from the native time office when necessary, otherwise all native time on the same class of work all over the mine is placed under one head and cannot with ease be obtained for each working place.

Cost of the various Classes of Work.—In order therefore to obtain the actual cost of, breaking ground in any stope, reclaiming an old drift, robbing pillars, or other work, the costs of the following items have first to be determined :—

(1) Drills, drill-sharpening and drill-distribution.
(2) Machine drill costs (*a*) upkeep, (*b*) air-consumption, (*c*) spares, (*d*) depreciation.
(3) Pipes and pipe-fitting.
(4) Track-laying.
(5) Trucks and truck repairs.
(6) Tramming.
(7) Timbering.
(8) Stone-walling.
(9) Sundries, *e.g.* saw sharpening, assaying and sanitation.

Such items as the cost of pumping, ventilation and hoisting can be kept as special accounts.

(1) *Drills, Drill-sharpening and Drill-distribution.*—All drills are stamped with a number and each miner has drills stamped with his given number. The first cost of all new drills issued to each miner and the cost of sharpening any

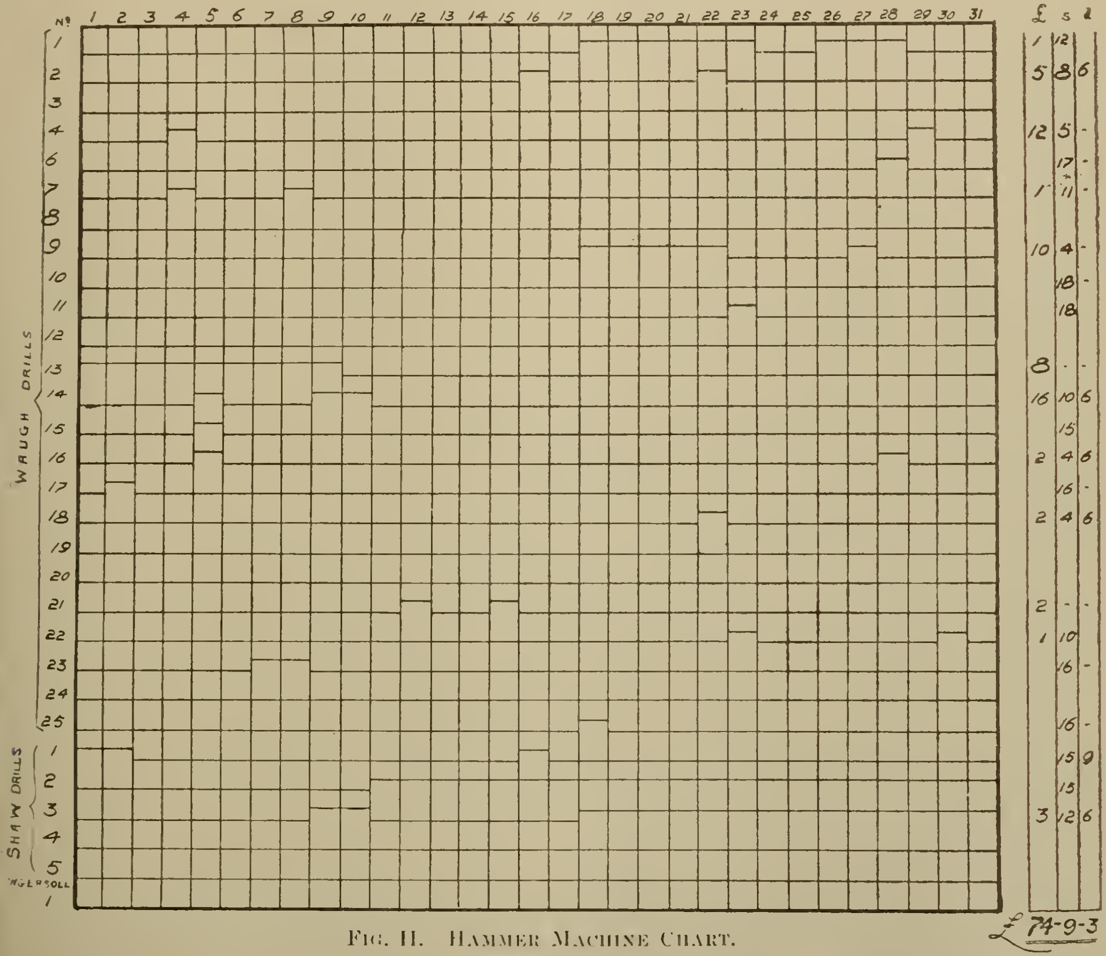


Fig. II. Hammer Machine Chart.

Labour cost per machine repaired = 15/-. Average cost per machine repair effected, £2 9s. 7½d.
449 Machine Shifts. Cost per shift, 3s. 4d.

class of drill is obtained from the mechanical engineering department. The white employee in charge of the drill-distribution, gives in the number of drills issued to each miner per shift and from this the cost of drill-sharpening, for each stope or other working place, is easily arrived at and to this cost is added, in proportion, the cost of labour for drill-distribution.

(2) *Machine Drill Costs.*—The cost of running machine-drills consists of :—

- (*a*) Upkeep.
- (*b*) Air-consumption.
- (*c*) Depreciation.
- (*d*) Spares issued to miners.
- (*e*) Rigging gear, *e.g.* arms, bars and clamps.

All machine drills are stamped with a number, the numbers of machines issued to each miner are known, all spares and rigging gear required by the miner are ordered by the shift bosses, so that the cost of the same can be arrived at, at the end of each month. The air consumption for each type of machine is taken by a Kent & Sons Manometer and an average figure arrived at for each type, then by allowing an average drilling time, the air consumption per machine per shift is obtained. The cost of upkeep of the machine-drills, that is the actual cost of the machine-fitting and repairing done in the fitting shops is kept on a chart as shown in Fig. 2.

From which may be seen at a glance the time during which any machine is on the surface, the repairs done to the machine while it was at the surface and the cost of same. Separate charts are kept for the reciprocating and hammer types of machine drills. At the end of the month an average cost for repairs per machine per shift is then an easy figure to arrive at—the number of machine shifts being kept in a special book for that purpose in which the machine shifts are entered up by shift bosses.

(3) *Pipes and Pipe-fitting.*—Pipe-fitters and pipe-fitters' native labour costs are obtained in the usual way, the same applies to all pipes and pipe-fittings ordered. The total cost of pipe-fitting is apportioned over all sections of the mine.

(4 and 5) *Track-laying, Trucks and Truck-repairing costs* are also obtained in the same way and apportioned over all sections of the mine.

(6) *Tramming.*—Tramming costs are worked out to give a cost figure per foot-ton trammed, the distance trammed being obtained from the survey department and the tonnage from the truck tally book. The tramming efficiency and cost of tramming may then be worked out separately for any one working place as required. The system of keeping and checking the tally tonnages throughout the mine is described under the heading of "Truck Tally System."

Fig. 3 shows the cost figures for plate-laying, pipe-fitting, truck repairs and sanitation for March 1912.

(7) *Timbering.*—All timber required by miners for use in stopes and drives is ordered, on duplicate order forms, by the shift bosses. On the order is stated the level to which the timber is to be delivered and the name of the miner requiring the timber. Thus the cost of timbering each working place can then be easily obtained. The stock of timber in the timber-yard is taken at the end of each month when a check on all timber used is made by taking into consideration the quantity of new timber delivered at the mine during the month and the quantity sent underground. As a further check the banksmen at each shaft record the number of pieces of lagging sent underground.

Fig. 4 shows the form of timber chart kept from which the various quantities of all classes of timber used and the cost of same for each month may be seen at a glance :

(8) *Stonewalling.*—The tonnage of either reef or waste packed in any stope or working place is obtained from figures kept in the survey office, the cost of this work per ton packed or per ton broken is calculated and allowed for either for each stope or over the whole mine.

(9) *Sundries.*—These include such items as sanitation, saw-sharpening and assaying, for some of these items such as sanitation a separate account is kept (see Fig. 3), in other cases where the item is a small one it is charged under the heading of "Sundries" to other accounts. In the same manner the running cost of pumping, ventilating and other plant in use underground can be obtained as required.

Shaft Repair Records.—A cross-section to scale of each shaft is kept on which the class of timbering, or what not, in any one section is shown with notes as to its condition. As repairs and renewals of timber, rollers and rails in the shafts are made, they are noted on the chart with the dates on which the work was done. The condition of the shafts throughout can then be seen at a glance. Note is also made of points where repairs are required. A summary of the work done and its cost is recorded for each month in tabulated form, on a sheet attached to the cross-section.

Truck Tally System.—In order that the costs for each shift boss' section, may be compared month by month, it is necessary to have as good a method as possible of obtaining the correct tonnage trammed from each working place during the month. The shift bosses must in the first

Fig. III.

UNDERGROUND SANITATION.

1912	Name.	White Shifts.	Native Shifts.	White Wages.	Native Wages and Food.	Disin-fectant.	Lime.	Sundry.	Total.	Cost per Ton Mined.	Remarks.
March	...	26	140	£19 10 0	£20 10 0	£1 16 0	£3 0 0	£1 0 0	£45 16 0	·47d.	

PLATE LAYING AND TRACK REPAIRS.

1912	Name.	White Shifts.	Native Shifts.	White Wages.	Native Wages and Food.	Stores.	Total.	Cost per Ton Mined.	Remarks.
March	...	26	156	£26 0 0	£22 2 0	£8 9 8	£56 11 8	0·58d.	

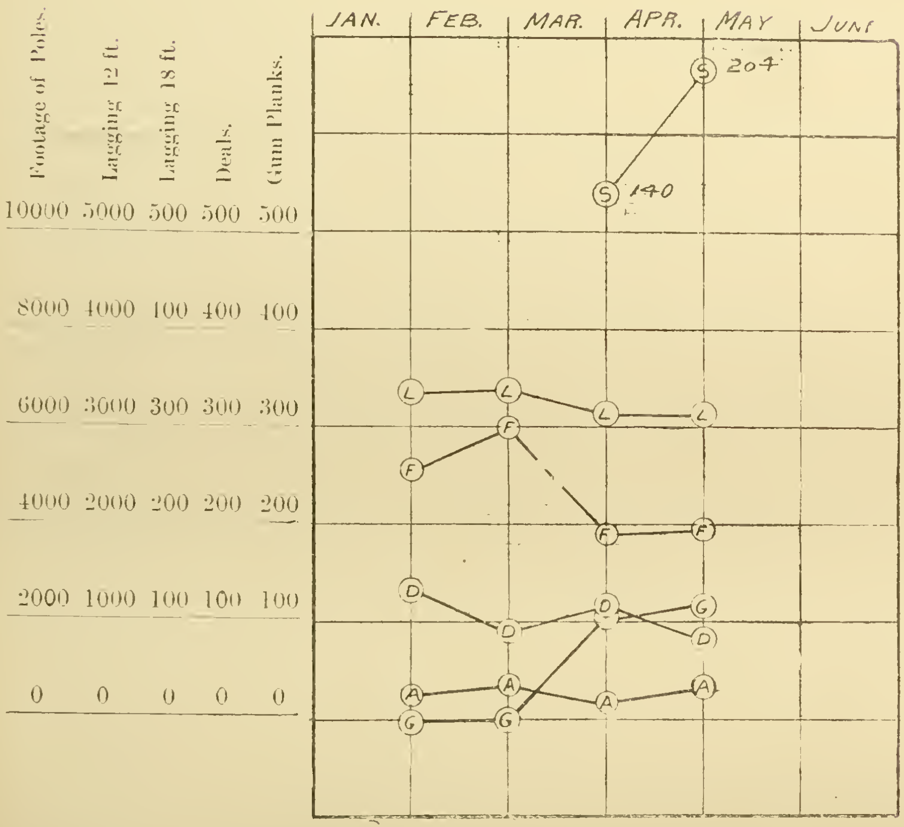
PIPE FITTING.

1912	Name.	White Shifts.	Native Shifts.	White Wages.	Native Wages and Food.	Stores.	Total.	Cost per Ton Mined.	Remarks.
March	...	50	92	£50 0 0	£13 3 9	£28 8 4	£91 12 1	0·93d.	Two Pipe-fitters employed.

TRUCK REPAIRS.

1912	Name.	White Shifts.	Native Shifts.	White Wages.	Native Wages and Food.	Stores.	Total.	Cost per Ton Mined.	Remarks.
March	...	37	52	£32 11 4	£7 16 0	£16 12 9	£57 0 1	0·57d.	These figures represent repairs effected both underground and on surface in Boilersmith's Shop.

FIG. IV. TIMBER CHART.



Tons Mined.	Poles.	12 ft. Lagging.	18 ft. Lagging.	Deals.	Blue Gum.	Totals.	REMARKS.
23972 Jan.	3·23d.	1·41d.	·04d.	·61d.	—	5·29d.	
22917 Feb.	3·93d.	1·48d.	·06d.	·43d.	—	5·90d.	
23425 Mar.	2·17d.	1·34d.	·03d.	·55d.	·22d.	4·73d.	Sleepers 12d.
April							
May							
June							

TIMBER. COST PER TON MINED. 1912.

Fig 8.

MARCH, 1912.—COST OF BREAKING GROUND BY MACHINE DRILLS.

Section.	Type of Machine.	Stope No.	Stope Width.	Stope Value.	Fathoms Broken.	Measured Tons.	Tally Tons.	White Shifts.	Native Shifts.	Machine Shifts.	Miner.	Native Wages and Food.	White Wages.	Gelatine.	Gelignite	Detonators.	Fuse.	Sundries.	Drills and Sharpening.	Machine Maintenance.	Compressed Air.	Spares.	Timber.	Packing.	Supervision.	Tramming Costs.	TOTAL.	Cost per Ton Broken.	Fathoms Broken per Machine per Shift.	Tons Broken per Machine per Shift.	Cost per Ton-Dwt.	REMARKS.
												£ s d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.					
IV.	R.	B.S. 320 M.R. & L.	108"	..	58·03	1567	672	21	213	78	A.	33 19 9	35 9 2	46 0 0	4 5 0	0 5 0	0 19 4	2 8 0	3 17 0	4 12 6	22 8 6	2 5 0	1[illegible] 7 2	Nil.	18 5 8	26 0 0	211 2 1	2/8½	0·74	20·09	.	
II.	R.	1520/9 M.R.L.	72"	...	71·53	1318	1091	24	262	100	B.	41 15 0	59 4 1	77 12 6	0 0 0	0 10 0	1 18 8	4 0 8	4 18 0	6 3 4	28 15 0	1 5 0	[illegible] 7 3	24 18 1	15 7 6	1 9 5	274 4 6	4 2	0 71	13·18		T[illegible] installed [illegible] figure. [illegible]
III.	H.	B.S. 620 M.R.L.	72"		64·75	1363	1177	24	174	112	C.	27 4 4	90 16 1	40 5 0	2 2 6	0 17 6	3 7 8	3 13 2	11 18 0	10 3 7	26 16 6	0 10 0	45 2 4	Nil.	15 18 0	40 7 6	319 2 2	4 8	0·58	12·17	...	Tonnage [illegible]

Fig. 9.

MARCH, 1912.—COST OF BREAKING GROUND BY HAND LABOUR.

Section.	Stope No.	Stope Width.	Stope Value.	Fathoms Broken.	Measured Tons.	Tally Tons.	Miner.	White Shifts.	Native Shifts.	Native Wages and Food.	White Wages.	Gelatine.	Gelignite.	Detonators.	Fuse.	Sundries.	Drills and Sharpening.	Packing.	Timber.	Tramming Costs.	Supervision.	Box-holing.	TOTAL.	Cost per ton Broken.	Cost per Ton-Dwt.	Broken per Boy per Shift.		REMARKS.
																										Tons.	Fath'ms	
										£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.		£ s. d.					
IV.	1·320 M.R.L.	54"		23·69	351	293	D.	25	376	66 [illegible] 6	25 0 0	14 7 6	2 2 6	0 10 0	1 18 8	1 18 0	1 2 6	Nil.	4 10 4	11 19 9	4 1 10	...	133 16 7	7 7½	.	0·96	0·06	Footage = 8ft.
III.	Open Cut	64"	..	...	1404	1130	E.	25	513	93 5 8	39 13 8	...	19 2 6	1 0 0	3 17 4	1 5 4	1 2 6	Nil.	43 6 4	44 17 8	16 7 7	...	263 18 7	3/9	...	2·7	...	Footage = 65ft
I.	1520/1 S.R.	48"	...	67·67	812	778	F.	23	394	55 15 6	75 7 6	11 10 0	6 7 6	0 15 0	0 19 4	1 10 4	1 4 0	50 3 4	10 14 0	57 9 8	9 9 5	...	281 5 7	6/11	.	2·06	0·17	421 tons packed—very [illegible] packs carried close up [illegible]

place be made responsible for the correct tallying of all the broken ore trammed from their sections. The form of tally ticket used is as shown in Fig. 5:

TALLY TICKET.

Date.................Day or Night Shift....................
Working Place..
Tramming Level...
Shaft No...
No. of Trucks Trammed..................................
Number of Boys { Tramming..............................
{ Shovelling............................
Trammer's Name...
Shift Boss...

FIG. V.

All tally tickets are initialled by the shift boss of the section to which they refer. The shift bosses sections are numbered 1, 2, 3 and so on, the section number being stamped on the ticket. The number of trucks trammed is entered in the truck tally book in column form, a separate column being alloted to each working place. As mistakes, intentional or otherwise, are always possible in dealing with truck-tally figures, it is necessary that a check should be made on the tallying for each level—the skip tonnages hoisted from each level are therefore taken. These are obtained correctly by measurement in trucks, each holding one ton of broken ore, into which all the ore hoisted passes before being tipped on to the grizzlies. The tonnage figures thus obtained are plotted on the level tonnage chart as shown on Fig. 6.

From these are compiled charts showing the tonnage hoisted at each shaft for each day, a summary of the whole is shown on the "Gross Total Tons Hoisted" Chart Fig. 7.

The tonnage charts for each month are finally pasted together and the rise or fall in the tonnage mined over any period, may then be clearly seen. The result is that the underground tallying is checked, and the proper filling of the skips by skipmen can also be checked as the number of skips hoisted from each level is also recorded in a book kept for that purpose at each shaft. The best work in hoisting and tramming generally is thus encouraged,

Shift Boss Sectional Costs.—By a system such as described it is a simple matter to obtain the cost and efficiency of all work done in any one section of the mine, and this in the author's opinion is most important, as by keeping figures showing, what tonnage has been mined, tonnage per boy shift, cost of timber used, total cost per ton mined for each shift bosses section, the results are that shift bosses become trained to look upon their work in a new light—a greater interest is taken by them in their work and by the adoption of a bonus scheme, a bonus being paid to shift bosses on the results of the month's work in their respective sections, the best work is further encouraged. In this way good or bad work by different men can be at once detected. The unsettled condition of the underground white labour force on the Rand would be improved upon by taking into account the number of dismissals of workmen by each shift boss or mine captain, if figures relating to this for each section were continually kept, provided those in authority were really desirous of improving on the present unsettled state of affairs in this respect.

Conclusion.—To-day, any figures relating to the cost of the various classes of work, such as cutting shaft-stations, pump-chambers, re-timbering shafts, or what not, are conspicuous by their absence. This being so it is impossible either to estimate the cost of doing any such class of work or to know the cheapest and best method of doing it. The same applies to the cost of breaking ground in the various stopes of a mine, where in order to obtain the best results, the cheapest methods of breaking ground and supporting the hanging-wall under the existing conditions should be known. In order that this may be the case, such tables for example as are shown in Figs. 8 and 9 relating to machine and hand stoping respectively must necessarily be kept. The same applies to development, reclaiming and other work.

By means of the system described the cost of any piece of work underground can be arrived at and all costs, as they should be, are ready to hand for the edification of those in control of the underground department. The advantages thus gained by some method such as described, over the, at present, too common methods of guessing the best and cheapest way of doing any underground work, are obvious. New methods of working can be experimented with, figures showing the exact results obtained being properly recorded. The best methods of working can then be chosen with certainty. The greatest scope for improvement on the mines of the Rand is in the underground departments, but no improvement can be made unless it is first known what results are actually being obtained. At present in respect to the underground work, the results obtained in detail, with regard to costs and efficiency are unknown. It is noticeable that a change is taking place in respect to underground supervision on the Rand. The personnel of the underground staffs of the mines is undergoing a change, technically trained men, who have had the necessary practical experience, are replacing the purely practical men. The time for the

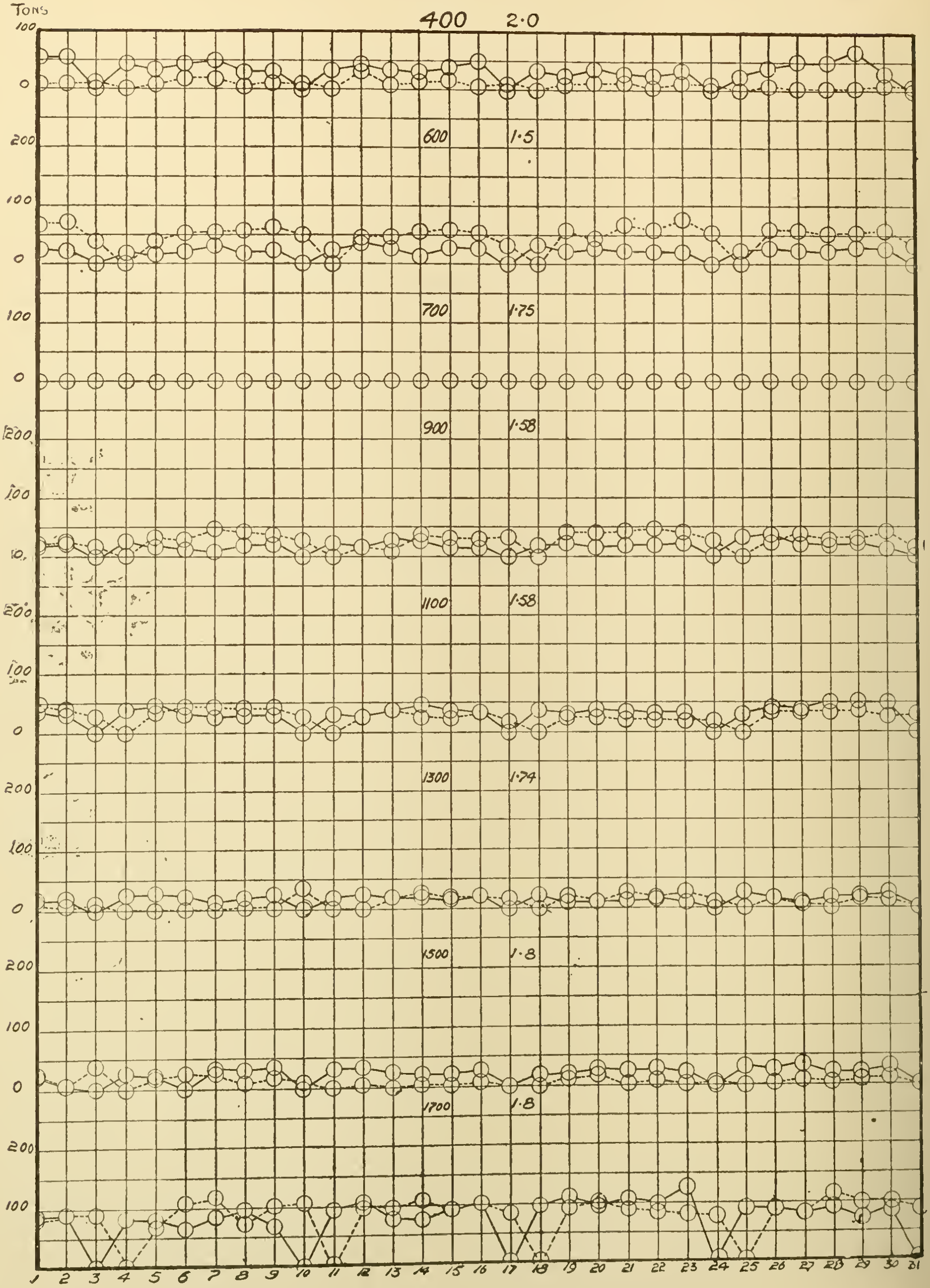


Fig. VI. No. 1 Shaft—Level Tonnages—March, 1912.

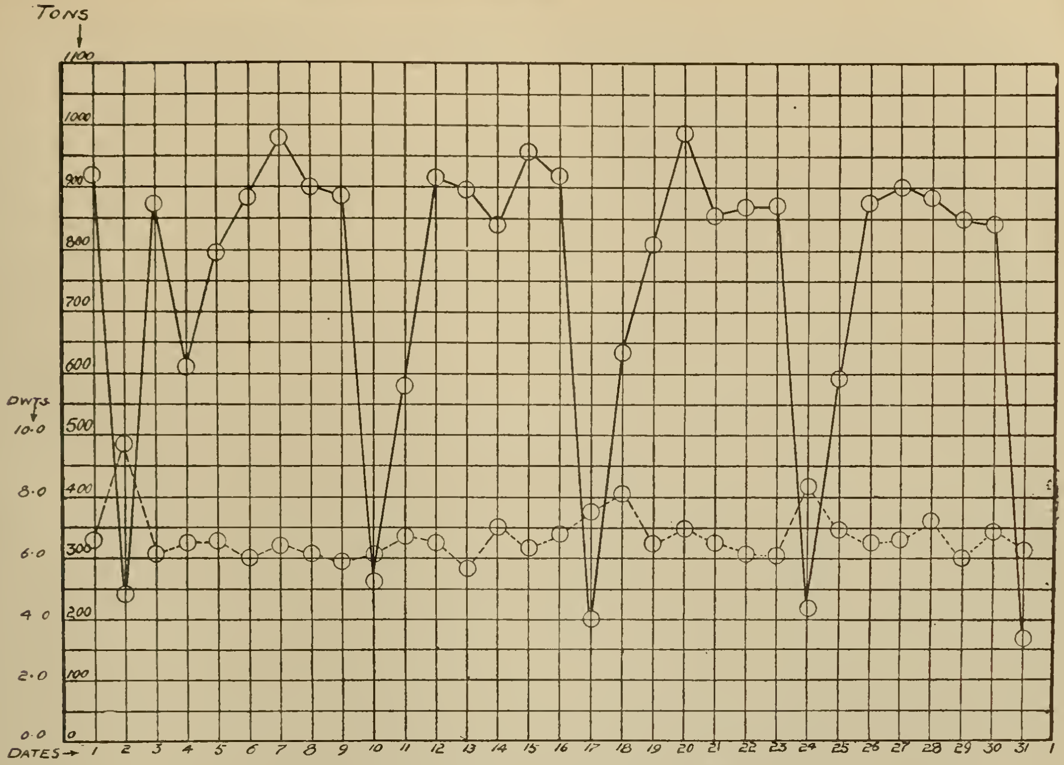


Fig. VII. Gross Total Tons Hoisted. March, 1912.

adoption of some such system as described, has certainly arrived and a system of keeping complete and detailed underground costs should be standardised by each of the mining groups. If only half the clerical work, at present being done on the compilation of head office returns for the various groups, was done in keeping in detail figures and returns relating to the underground work only, money which is to-day being wasted on underground work would be saved, and in addition health or other conditions underground would be improved upon. Work below the surface is being carried on in the dark in every way, and it is time that those responsible for the work going on in the underground departments of the mines of the Rand adopted new methods in order to improve on the careless work which is too commonly done.

The author hopes that this paper will be fully discussed by many of the mining members in order that the best method of keeping figures relating to underground work may be devised and generally adopted.

Mr. A. Richardson (*Member of Council*): Statistics are always dry and, according to Disraeli, rank very low in the scale of authenticity; our thanks are, therefore, due to Mr. Hildick Smith for presenting the matter of systematising underground costs and records in an interesting and suggestive way and for proving, or rather for endeavouring to prove, that the extension of our present system of cost-keeping will furnish us with more reliable data than we possess at present and be a real benefit to the industry. Of the difficulty of arriving at the exact cost of any piece of underground work most engineers are aware, and hence the difficulty of estimating what a proposed piece of work will cost. Even with the best system that can be devised it is, in view of the ever changing conditions, difficult to see how more than a moderate reliance can be placed on figures obtained from past work when applying them to work ahead. As the author states: "the greatest scope for improvement on the mines of the Rand is in the underground departments"; and it may be pointed out here that this statement could have been made with equal truth in any of the years that intervened between the time when the cyanide process came into its own and the present day; but this of course does not do away with the urgent necessity of applying science and system to underground work; and we must always give a hearty welcome to any suggestion which may tend to better conditions underground and improve the efficiency of the work done there.

Mr. E. J. Laschinger (*Vice-President*): This subject is not by any means a new one. I appreciate the paper nevertheless, because I think it deals with a subject the importance of which, to some extent, is lost sight of. At the outset, I must say that it is quite impossible to obtain any great degree of accuracy with regard to underground costs. The work is so varied, the staff is always changing and there are so many factors which are not the same that to get figures which are comparable from time to time is, to some extent, hopeless and, to some extent, useless. The idea of getting costs pertaining to shift bosses' sections, for instance, is a very good one, because one can check the work of the same shift boss throughout his career on the mine and it provides one with the means of keeping him up to the scratch in his work, because he can see the progress that is being made and therefore does his very best; and one can compare, to some extent, the work of one shift boss on a level with the work of the next shift boss who takes up that level. At the same time, these statistics are not strictly comparable because the nature of the work changes from day to day, more from week to week, very much more from month to month and enormously from year to year. If one goes to a mine three months hence it will be found that the same level is a different proposition altogether from what it is to-day. Another subject with which I have been closely identified for a good many years now is the matter of air consumption in rock drills. After all the detailed study which I have made of this subject, the tests and the actual experiments to show good working conditions with regard to air consumption and rock drills, I seem to know less about it as time goes on. We may, perhaps, in the near future, have some means of determining exactly (commercially exactly) how much air is used in each level. I think that is quite within the bounds of possibility; but, in order to do that, it means more expense. Air meters and their supervision cost money, and naturally we must look to those responsible for the management of the mines to see whether the money they would spend on such instruments and supervision, the taking of the readings and the compiling of statistics is worth the expense. Personally, I think it is: but, at the same time, I have never yet seen any figures on the air consumption of rock drills put down on statistics on which I could place implicit reliance.

I believe, in connection with this matter of underground costs that if we had the experience and opinions of members from Kimberley they could do the members of this Society a very good turn by explaining to us in a criticism of this paper, or as an original paper, the system by which the De Beers Mine encourage their underground mine workers in supplying statistics.

I think it was about five or six years ago that Mr. C. D. Leslie, the then Manager of the Simmer and Jack Proprietary Mines, introduced a very complete system of underground statistics on that mine, and I think he touched one very crucial point in the whole business of statistics as encouraging workers, by having these statistics compiled from day to day, so that the miner could see to-day the results of his work yesterday. When one considers the enormous quantities of dynamite, candles, drills, timber and all the other materials which are consumed underground on every shift, in order to do any real good the Manager must keep his pulse upon this business daily. He ought to know to-morrow morning the result of to-day's work, so that he can at once find out where the trouble is, because I do not think anybody's memory is so short as that of the miner or man who has to do underground work: they have so many different things to do during a shift that what was done a week ago is absolutely lost to memory in that short space of time; I do not throw any blame on them for it at all, because I think "Sufficient unto the day is the evil thereof," and when the day's work is done one does not want to go back on the details as to what was done on any particular shift a week or a month ago. I would like to prepare some better criticism than I can offer here verbally to-night, because I think the paper is worthy of discussion before this Society

ACCIDENTS IN TRANSVAAL MINES.

(With Special Reference to Falls of Ground.)

(*Read at November Meeting, 1911.*)

By James Chilton (Member).

REPLY TO DISCUSSION.—PART I.

Mr. James Chilton (*Member*): The author desires to thank those members who have contributed to the discussion of this paper and congratulates himself upon breaking the solemn stony silence that has so long hung over the mining members of this Society. After hearing the criticisms and re-reading the paper the author sees no reason to recede from the position taken up. The charge made, that there is an excessive sacrifice of human life going on in the Transvaal Mines has not been disproved either by argument or by evidence and the figures given have passed unchallenged. The paper was written as an appeal to the mining conscience, and though the statements made were frank and free they were made deliberately with full knowledge of their import and the writer can find nothing to regret and nothing to recant. Probably the most remarkable feature of the discussion has been its perfect candour and openness. There has been no desire to conceal our mining sores, no fear of offence in high technical circles, but every contributor has been animated by a deep sense of the gravity of the question, and though all classes of technical workers from consulting engineers to Kafir drill-boys have been subjected to a good deal of criticism, this criticism has been such as to give offence to none.

The author feels that our discussion will be fruitless unless we can arouse public opinion to a sense of the importance of this subject. Unless a fierce ethical light beats upon the squalid facts of our industrial life there is a danger that discussion will be useless and a waste of energy. Any machinery or any method of working that is likely to increase the danger to life and limb should be the subject of special investigation by an officer representing the Government, and every effort should be made both by magnate and by miner to improve the industrial conditions. This is the day of Royal Commissions, and the author would plead for the appointment of a commission to enquire into the causes of our excessive accident rate. England with an accident rate only one-fourth as large as ours appointed such a body, and the reduction in their accident rate this last ten years may be traced to the labours of that commission.

One difficulty of the management is to express in pounds, shillings and pence, the result of safety measures, for the result of a safety campaign is not always apparent, still its effects do appear in the dividend list, and accident prevention has its economic as well as its humanitarian aspect. Managers can easily point out the cost entailed by trying to prevent accidents, but it is difficult to show where an accident was avoided, and the money saved thereby.

Many of his critics have questioned the wisdom and the fairness of any comparison between the Transvaal and the English fatal accident rates. It is readily admitted that there are many points of difference between the mining practices of these countries, but in any comparison between coal and metal mining, the advantages of all are in favour of the latter. Coal mines in England are invariably troubled with explosive gases and coal dust, and an ignorant lad often carries the lives of hundreds of his fellow-workmen in his careless hands. A defective safety lamp, an electric spark from a coal cutter, the ignition of a friction clutch, a forgotten match, a reckless miner desiring a smoke, all these trivial causes have resulted in disastrous explosions. In 1910 England had 18 explosions of gas or coal dust, which cost 500 lives. This form of accident does not occur in the Transvaal mines, but in England it is sadly too common. Then again the character of the country rock differs in coal and metal mines. The latter usually has a hard country rock of tough texture such as granite or hard quartzite, needing little or no support, so that falls of roof and sides, are not as frequent in metal mines. On the other hand coal mine roofs are often composed of shale short in texture and clayey in nature, which readily drops away around the head of the timber supports, so that in the picturesque language of the Northern miner "The timber hes te gan in afore the coal comes oot." As coal beds were originally laid down in a swamp, saturated roofs are the rule, and falls of roof and sides are naturally numerous. The fire clay floor also swells and becomes soft and the props are slowly forced into it, thus depriving the roof of the needed support. The startling difference between the two country rocks is clearly shown by comparing the accident rates from falls of ground in England and in the Transvaal. In 1910 falls were the cause of 40% of English accidents which resulted in 634 deaths, while over the same period, the Transvaal rate was 24% of the total death rate, and the numbers killed were 268. Accidents on haulage roads due mainly to meeting the haulage train or sett caused the death of 234 persons in England. This is a class of accident that has not yet appeared in the Transvaal lists, but as a few underground haulages are being installed this class of accident will soon make itself felt. In comparing coal mines and metal mines the quantity of material excavated must also have a bearing on the accident rate, for except in long wall workings the bigger the space excavated the larger is the roof area that needs support and the greater the probability of falls of roof and sides. As England in 1910 raised over 271 million tons against the Transvaal's 26 million tons, the difference in the area of the exposed surfaces is at once apparent. The State Mining Engineer in his report for 1908 supports this view, for he says "It is evident that the rate at which ore is extracted—must have a considerable bearing on the number of accidents and persons killed." Yet in spite of our advantages due to smaller tonnage extracted and its greater specific gravity our death rate is 31·5 per million tons against 6·54 the English rate. The author submits that the charge of unfairness has not been proved, and that in any comparison between coal and metal mine risks the advantages are all in favour of the latter. The Johannesburg correspondent of the *Mining Magazine*, writing on Transvaal mine accidents under date March 1912, says "The high death rate from accidents on the Witwatersrand is certainly not due to these mines being more dangerous than mines in other parts of the world. On the contrary they should be safer." With this expression of opinion the writer cordially agrees.

It has been gently suggested that this paper seeks to establish hopelessly ideal conditions; that it is the work of a visionary intent upon counsels of perfection; that the English accident rate standard is too high for our local conditions. Many members have spoken of the language difficulty, the ever changing force of native workers, the dearth of skilled miners, the nomadic habits of mine managers, and the ceaseless tide of change, all have been brought forward as excuses for, or explanations of high death rates, but none have pointed out, that in spite of these obstacles, a goodly number of our mines have a death accident rate below the average English rate. Too often is it assumed that accidents cannot be avoided, but the list of mines given below shows what can be done if only some heart is put into the work. Taking the 66 producing mines that appear in the accident list for the year ending 1910, it will be seen that 22 mines have a rate that is less than 1·91, the English rate for that year, and four mines, the Primrose, the Simmer Deep, the Knights Deep and the Village Main Reef have accident rates that are below the million tons rate of England. That this immunity from accident has not been secured by sacrificing dividends is evident from the list produced in Table 1.

If these mines can keep their accident rate down to reasonable proportions, surely it is not a counsel of perfection to ask others to fall into line with them. It will be noticed that these mines are scattered along the Rand and their freedom from a high accident rate is not due to anything peculiar in their local conditions. They have every difficulty that is common to mines, yet in spite of these their accident rate remains low.

TABLE 1.

Death Accident Rate for six months ending December, 1910.

Gold Mines.	No. of Employees.	Accident Death Rate.	Per million tons hauled.	Dividends for year 1910
New Minerva.. ...	247	Nil.	...	...
West Rand Central ..	647	,,	...	...
Treasury	788	,,	...	...
Salisbury	658	,,	...	15
Champ d'Or	985	,,	...	20
Wolhuter	2,529	,,	...	10
New Princess.. ...	1,464	,,	...	20
New Unified	1,558	,,	...	10
New Heriot	1,482	,,	...	80
Meyer & Charlton ...	1,361	,,	...	40
New Primrose ...	2,011	·99	5·02	40
Main Reef West ...	1,934	1·03	..	25
Geduld Proprietary ...	1,816	1·10	..	...
Consolidated Langlaagte	3,000	1·33	..	...
Roodepoort Central Deep...	1·428	1·40	...	...
New Rietfontein ...	2,707	1·48	...	...
West Rand Consolidated	2,699	1·48	...	3½
Simmer Deep... ...	3,793	1·58	5·20	...
Knights Deep ...	3,717	1·71	4·76	30
Village Main Reef ...	3,488	1·72	5·85	70
Durban Roodepoort ...	1,114	1·80	...	40
Roodepoort U.M.R. ...	3,081	1·83	...	5

Mr. W. S. V. Price's positiveness of statement is probably due to the sincerity of his convictions, but I fear his sweeping condemnations are often unjust. He sneers at various classes of underground workers as if intelligence were the monopoly of the hand or machine stoper. His attempt to hold a mine inspector responsible for a fall of ground that took place a week after his visit is simply amusing. As he points out, the language difficulty is certainly productive of accident, but the problem is difficult to solve. In America, where this question has aroused some interest, the mining regulations are posted up in seven or eight languages, but it was found that this procedure was not effective as many foreign miners could not read the regulations even wh written in their own language. To prohibit persons unacquainted with English working in our mines would be too drastic a measure in a cosmopolitan mining district like the Transvaal, though this policy is carried out in Germany where every person working in the mines must be able to speak the language of that country. Some years ago, the coal miners of Fife and Lanarkshire interviewed the English Home Secretary in reference to the large number of Poles that were employed in the mines of these counties pointing out that their presence was a distinct danger through their ignorance of the English language, and asked that these miners should be prohibited from engaging in mining work. The Home Office did not entertain the idea, and refused to interfere in the matter, maintaining that Fife and Lanarkshire had an accident death-rate that compared favourably with other mining districts and therefore the need for prohibition was not apparent. The author is connected with a mine that had 450 natives from tropical Africa drafted to it. These workers are totally ignorant of Rand Kafir, and not a single man underground can speak their language, yet in spite of this difficulty we have not yet had a serious accident to any of these natives; a result that has only been brought about by great care, and attention to safety conditions. The writer believes that even the language need not unduly swell the accident rate if some thought is given to the subject.

Mr. D. Rees' contribution dealing with English coal mining practice is specially interesting and valuable. He asserts that as systematic timbering has not reduced the accident death rate in England, he fails to see the advantage of it. It must be admitted that the enforcement of rules for systematic timbering has not produced the results expected. The general principle is admirable, but unless it is rationally applied it is useless. Many of the English mine inspectors when investigating accidents due to falls of ground, complain that the distances fixed between the supports are too large, and that it is to this, that the partial failure of systematic timbering is due, rather than to any failure in the system itself. A wide discretion has naturally to be left in the hands of managers, and some have fixed such distances as to render the method of no effect. If Mr. Rees reads his journal a little more carefully he will see that the figures given for the English death rate per million tons were 4·472, not 1·116 as he assumes. While we thank the press for the publicity it has given to this discussion, we cannot hold ourselves responsible for any errors that may appear in the reports.

Mr. Rees' suggestion that shift bosses should examine every working place before the commencement of the shift is new in this country though the rule in England. This idea if enforced would certainly necessitate a large increase in the number of underground officials. Probably on an average a shift boss has 300 native workmen in his section, and as each one has a working place to be examined, beside the long tramming and haulage roads, the task would take up a large part of the day. Under present conditions this official can only just perform his duties in the eight hours of the mining shift. To enable him to perform the work of an English deputy-overman the districts would have to be split up into very small sections. The author in his ten years experience of coal-mining does not remember any section that contained more than 30 working places and some had as few as 16. The adoption of the English rule would mean a heavy increase in costs, and this item alone puts his suggestion out of court.

Mr. B. C. Gullachsen, in his striking contribution, emphasises an aspect of the accident question that is too often neglected. While a knowledge of first aid may not prevent accidents, it will at least lessen the sum of human pain and misery and will in many cases save human life. One of the most painful incidents in the author's life was standing by and after making all sorts of clumsy efforts to stay bleeding, see a native bleed to death from a severed artery in the thigh. One of the most admirable features of the Kotze code is the stress that is laid upon a knowledge of first aid, and the author hopes that this knowledge will eventually be demanded from all persons applying for blasting certificates. Mr. Gullachsen deplores our backwardness in ambulance work and his complaint is well founded. The local officers of the St. John Association do not seem to be too fanatical in their enthusiasm for the object of the Society, but it is understood that this is partly due to lack of funds, a condition that is likely to be overcome in the near future. The author does not know what influenced the Mines Department to decide upon a plank stretcher in preference to the Navy stretcher, but knows that the humidity of the mine air soon causes any form of canvas stretcher to become useless owing to the rotting of the canvas body. After being underground for only three weeks a stretcher has been known to become useless from this cause. Straps for fastening the patient securely are necessary, especially in shafts with long narrow cages where the injured have to be slung in the stretchers.

Dr. Lindsay Johnson's contribution is specially welcome and his surgical advice beyond criticism. His counsel "When in doubt do nothing," is doubtless a hard saying, yet it contains that kernel of wisdom that characterises the whole of his valuable paper.

Mr. E. R. Bawden believes that the "mushroom type" of miner is responsible for our excessive accident rate and probably has some reason on his side, but seems to forget that the demand for old experienced miners is greater than the supply, and so mine managers have either to hang up stamps or engage incompetents. In engaging this latter class, they hope to make good men of them or fill up the place till a good man comes along. Miners are not made in a day, and if no young men are allowed in charge of stopes how is the industry to be carried on?

The endorsement of certificates after an accident is open to strong objection, savouring too much of the eternal punishment of the theologians. If a man is careless and causes an accident, punish him with the utmost rigour of the law, but after he has paid the penalty, justice ought to be satisfied, and the way left open to return to a better state of mind and conduct. It often happens that after a certificate has been endorsed, that the miner appears before a magistrate charged with some offence under the mining regulations and is discharged, but this does not remove the objectionable inscription from his blasting certificate. The endorsement under such circumstances only brings the Mines Department into contempt and creates a feeling of injustice in the mind of the miner, who promptly loses or mislays his certificate, and is furnished with a new one, so that the operation of endorsement is as futile as it is distasteful.

Mr. C. B. Saner assures us that if the manager and his staff have the safety of his workmen at heart, the result will be seen in decreased accidents. In this I agree, for the mining atmosphere is the product of the personality of the mine manager. Every day he is in touch with his staff and if he shows any interest in the safety conditions on his mine, his staff will unconsciously take their tone from him. If he is concerned so will they be, and a safety atmosphere or influence will be created that will be felt in the remotest corners of the mine, and accidents will be few. The author admits without hesitation that mine managers are as humane as any other class or profession, yet he is not blind to the fact that safety measures have been neglected and are still being neglected in a large number of our mines. Take the mines that stand at the top of the accident rate and any person conversant with Rand mining will at once admit that there is nothing in their local conditions to warrant these high rates. Our "safe" mines are not safer than others, because of better roofs, better safety conditions or superior miners, but in the author's opinion owe their happy condition to better managers The Transvaal mines are handicapped by various forms of inefficiency, and that aspect of inefficiency that is represented by a high death rate should not be ignored, for a manager who is carelesss with the lives of his workmen will probably be careless with the dividends of his Company. The time has come when technical skill, however brilliant, industrial talent, however dazzling, must be rejected and passed over if these qualities are accompanied by a callous regard for human life and suffering.

Mr. R. Stokes' somewhat indignant contribution does not deal with the subject matter of the paper, but the author agrees with him that the old subject of practical *versus* theoretical education has been worked to death, and ought to be decently buried. This argument is as old as the hen or the egg story, and is just as futile. The rivals of the different schools are not likely to be convinced, and the controversy always becomes personal.

Mr. R. A. Barry considers that comparisons between coal and metal mine rates are unfair and misleading. Let us then take the metal mining districts of the world where work is pursued under conditions that are similar to those of the Transvaal, *i.e.*, where native workers are superintended by foreign overseers. These districts have the language difficulty, the moving population and the semi-civilised workers that are regarded as the cause of our high accident rate. The rates given have been taken from a list published by Mr. J. K. Leyds, Statistician to the Mines Department. The Transvaal figures are official.

Death rate per 1,000 employees.

	1907.	1908.	1909.
Gold Coast	2·75	2·66	3·71
British Guiana ...	0·26	0·69	0·86
India	1·90	1·50	1·42
Ceylon	0·33	0·24	0·11
Mexico	5·18	4·64	5·78
Cape Colony	2·29	2 35	1·25
Egypt	1·36	?	3·08
Orange Free State ...	2·00	1·56	1·12
Kimberley	3·17	2·19	1·63
Rhodesia	3·96	4·28	4·92
Malay States	1·00	0·74	0·90
Peru	3·10	2·31	2·70
Portuguese E. Africa ..	3·0	2·02	1·03
Transvaal	4·71	4·18	5·29

From the above list it will be seen that when compared with other metal mining districts our position is very discreditable.

The merits or faults of Chinese labour are somewhat beyond the scope of this paper, but Mr. Barry is not correct when he affirms that accidents were less numerous amongst our Chinese labourers, as a glance will prove.

Mr. Barry seems despondent over the "vast field of impossibility" and the "unapproachable fields" of safety, but seems to forget what is being done on other mines. If they can bring down their accident rate to below the average English rate, surely it is not a counsel of perfection to expect all other mines to fall into line. This talk of "safe" mines and "difficult" mines is unconvincing. No mine is safe except it is made so by the vigilance of its mine officials and workmen, and no mine is so difficult that a high accident rate is excusable. The Belgian coal mines are admittedly the most dangerous in the world, yet their accident rate for the three years ending 1909 averaged only 1·01 per thousand. If every mine on the Rand is alike affected by the scarcity of good miners, and all are alike cursed by native labour, how is it that with the same conditions the accident death rate varies from 27·42 down to nil?

The Transvaal Mines Department has been very fortunate in securing the services of men like Mr. R. N. Kotzè and the late Mr. H. Weldon as

Government Mining Engineers. These gentlemen have not hesitated to leave the cold official atmosphere of the Civil Service, and to step down into the arena of our Society, delivering and receiving those blows that always accompany candid criticism. The Society is to be congratulated on receiving this contribution, for Mr. Kotzè has access to information that is denied to the ordinary member, and this matter of mining accidents certainly is worthy of consideration by our higher technical authorities. It is commonly reported in mining circles that Civil Servants are inaccessible to ideas and merely polite, but Mr. Kotzè's contribution effectively dispels this fable. Generally, we find it hard to resist his conclusions, but a few errors may be pointed out.

He starts out by remarking that the author's statement, that our accident rate is increasing, is not correct. The correctness of the Government Mining Engineer's figures may be doubted by Mr. Kotzè, but they have the author's complete confidence and are as follows:—

	Rate per 1000 Employees.	No. of Accidents.
1908	8·54	1,620
1909	9·45	1,883
1910	10·23	2,183

He evidently confuses the ordinary accident rate, with the accident death rate. This distinction is quite clear and is plainly set forth in the official tables.

The author is glad to notice that the death-rate for 1911 is only 3·76, an apparent reduction on our previous figures. Unfortunately this reduction is more apparent than real and has only been brought about by new methods of tabulation, and not by any increase in the safety factor. This new method has been adopted, we are told, for the sake of securing uniformity with other provinces, and also with other countries, and is now "calculated on the basis of persons in the service of companies and not as hitherto on the actual at work figures." Uniformity that is secured by adopting a less correct mode of tabulation is a uniformity that is undesirable, for by it the stream of reform will be checked and the man in the street and mine will be led to believe that matters are getting better while only the recording methods have been altered. The author believes that a correct sequence of figures on the old basis would be more valuable than the new methods, and would show our real position more clearly. Provincial uniformity could have been secured by bringing the 50,260 other colonial employees into line with the 261,635 Transvaalers, but evidently the Mines Department believes in bringing the mountain to Mahomet rather than Mahomet to the mountain. This new classification may be regarded as some evidence of grace in the Mines Department and as showing a desire to stand higher in the estimation of the industrial world, but should a big mining disaster suddenly crystallise public opinion, the new method and its author would both undergo severe criticism. My censor disputes my assertion that the black ribbon of the mining world is held by the Transvaal and gives statistics from 38 countries and districts to show that we are not the greatest sinners. While not disputing the general accuracy of these figures there is some evidence of carelessness and haste in the making of the list. When Mr. Kotzè blunders in the Transvaal figures probably his accuracy does not increase when he wanders away to far off countries like British Columbia and Mexico.

A comparison between his figures and the Transvaal official figures is interesting:—

	Transvaal Official Figures.	Mr. Kotze's
1907	4·71	4·58
1908	4·18	4·03
1909	5·29	5·32

As this difference represents an average loss of 42 lives per annum, it will be seen that it is more than a statistical pedantry.

Perhaps Mr. Kotzè is more accurate than the Government Mining Engineer, but most people will rely upon the official figures, ignoring the Gilbertian situation therein admitted. Of the 38 countries and districts, given in the list, only four, British Columbia, Mexico, Alabama, and West Virginia are our rivals in this unholy competition. The data for British Columbia given by Mr. Kotzè differ widely from those given by Mr. H. Hoffman in the *Engineering and Mining Journal*, December 25th, 1909, and are as follows:—

Accident rates per 1000 employees.

	British Columbia.	Transvaal.
1905	2·72	4·52
1906	3·12	5·55
1907	5·12	4·71
1908	2·95	4·18
1909	9·24	5·29

The Transvaal figures are official.

The abnormal figures for British Columbia for 1909 are due to an explosion at the Extension Coal Mines, October 5th, 1909, where 32 lives were lost. The author considers it unfair to compare small mining districts with very large ones, for small rates when spread over a small number of employees bulk largely in statistical returns. Recently we have had some instances of the mischief wrought by compiling averages ranged over a small number of employees, and Mr. Kotzè has not altogether escaped from the perils that surround the unwary statistician. British Columbia has less than 7,000 mine employees and it is statistically absurd to compare that number with the 261,625 employees of the Transvaal and form averages that are fair to the former.

Mr. Kotzè must have been exceedingly fortunate in securing Mexican accident rates, as that very influential magazine, *The Mexican Mining Journal*, which is produced and printed in Mexico, is unable to obtain them. Dealing with "Accidents in mines" in the issue for April, 1912 it says "No accurate figures can be had as to the number of injuries received in mining operations in Mexico." After such an admission from the leading technical paper of the country, we will pass on to consider Alabama. Alabama during the years under review was visited by a succession of disastrous explosions, so that the figures given do not represent the normal conditions of affairs. Amongst these were the Yoland explosion, December 16th, 1907, where 56 lives were lost, and the explosion at Palos, May 5th, 1910, which resulted in a loss of 84 lives. The latter catastrophe though taking place in 1910 was included in the statistical year 1909. West Virginian statistics suffer from the inclusion of the Monogah explosion, December 6th, 1907, where 350 persons were killed.

After considering the exceptions pointed out by Mr. Kotzè and the local circumstances such as abnormal explosions and doubtful statistics, the author feels that he cannot retreat from the position taken up and considers his charge proved.

Mr. Kotzè objects to my comparison between England and the Transvaal because the tonnage broken per man in a coal mine is greater than that of quartz in a gold mine. His objection would have some solid basis if England's output was exclusively coal and the Transvaal's output was restricted to quartz, but we find that England produces 90 million tons of mineral other than coal and the Transvaal produces 6,000,000 tons of mineral other than quartz. It would be instructive if we find out how low the English accident rate would be if the coal mining alone were considered, and how high the Transvaal rates would tower if calculated on quartzite alone without including coal, when comparing million ton rates.

Mr. Kotzè thinks that differences between deep level and outcrop rates could be better expressed by dividing these into large and small mine rates because the small mines are on the outcrop and the larger ones mainly deep levels. This has led the writer to go into the question with the following results. The figures given have been taken from the Mining Engineer's report on the death rates in mines, 1910.

	No. of Employees.	Accident Death Rate.
Over	10,000	4·32
,,	6,000	4·28
,,	5,000	3·16
,,	4,000	4·64
,,	3,000	3·92
,,	2,000	3·18
,,	1,000	2·56
,,	500	·84
Below	500	1·47

His contention that the best run mines on the Rand have the lowest accident rate is important and is borne out by Table 1, where dividends and low accident rate go hand in hand.

Mr. Kotzè does not favour the idea of education in safety measures by lectures and doubts the success of such a scheme. When Mr. Cullen and Dr. Macaulay gave their lectures on "Explosives" and "Miners' Phthisis" there was little doubt of their success, as miners stood at the open doors unable to find a seat, and in the discussion that followed after being bombarded with questions for 1½ hours, Mr. Cullen pleaded for the indulgence of the 18 hours per day Act, and was reluctantly allowed to go. Under Dr. Macaulay similar conditions existed. While admitting the difficulty of securing men of similar calibre to the gentlemen mentioned, the author still thinks that good could be done by this form of education. These lectures could be made bright and interesting and not too offensively didactic, and if the mine manager displayed some interest could be made a success. Mr. H. Briggs in his paper on "Mine Accidents," reported in the *Iron and Coal Trades Review*, February 9th, 1912, says "Some years ago the percentage of fatal accidents in the Fife pits was abnormally high. An instructor was then appointed, an experienced and highly intelligent man thoroughly familiar with the rules, whose duty it was to lecture to the workmen in the evenings, on the meaning of the rules and the best way of carrying them out, and he went into the pits and called attention of both manager and miners alike to any appearance of infringement of the regulations. Although it is hardly possible to say how far the result is due to the instructor's work, it is nevertheless true that the accident rate has decreased since his appointment, and from the fact that the instructor's services are still being continued we are able to gather that the mining companies do not feel themselves deceived in the policy."

The writer cannot agree with Mr. Kotzè's contention that owing to the condition of weight and pressure met with, systematic timbering would be useless in this district. Roof weights and pressures here operate as they do in other countries; there comes a time in the history of almost every mine when roof pressure cannot be controlled and collapse takes place, but before that time arrives systematic timbering can put off the evil day, for collapse can only take place after the rupture of the roof beds. Systematic timbering will not restrain the movement of large areas, but it will prevent this in its early stages. As a form of efficiency, this method is entitled to consideration, for if used we should not have consulting engineers lamenting the collapse of the richest stopes in the mine, when asked to account for a diminished gold output, as happened a few months ago.

The author regrets that he cannot agree with Mr. Kotzè on many questions of method, conditions and remedies, but he certainly does agree with his summing up of the whole matter, viz., "that efficiency and safety on a mine march together."

THE "LUMINATOR" PURIFICATION TREATMENT OF WATER FOR STEAM BOILERS.

(*Read at March Meeting, 1912.*)

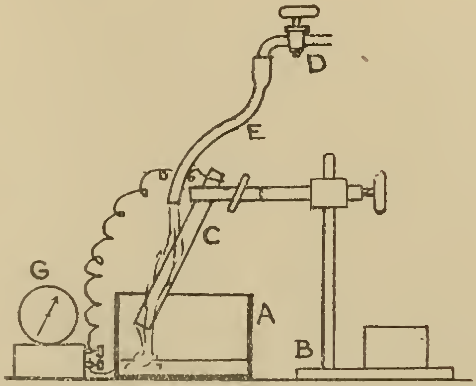
By William Cullen (Past-President).

DISCUSSION.

Mr. W. Alexander, A.M.Inst.C.E. (Contributed): Mr. Cullen's paper is very interesting for, since seeing the "Luminator" in use two years ago in the works of Messrs. Bell's United Asbestos Company at Greenwich and elsewhere, its action has repeatedly recurred to me as requiring explanation. At the time, Mr. Bell's statement—that the good results he was getting were due to the Luminator—was accepted with much doubt, which was increased when the work's chemist, Mr. Meldrum, explained that there was no chemical change in the water when it had been passed over the luminator. Afterwards, when it was known that large railway companies and prominent engineers were using the device, and a leader appeared in *Engineering* for February 18, 1910, with a subsequent correspondence, doubt was no longer reasonable.

In order to test the electrical theory of the action, experiments have been made to determine whether there is any electric current set up by the luminator when working. Mr. Murray kindly collaborated in the experiments, which were performed in the Metallurgical Laboratory of the School of Mines and Technology.

Apparatus, diagramatically shown here, was arranged to represent as nearly as possible the actual electrical conditions obtaining, outside of the boiler.



In the diagram, A is an iron vessel representing the boiler, the piping, the feed pumps, and the feed tank, all of which would be at the same electrical potential. On a wooden stand B is a bar of bright aluminium C which represents the luminator. A water tap D supplies water (alkaline) from the mains through an india-rubber pipe E. The length of this pipe was large compared with the distance between the aluminium and the iron vessel.

Experiment 1. A galvanometer G was connected up as shown to the iron vessel and to the aluminium plate. When the water was turned on a deflection of about 1 milli-volt was obtained. The deflection increased more and more the nearer the water in vessel A approached the lower end of the aluminium, and the maximum deflection, 4 milli-volts, was reached when the water eventually rose to touch the bar.

Experiment 2. The galvanometer was connected up to the vessel and the water tap, when there was no deflection at all.

Experiment 3. This was done to ascertain if there was any polarisation with the aluminium suspended in quiescent water, the galvanometer being connected to A and C, and the tap closed. It was found that the current decreased with time. On rubbing the immersed portion of the bar the current increased. This points to the fact of there being something deposited which offers electrical resistance, and Mr. Murray suggests the probability of the substance being aluminium hydroxide, $Al_2(HO)_6$, aluminium being electropositive to iron.

Experiment 4. This was done to ascertain the effect of light. Experiment 1 was tried in strong and weak light. No difference in the results was observable.

Other experiments were made, with water made more strongly alkaline, and with acidulated water, giving the same sort of results magnified.

Now, supposing the subsequent action which prevents the deposition of hard scale is dependent on an initial electrical action, we arrive at these conclusions: (a) Aluminium is used because it is electrolytically active with both alkalies and acids (zinc is known to be active with acid but inactive with alkalies) and it is known to be better than other metals in this double capacity. (b) The plate is grooved in the direction of flow (not as stated in the paper at right angles to the flow) and placed at a steep angle to give the water rapid motion in order to scour off any deposit of material on the metal and so keep down electrical resistance, and sweep the material, which may be necessary for catalytic action, into the feed tank. The material is partly a product of the electrolytic action, and part of it probably comes from the suspended matter in the initial water. (c) To get the best results the electrical resistance should be kept as low as possible; therefore, the plate should dip a little in the water in the feed tank and should be placed near the side; also, good electrical connection in the external circuit should be provided between the plate and the feed tank. The reason for the periodical burnishing is apparent. (d) "Treated water," before it can be considered such, and suitable for any experiment of boiling in glass or porcelain vessels, should be obtained in some manner which provides, as in the experiments here described, that the circuit is complete during treatment in order to allow the proper electrical action to take place freely. That is, the receiving vessel must be iron for one thing, this being the usual metal of a feed tank.

A question that suggests itself is this: would the actions that occur in the water after it leaves the luminator be the same whether the water be kept running or not? That is to say: is the necessary electrolytic change in the water complete after passing over the aluminium, and are the later changes therefore purely chemical; or, do the final actions require that the electric circuit through the plate be maintained by keeping the water running? There is plenty of scope for a research to determine whether there is a catalytic action of the electrolytic product on the constituents of the water under the high pressures and temperatures existing in a boiler. The solution of these problems would seem to be congenial work for the chemists.

Mr. F. J. Pooler, B.Sc. (*Associate*): The author states that no one has so far as he can gather, succeeded in proving a chemical change in the launders conveying feed-water to the boilers. Is the reason of this, perhaps, that working on the large scale sufficient care has not been taken to make chemical analyses of sufficient accuracy? May it not, therefore, be of value, if "the dirt" on the launder is analysed after a long run? However slight any effect is, it may possibly be cumulative and in time sufficient change may have taken place to give an appreciable effect on analysis.

As I had the good fortune to be present when Messrs. Alexander and Murray were conducting their tests to show that an electric current is obtained when water flows over an aluminium rod into an iron vessel, it seemed that some light might be thrown on the action if the effects of (1) aluminium, (2) iron, and (3) aluminium and iron, on hot and cold distilled and tap water were studied. I believe that there is in operation a commercial process for producing hydrogen by use of an aluminium-mercury "couple" obtained by immersing Al. in solution of mercury chloride, when in presence of a catalyst such as potassium sulphocyanide (I think) enormous quantities of gas are obtained. Now iron is much nearer than mercury to aluminium in the electro-motive series of metals*, but as aluminium is electro-positive to iron it should be possible to form an Al-Fe "couple," which gives hydrogen with hot water. It is surprising how little information even recent text-books give on matters of this kind. It is, of course, well known that aluminium slowly decomposes boiling water, hydrogen being evolved. Perkin and Kipping† state that this is a property of "the finely divided metal." Using fairly rough filings and some quite large scrapings from a sheet of foil about $\frac{1}{8}$ in. thick, I have collected hydrogen from local tap water (alkaline) with little difficulty. Samples of distilled and tap water were boiled for half an-hour or so, to expel air, then the metal introduced. This caused a little more air to be boiled out, but adding iron (steel) filings to the aluminium caused a fairly rapid evolution of hydrogen as is shown below. Chemically equivalent quantities of the two metals were first used, but it is obvious that quite a small quantity of the iron is required if sufficiently close contact can be obtained between the two metals.

A summary of the few simple experiments showing the influence of the iron on the rate of hydrogen evolution is given:—

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*See Alex. Smith "Inorganic Chemistry" (1910), 411, p. 670.

†Perkin and Kipping "Inorganic Chemistry" (1911), p. 612.

TABLE A.

Metal.	Time.	Vol. of Gas.	Nature of Gas.
Al.	5 min.	2 cc.	Air (?)
Al. & Fe.	5 ,,	14 cc.	H_2 and air in slightly explosive mixture.
•	30 ,,	14 cc.	

Here boiled distilled water was used.

TABLE B.

With Boiled Tap Water (Alkaline).

Metal.	Time.	Vol. of Gas	Nature of Gas.
Al.	5 min.	10 cc.	Fairly pure Hydrogen.
	18 ,,	26 cc.	
	30 ,,	29 cc.	
Fe.	5 ,,	4 cc.	Air (?)
Al. & Fe.	4½ ,,	26 cc.	Almost pure hydrogen as far as burning quietly showed.

The results show that the action of Al. alone is in some way dependent on the "hardness" of the water, being increased thereby and also that Fe increases the rate at which the gas is evolved. The last experiment, Table B., could not be continued beyond 4½ min. because a violent explosion shattered the flask. This was due, I think, to the finer particles of the filings clogging up the delivery tube, for air had been excluded by filling the flask up to the cork with boiled water.

No reason is assigned for the rapid decrease in the rate of supply of gas after the first few minutes using Al. and tap water, but some inhibiting substance may have coated the metal, or merely the action have been decreased as the concentration of one of the products increased. A somewhat similar result is shown using the "couple" and distilled water, but I am inclined to think this is due to air being first boiled out of the water on adding the iron filings. These decreases are worth noting, in view of the fall in E.M.F. noted by Mr. Murray.

A fresh lot of tap water (300 cc.) was then boiled with the "couple" in an open flask for some hours, well shaken to get out gas bubbles and allowed to stand overnight. In the morning a little more hydrogen was obtained. This was repeated at intervals for two weeks. Hence this "couple" action seems able to go on slowly in the cold. I shall be glad if some one will try and confirm this action.

The liquid from this action was then re-boiled and filtered. The filtrate showed no iron, but after a little trouble traces of aluminium were detected after treating with dil. HCl. This may be due to the formation of "colloidal" aluminium hydroxide, as Mr. Murray has suggested. Such a solution‡ may be obtained when freshly prepared $Al_2(OH)_6$ "is dissolved in aqueous solution of aluminium chloride," or as suggested by Fenton§. "Aluminium hydroxide dissolves to some extent in water forming a colloidal solution (hydrosol) which is changed by salts (e.g. NH_4Cl) to the insoluble form (hydrogel)." The NH_4Cl was of course in solution after the HCl solution was treated with ammonium hydroxide.

Supposing however that $Al_2(OH)_6$ was formed in such small quantity as to be in solution in the usual way. This with the alkali in the tap water would form sodium aluminate (say). Then calcium and magnesium salts§§ in solution might be precipitated e.g. as calcium met-aluminate, thus:—

$$2Na(AlO_2) + CaCl_2 \rightarrow \underset{\downarrow}{Ca(AlO_2)_2} + 2NaCl.$$

In this case the Al. would be found with the unchanged metals on the filter paper. I have no means of determining whether this is so or not, and while not offering this as a reason for the indefinite results obtained so far, suggest it as a reason for examining "the scrape" from the launders after being in use some time.

Again, the suggestion that colloidal aluminium itself is obtained requires further consideration. It is well known that colloidal metals such as gold, silver and platinum have different colours from the metals as ordinarily found, also they possess great activity as catalysts. May it not, in this case, be due to the actinic properties of ultra-violet light, since daylight seems necessary for the successful carrying out of the operations? This presents an opportunity for research by physicists. Perhaps the ultra-microscope would reveal the existence of colloidal aluminium. So far as I know, such has not yet been found. I have to record an unsuccessful attempt to obtain any change in weight when tap water flowed over an aluminium plate, for as many hours on several days as the school Kafir could be prevented from turning off the tap. Is it not, then, worth while taking a small weight of aluminium filings and running water over them for a long time and carefully examining the filings. The balances at my command offer no hope of such accuracy as this requires Why not try the action in direct sunlight and in ultra-violet rays, if such is possible?

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‡Perkin and Kipping (1911), p. 614 "Inorganic Chemistry."

§Fenton "Qualitative Analysis (1910), p. 69.

§§See Alexander Smith "Inorganic Chemistry" (1910), pp. 684-5.

It will be observed that no claim is here made to have solved the mystery. A few tentative suggestions in the light of some modern work are offered. My experiments have merely dealt with that phase of the matter which was suggested by Mr. Alexander's idea that the aluminium launder and the iron boiler should be in electrical contact. So far as they showed, little change could be detected in the aluminium, except for the deposition on some of the larger pieces of a yellow stain—the nature of which could not be determined. This may have been iron rust, but in one case, at least, it formed so quickly as to render this improbable.

In conclusion I would like to thank the author for bringing forward this subject which offers opportunities for such interesting investigation and research.

Mr. M. T. Murray (*Member*): At Mr. Alexander's suggestion, a few simple experiments were carried out, with a view to the possible further elucidation of the phenomena underlying the "Luminator" process, as dealt with in Mr. Cullen's paper.

As Mr. Alexander has remarked, we wished to discover whether any electro-motive force was set up between the aluminium plate and the feed tank, or (since feed tank and boiler were probably in electrical contact) between the aluminium plate, or launder, and the boiler itself. As he has stated (and I think this little experiment will tend to support his statement) we think that this electro-motive force almost certainly exists.

Mr. Alexander's theory having been substantiated, it seemed that a further investigation on these lines might be advisable. To begin with, we wished to discover what particular virtue aluminium possessed, which other metals did not. It is electro-positive to iron, of course, a position which is shared by only one other commercially practicable metal, viz.: zinc. When the aluminium plate was replaced by one of zinc, it was found that with acid water, the e.m.f. was of a similar magnitude to that produced by the aluminium, but that with alkaline water, practically no e.m.f. was produced, while aluminium gave almost equal e.m.f.'s with acid and with alkaline water. This is probably due to the greater readiness of formation of alkaline aluminates to zincates.

In order to discover whether the action was merely due to mechanical effect—as has been suggested in one of the author's quotations—distilled water was substituted for the ordinary Rand water, which is slightly alkaline. Practically no e.m.f. was produced in this case. Again, since sunlight is supposed to be an aid to the luminator process, the experiments, which had first been carried out in the basement, were repeated on the roof of the School of Mines building; no appreciable increase in the e.m.f. produced was noted under the latter conditions. Whether the strong sunlight has any action on the colloids formed, I am not prepared to say.

These few rough experiments, performed in collaboration with Mr. Alexander, were sufficient to arouse interest in the question, so that they were repeated more carefully, without, I believe, wholly negative results.

It was argued that if an e.m.f. was set up, some chemical action must be taking place, and (since the aluminium is electro-positive to the iron) that the most probable action would be the transformation of the aluminium into aluminium hydroxide, which, in an alkaline water, might pass into solution as an aluminate, and this aluminate might form (*vide* Mr. Pooler's contribution) calcium met-aluminate. It seems much more probable that any colloidal nuclei in the water treated by the luminator process, are either aluminium hydroxide, or calcium met-aluminate, than that they are actually "colloidal aluminium"—of the existence of which we have no experimental evidence.

Obviously, if any chemical change occurred, we should expect the aluminium to lose or gain weight (most probably the former). A rough imitation of the luminator process was therefore set up, using a very thin aluminium plate, which exposed some 12 square inches, and weighed only about 6 grammes. This was weighed carefully and exposed to the action of flowing water (the plate being in electrical contact with an iron vessel) for 115 hours. At the end of this time there was no loss in weight—in fact there was a small *gain* (about 0·04%) which might have been due to deposited impurity from the water, or to the production of aluminium hydroxide which the stream of water had not been strong enough to wash from the plate. The plate was not brushed or rubbed during the experiment.

This somewhat disappointing result pointed to the probability that the e.m.f. was not constant, or of one kind. A recording galvanometer was therefore placed in circuit with the plate and iron vessel, and the e.m.f. recorded at half-minute intervals. It was found that the e.m.f. (which was about 1 milli-volt) fell away rapidly to zero, and then that the galvanometer needle passed through the zero, and registered a *back* e.m.f. of about equal magnitude to the first one. This again fell away very slowly to zero, and remained there. It might have been thought that this reversal was due to a simple polarisation effect—*i.e.*, to the collection of bubbles of hydrogen on the negative pole (the iron) which were electro-positive to the aluminium. This theory was disproved, however, by the fact that the original, positive e.m.f. could be at once restored by gently rubbing the lip of the aluminium plate, over which the water was falling, without disturbing the iron, or the water in the iron vessel, at all. When the apparatus was again left, a similar reversal of poles again occurred. I believe that the little experiment which I have set up here will illustrate what takes place.

The only feasible explanation of this phenomenon seems to me to be as follows:—The aluminium is transformed into aluminium hydroxide, which (unless the water current is exceedingly rapid and strong) adheres to the plate, and causes, first a falling off of the current, due to the great resistance of the substance, and, secondly a reversal of the direction of the current, due to the electro-positiveness of the iron to the aluminium hydroxide. It is possible that this reverse e.m.f. continues, until conditions of equilibrium are established, and that no further action takes place after that, until the layer of aluminium hydroxide is removed by brushing. It might, perhaps, be too much to assert that there is no action in the luminator process, other than that taking place for the few moments after brushing, but the experiments seem to point to some such conclusion. Under these conditions the quantity of aluminium passing into the boiler, in so brief a period of time, would be so small as to be almost imperceptible, on weighing the large sheet. But it must be borne in mind that the aluminium—in whatever form it exists—would (at least between "blow downs") concentrate in the boiler, so that its effect would become to a certain extent cumulative.

The foregoing experiments, it is true, have not gone much farther in the elucidation of the problem of the luminator process, in that (in common with most boiler problems) one is at once met by the difficulty of reproducing in laboratory experiments the conditions of temperature, pressure, etc., which obtain in practice, and without these it would be foolish to dogmatise, as a result of those experiments.

However, those few which have been done have, I think, thrown a little further light on the subject, and have at least opened up fresh possibilities I would venture to suggest, for instance, that the experiment of brushing or burnishing the aluminium plate *continuously* be tried, or that the effect of the introduction of freshly-precipitated aluminium hydroxide into the feed water, e investigated. Again, bringing into the circuit small external current of electricity, making he aluminium plate the anode, might have a eneficial effect on the process, by causing the luminium to pass more quickly into the boiler. These experiments are only a few of the many which suggest themselves as worthy of trial, in further investigation of this mysterious phenomenon.

I have to apologise to the author for offering to the discussion a contribution which cannot be classed as criticism, but excuse is to be found in the author's hope that his paper would lead to discussion. I wish to thank him too, for a paper which has provided much food for thought, and a field for experiment which has already become larger than the Luminator process, and threatens to become too wide for an investigator, hampered by considerations of bread and butter, successfully to cover.

NOTES ON ABSORPTION OF GOLD BY AMALGAMATED COPPER PLATES, CAST IRON RETORTS AND AMALGAM TRAYS.

(*Read at December Meeting, 1911.*)

By Prof. G. H. Stanley, A.R.S.M., F.I.C. and
M. Thornton Murray, M.Sc.

REPLY TO DISCUSSION.

Prof: G. H. Stanley (*Member of Council*): The authors desire in the first place to express their gratification at the appreciation expressed by contributors to the discussion and their thanks for the valuable information elicited thereby.

Mr. Dowling's figures from actual practice, besides confirming the authors' conclusions, testify to the efficiency of the method of scaling which he describes and to the care with which the operation was performed. However, as the plates instanced had only been in use 2⅓ years the surfaces were probably but little worn or irregular, a circumstance which in the case of older plates causes more difficulty in the complete detachment of gold and has probably given rise to the impression that gold had actually been absorbed in order to account for the gold shown by assay or found on other treatment of the plate for its recovery.

The figures given show that 12·64 oz. per plate were recovered by scaling, and as the authors estimated that the plate they examined carried about 45·75 oz. after sweating, this would tend to show that the hard surface scale which is not removed by sweating in the ordinary way and not always completely even by scaling grows gradually thicker during the life of the plate.

Another explanation which might be put forward is that as the plates were originally set with silver amalgam, a larger proportion of the scale might consist of silver, thus reducing the amount of gold: this, however, is not stated, and considering that plates become worn and thin in use, it would appear that the original silver amalgam scale must become worn off and replaced by gold. Still, the use of silver amalgam would undoubtedly decrease the amount of gold taken up or "fixed" on the plate in its early life.

Mr. Laschinger's comment on the facility with which mercury penetrates the most minute holes also helps to confirm the suggested explanation as to the method by which the gold, which undoubtedly exists in the interior of plates, arrived there, and this assumption, that it enters by minute holes, also has a bearing on the possible amount present.

If the gold enters only in solution in the mercury, its amount is obviously limited by the solubility of gold in mercury and the actual amount of mercury absorbed.

On the other hand it is conceivable that some of the gold (or particles of amalgam) in suspension may be so extremely fine as to be carried in still in suspension in the mercury, and the amount of gold "absorbed" in the working of different ores in different localities, vary to some extent therefore with the fineness of the state of division of the gold.

In the absence of more definite knowledge as to the mode of existence of gold in liquid amalgam this point cannot be pursued further at present, but the authors have it on their list for the future.

In the case examined, the ratio $\frac{\text{gold}}{\text{mercury}}$ in the mercury coating the bottom of the plate was determined at ·00105. Now the solubility of gold in mercury has been determined at about ·12% at ordinary temperatures, corresponding to a ratio ·0012 which is sufficiently near to indicate that all the gold was in solution and probably carried thus through the plate by the mercury.

With regard to retorts, they suggest that it would be an improvement to have a removable liner of say $\frac{1}{8}''$ sheet metal inside the cast iron, which should be slightly enlarged for the purpose. This would certainly take up less gold and in addition would greatly simplify the extraction of gold from it subsequently.

Naturally, the distortion of the retort during use would not render the removal of the lining easy, but the objection does not seem to be insuperable, and the suggestion may be worth the attention of the mines' trials committee.

They would further suggest white iron as a better material than grey iron for the manufacture of retorts on account of the absence of graphite: but graphite may appear in white iron as a result of a long series of heatings and cooling.

Replying to Mr. White, the authors did not intend to convey the impression that they did not accept the fact of gold being to some extent locked up in a reduction plant, but rather to ascertain whether the explanation afforded by the assumption that gold was really absorbed was a correct one or not.

His enumeration of the many ways in which gold is locked up in the reduction plant is a much more satisfactory explanation of the usual initial shortage and demonstrates how really unavoidable this is.

On the other hand, a want of appreciation of these often results in a lot of gold being left in disused and discarded plant or portions of it which is then easily recoverable, although it may have been inaccessible during the life of the plant

The authors have known instances of old launders flung aside as useless for anything but firewood, carrying scale assaying 20 to 200 ounces per ton, of which several hundred pounds can be recovered about most plants.

Turning to retorts: *suitable* trays will possibly prevent any gold coming into contact with the retort itself. But there is always the liability to accidental spilling over in some way and perhaps even to projection through formation of steam from entangled water, and besides this, the trays examined were so saturated with gold that most probably gold had escaped through them and on to the retort: an accidental crack would always cause this, and with the usual cast iron trays it would seem to be impossible to prevent the retort carrying more than a little gold at the end of its service. The difficulty of dealing with the retort certainly lessens its value to the by-product smelter, but it should be worth treating. Perhaps stamped sheet metal trays, which would certainly absorb less gold, would in this connection justify their increased cost.

The authors confess themselves rather hopeless as regards a remedy for absorption of lead by furnace bottoms. The amount of gold also absorbed is of course decreased by melting down in them barren lead at first, but, within limits, the bottom probably absorbs more lead at each successive cooling and reheating owing to expansion and contraction, etc., and apart from the first time, the absorbed lead carries gold.

The suggested analogy between oiled blotting paper and water, and a brick bottom saturated with some liquid and lead, is not quite a parallel, as in the latter case we have small cracks, comparable to pin holes in the blotting paper, which would let water pass; and a much greater difference in specific gravity involved. Individual bricks can be made less absorbent, like crucibles, by dipping in a wash of lime and borax, drying and firing, and this would probably close the very minute cracks referred to.

With regard to the actual amount of gold in old retorts, Mr. Watson's figures are very much to the point, though naturally it is impossible to obtain accurate samples from such material. But as the whole retorts assayed say 46 oz. per ton and the turnings to a depth of $\frac{1}{4}$ in. 190 oz., this indicates that only somewhere in the neighbourhood of 50% of the gold was contained in that $\frac{1}{4}$ in. (assuming its thickness to be about 2 in.) and that possibly owing to the long while it had been in use, and to working without trays, the the gold had penetrated to a greater depth than in the specimen examined by the authors. Gold having been found on a bearer bar would probably point to a definite crack or flaw. His figures also confirm Mr. White's statement as to the generality of retorts being much poorer than the piece dealt with in the paper. Smelting with lead in the Tavener furnace would appear to be the best method of dealing with the turnings, but if this is impossible we suggest acid treatment on the lines of, or along with, zinc box precipitate.

The method of dry scraping advocated by Mr. G. A. Robertson should certainly leave less gold on the plates and result in less "lock-up" of gold, but on the other hand the plate would probably not be very efficient at first when again started. If this were carried out in regular practice there should not be the accumulation of hard amalgam scale found at present, but if it were to be used to remove this scale when formed an excessive amount of labour would be necessary.

However, if it has been tried and condemned on several mines, there may be other circumstances in connection with it which militate against its use in regular practice.

Mr. Whitby, as usual, has something of interest to contribute and shows with what facility gold adheres to iron surfaces and becomes "locked up" there.

The authors are afraid, however, that he has misread or misunderstood them when he says: "The authors have endeavoured to prove that old retorts do not show gold on the outside"

They did actually find gold on the exterior, but were not certain as to how it arrived there, *i.e.*, whether by diffusion or percolation through the "solid" metal, or through pronounced cracks or flaws: probably both methods contribute to its presence.

Mr. Whitby kindly allowed his specimen to be sectioned for the microscope and it is on exhibition this evening. It is much thinner than the previous specimen and gold can be seen throughout the section.

In conclusion, the authors would again thank those members who contributed to the discussion and brought forward facts and figures of utility to gold metallurgists in general.

ZINC DUST TESTS.

(*Read at February Meeting, 1912.*)

By W. J. Sharwood, Ph. D., A.R.S.M., M.I.M.M.
(*Corresponding Member of Council*).

DISCUSSION.

Mr. M. T. Murray (*Member*): At a time when precipitation by means of zinc dust is being seriously considered, the author's notes are peculiarly opportune.

The paper does not lend itself to lengthy criticism, in that it is largely a summary of data, culled from various sources, together with the results of experimental work performed by the author.

As the author points out, the method given for the determination of the precipitating effect of zinc dust, is open to criticism, but it is difficult to see how a much better one could be devised. I would suggest, however, that the "occasional stirring" should be replaced by continuous agitation by means of a motor-driven stirrer, and that (at least for our local conditions) a gold solution —preferably a portion of that already in circulation on the mine where the zinc dust is to be used, and containing the usual impurities—should be substituted for the silver solution.

Elutriation is, of course, a useful method of classifying the material, but the author's sketches of the appearance of the particles, under the microscope, seem to furnish more useful and interesting data with regard to the form and size of the zinc dust particles.

The author remarks upon the bright appearance of the surface of the globules—a point to which I drew attention in my brief note on zinc dust, in the October *Journal*.

Dr. Sharwood, in his addendum, objects to my methods of calculating the mean diameter of the particles of the dust. His objection is, of course, perfectly valid, and, did the percentage of the coarser material become seriously large, the method would undoubtedly lead to inaccurate results. In this case, however, the percentage of coarse is so low that calculation on these lines is feasible. I would point out, too, that my result differs from the author's by less than 1%—a scarcely noticeable difference in a calculation based (in the author's own words) "almost entirely upon the average diameter assumed for material passing the finest sieve used, which we have no means of accurately determining.”

Indeed, as will have been noted, the author’s estimate for the average diameter of the “ – 200” particles, is 0·0005″, while mine was 0·00085″—a difference of 0·00035″, or nearly 70 % greater.

My figure was obtained by different, independent observers, each measuring a large number of the particles of the finest grade, under a microscope provided with a carefully-calibrated measurement grating, in the eye-piece, and averaging the results, taking into consideration the apparent proportion that the amount of each “size” bore to the whole. The sample which I used was, as has been noted, representative of material which was in practical use here, and was, as far as I am aware, quite an average one. In view of this fact, and of the care taken to eliminate, as far as possible, the personal error, I am reluctant to admit that the average diameter which I assumed was either due to abnormality of sample, or to inaccuracy of measurement, and I would submit that it was scarcely possible for the author, from his examination of different samples from mine—probably of entirely different origin—reasonably to question my estimate.

There are, however, many points of interest in the paper. I am surprised to find that zinc dust, when exposed in the open (as the material in the Montana freight car was) oxidised so rapidly. The author’s experience should be a warning to metallurgists using zinc dust on the Rand.

A point upon which the author might, at some future date give us further information, is the effect of lead on the precipitating power of zinc dust.

It would seem (from theoretical considerations) that the presence of lead in quantities up to (say) 1·8 % in the zinc-lead alloys tested would have only a small effect on the precipitating action of the zinc, since zinc, on solidification, generally retains up to this percentage of lead in solution. Above this percentage, the lead can be discerned *separated* from the zinc, under the microscope; under the latter condition it should certainly be more active.

Has the author tried the effect of the addition of a little lead acetate solution to the zinc dust? The spongy lead precipitated under these conditions, is generally more electro-chemically active, than when present in the alloy itself.

Zinc dust shows every indication of becoming more prominent in the metallurgical practice on the Rand, in the near future, and we are deeply indebted to the author for familiarising us with its properties, and with the methods in use in other parts of the world (where it is by no means a stranger) for analysing it, and testing its precipitating capabilities.

There being no further business, the meeting was declared closed.

Annual Dinner of the Society.

The Annual Dinner of the Society was held at the Grand National Hotel on Saturday, May 11th, Mr. W. R. Dowling (Vice-President) in the chair. There were also present:—

Messrs. W. Abel, Alex. Aiken, Sir Geo. Albu, Bart., R. A. Alston, R. G. Bevington (Past-President), F. L Bosqui, J. Frank Brown, M P.C., Dr. W. A. Caldecott (Past-President), W. Calder, W. C. Carbis, J. Chilton, M. H. Coombe, J. S. Cotterill, W. M. Coulter, Andrew F. Crosse (Past-President), W. Cullen (Past-President), G. A. Darling, Fred. H. Davis (President, S. A. Inst. of Engineers), C L. Dewar, J. M. Dixon, W. R. Eales, W. Easterbrook (President, Pharmaceutical Society of the Transvaal), *Evening Chronicle*, Ed. Farrar, J. F. Ferguson, G. Goodwin, Dr. Grant, Jas. Gray, B. C. Gullachsen, Dr. W. A. Hendryx, Alex. Heymann, C. B. Hilliard, I. Hodges, H. J. Hofmeyr (President, Rand Pioneers), H. L. Hoon, W. Ingham, J. Irvine, W. H. Jane, Edwd. H. Johnson (Past President), J. Dale Lace, A. E. Lamb, E. J. Laschinger (Vice-President), G. A. Lawson, Hy. Lea, James Lea, L. T. Leyson, James Littlejohn (Hon. Treasurer), A. N. Locke, G. J. Mackay, H. S. Mackay, T. A. Malpage, E. L. Matthews, W. McNeal, J. N. Meeser, H. Meyer, L. D. Mills, J. T. Mitchell, P. T. Morrisby, John Munro (Vice-President, Transvaal Chamber of Mines), S. Newton, A. Mackie Niven, Prof. John Orr, G. S. Paterson, Dr. Geo. Perkins, F. D. Phillips, J. Moyle Phillips, R. G. Campbell Pitt, R. A. Porter, J. F. Pyles, *Rand Daily Mail*, A. Richardson, J. H. Rider (President, S. A. Institute of Electrical Engineers), Fred. Rowland (Secretary), C. O. Schmitt, B. C. Travers-Solly, *S A. Mining Journal*, H. Stadler, Prof. G. H. Stanley, *Star*, A. N. Stuart, W. A. C. Tayler, A. Thomas, A. M. Thomas, D. J. Thomas, Jas. E. Thomas, Prof. G. R. Thompson (Principal, S.A. School of Mines), Dr. Ritchie Thomson. J. R. Thurlow, C. Toombs, M. Torrente, *Transvaal Leader*, F. J. Trump, Dr. G. A. Turner (President, Witwatersrand Branch of British Medical Association), A. Wagner, A. Wallace, R. C. Warriner (President, Association of Mine Managers), F. Wartenweiler, W. Waters, F. A. Westfield, W. S. Whaley, H. A. White (Vice-President), J. Whitehouse, John R. Williams (Past-President).

The Secretary read expressions of regret for inability to be present from:

H.E. the Governor-General, Lord de Villiers, Sir Reginald Hart, Generals Botha, Smuts and Hertzog, Sir David Graaff, Messrs. H. Burton, A. Fischer, H. C Hull, Col. Leuchars, F. S. Malan, J. W. Sauer, His Honour the Administrator, Sir Percy FitzPatrick, Sir George Farrar, Sir Lionel Phillips, Senator S. Marks, Messrs. F. D. P. Chaplin, D. Christopherson, J. Cowie, W. H. Dawe, P. Duncan, C.M.G., Max Elkan, P. Ross Frames, J. G. Hamilton, M.V.O., R. N. Kotzè, and R. W. Schumacher, and the Mayor of Johannesburg.

After the loyal toasts had been honoured,

THE SOCIETY.

Mr. A. Wagner, in proposing the toast of the Chemical, Metallurgical and Mining Society of South Africa, said: Mr. President and Gentlemen, while of course I duly appreciate the honour conferred upon me of proposing the toast of the Society, I must admire the courage displayed by the Council in choosing me to undertake that task. In former years you have had the advantage of great politicians, great scientists, Presidents of the Chamber of Mines, and others with handles to their names, to come forward and pat you on the back. To night you are satisfied with a man who lacks all these qualifications ("Question") and is not even connected with that part of the mining industry which interests you most. I approach the subject with a certain amount of fear as if there is any profession which gives me sleepless nights it is the chemical profession; I may wake up one fine morning to discover that they have discovered the way to make artificial diamonds and thus finish me—(Laughter)—but I believe that that part of the science refers mostly to alchemists rather than chemists.

However easy it might be for me to make a good speech in the good old days when these functions took place about three times a week, when men were not particular whether their allusions were good or strong and when everybody was ready to make an after dinner speech, when I see in front of me the expert shorthand writer and when I imagine the *Sunday Post* repeating to-morrow morning what I have said to-night after an excellent dinner, then I feel I have to be cautious indeed, and this toast being so very high up in the list makes it increasingly difficult to talk as one would like to.

The modesty of your Chairman in reference to the work of the Society during the past year deprives me of material with which to crack you up as much as I should like. When the Rand Central and Brakpan works were opened in 1896 by President Kruger, the Chairman of the evening was not so modest. After inviting the President to make a speech he himself spoke so fully about the undertaking, praising its merits and describing it to the full, that when the President was called upon to give the toast of the electrical works he said he was sorry the Chairman had said so much about the undertaking that there was nothing left for him to say. Had your Chairman done the same he would have given me a wider field, as I believe if I excel in anything it is in opposition (laughter) and I would have got over the difficulty of making a speech very much more easily than I do now.

Joking aside, it really gives me great pleasure to propose the toast, because if there is one Society in Johannesburg that always appealed to the onlooker, and to the man who has not had the advantage of a technical training or of higher education, it is your Society.

During the 25 years that I have been on these fields I have had occasion to follow that excellent publication of yours and have seen with what great zeal and painstaking care you go into the merits of every question affecting mining, and what appeals perhaps to the man who is more concerned with the finances of the industry than with the actual working of it is this, that a good deal of your work is really ideal, and, in a place where so often we are reproached that we only look to the practical side of any work, it is indeed elevating to know that there are men who are ready to devote their time to experiments and to discoveries even if those experiments and discoveries do not immediately benefit their own companies and in fact rarely affect their own pockets. Often in going through these discussions of yours and reading your papers I have deeply regretted that I was lacking in the higher education necessary to fully appreciate them, and I am reminded of the young man who, when in after life he was confronted with some difficult scientific problem and attempted to find the solution, exclaimed "It serves my father right. Why did he not give me a better education?"

This Society of yours is an old institution and serves a very useful purpose in this country. Some of you, like my friend Mr. Lace, will remember the chaotic circumstances under which these fields started. When one compares the panning of 1888 with the exact assay of to-day, worked out to the fourth decimal—at any rate on paper—then one marvels at the progress the industry has made in the short period of one generation.

When assayers first took the field people were almost inclined to say "Leave me alone with your science; give me the pan and I will tell you what the stuff will go." When one makes a comparison with the system in vogue to-day when one is perfectly satisfied to take the statement of the value of the ore reserves as if it were an extract from a ready reckoner then I think it is fairly proved how much your work has contributed to the stability of these fields, and furthermore has created a feeling of confidence, the absence of which we formerly had to deplore so very frequently. We hear from time to time how such and such a measure has shaken confidence but the public is not fair enough to say how many things have been done that have established confidence and managed to maintain it day after day.

Your sphere of work is a wide one and as I say you are not satisfied to simply carry on the work that is the most pressing at the moment; you are also taking a hand in educating the youth of the country and your publications reach the remotest corners of the world; your mining exhibition has certainly served a very useful purpose and you are devoting a great portion of your time to improving the mines as regards health and sanitation; in short the work you carry on on these fields is being appreciated all over the world. Well do I remember how in 1890 to 1891 we attempted to wrestle with the problems of the Hungarian process and it was a matter of great satisfaction to me last year to see that some Hungarian managers came out here in order to study Rand methods. Your society has had a glorious past, every year of its existence has added to its lustre and I am certain that you need not hesitate to look forward to the future.

I wish the members of the Society every success in their individual careers and ask all fellow guests to rise and drink with me the health and prosperity of the Chemical, Metallurgical and Mining Society of South Africa.—(Cheers.)

The Chairman (Mr. W. R. Dowling, *Vice-President*): I have in the first instance to apologise for the absence of our President Mr. Saner, whose duties take him so far from Johannesburg that he has been unable to give the attention to the Society which he otherwise would. Nobody regrets his inability to attend to the interests of the Society so much as Mr. Saner himself. Among other duties Mr. Saner would have responded to the toast just given, and now on your behalf I have to thank Mr. Wagner for the very kind expressions he has used.

I would like to refer to the work of the Society We are still prospering and have a growing membership all over the world; our monthly meetings are well attended and there is no lack of original papers and lively discussion. In this we are still keeping up our reputation.—(Hear, hear.)

Some of the surface men, like myself, have thought that the mining men have not taken their share of the work of the Society, but of late we must admit that they have been doing quite a lot of good work. I hope this is only the start and that they are going to keep it up.

One feature of our Journals, to which Mr. Wagner referred, is the large number of abstracts which we make from all the important proceedings of other Societies and of the technical Journals. We are spending quite a lot of money on this portion of the Journal and a word of advice from me to our younger members is to follow up the whole of the proceedings as well as these abstracts, because there they will find matter which is not to be found in any text book and when they begin to tackle practical propositions they will know something of what other people are doing and have done.

Members of the Society do not all entirely realise the amount of work that is done by the Council because all they see is the Journal; they read it and forget how it was produced. The Council, and particularly the members of the Editorial Committee, have quite a lot of work to do and have to give much of their spare time to it, in order properly to deal with the papers, abstracts and proofs and all that sort of thing. The Society, however, is fortunate in having such a good Secretary and to him is largely due the success which we have attained.

We are often accused by fellow workers from oversea, that on these fields we are too conservative and do not adopt the best processes that are available. Our critics are really speaking without knowledge of the local conditions. In any case any new proposition that is brought forward must "make good" before we can adopt it and before we can ask our financial people to put money into it. A great deal of money has actually been spent in testing various processes and those that have not proved successful have consequently not been adopted. Some people are disappointed and we are accused of not adopting the best processes. Of the recently introduced processes on these fields there are two deserving of mention, the Butters Filter and the Nissen Stamp. Both are working on a practical scale but only on one has a paper been read before the Society and that was on the Nissen stamp, by its designer, Mr. P. N. Nissen, who is to be congratulated on his enterprise. (Hear, hear.) The Society is looking forward to a paper on the Butters Filter. (Hear, hear.)

In the April number of the Journal appears an abstract which I should like to read to you, of the remarks of the President of the Institution of Mining and Metallurgy in London, who is reported as saying the following regarding the advantages of Metallurgical specialisation :—

> "Referring to the growth of the mineral industry, he said that the world's production of gold between the year 1861 and the year 1890 averaged less than 5½ million ounces per annum ; but in 1891 the cyanide process was introduced on the Rand, and since that date the world's annual production of gold had enormously increased, amounting for the year 1910 to about 23,000,000 ounces. Some credit for this increased production belonged, no doubt, to the mining engineer, who had applied himself so successfully to effecting economies in the cost of mining, raising and transporting ore ; but the larger credit was due to the metallurgical engineer who had so adapted the cyanide process to special conditions, that low grade deposits, which could not otherwise have been profitably exploited even subsequently to the introduction of that process on the Rand, had been worked to great advantage.
>
> In the growing tendency throughout all trades and professions to specialise, the members of their profession, and more particularly those belonging to their institution, had been among the foremost, with the result that they had contributed appreciably to the elevation of their professional status. For more years than he could remember, not only was a mining engineer supposed to know all about mineralogy, geology, and the working of mines (for which it was necessary to have a practical knowledge of mechanical and electrical engineering), but metallurgy in all its branches was included among his essential requirements. Even at the present day, there were no important universities of 'metallurgy,' although they had their school of 'mines,' and the 'Mining' academy at Freiberg, the fact was overlooked that the metallurgist was a specialist, whom they, as mining engineers, gladly recognised as being no less important to the success of mining enterprise than themselves."

I take it that members can take some credit to themselves for having contributed to the improved methods and results referred to.

I would like to mention a side line which members of the Society have taken up. On the menu card in the seal of the Society, there is a curious looking object which represents a mealie. You will be surprised to learn that quite a large number of our members are interested in practical farming and hope some day to give up the hurry and scurry of mining life and finish their days in the quiet atmosphere of farming life. The idea of identifying the future of the Society with agriculture is due to one of our Past-Presidents, Mr. J. R. Williams. The idea was a happy one and the prophecy has come true.

Mr. Wagner, I wish to thank you on behalf of the Members of the Society for proposing the toast in the pleasant way you have and you, gentlemen, for receiving it so well.

"THE MINING INDUSTRY."

Mr. H. A. White (*Vice-President*), in proposing "The Mining Industry," said that when the society wished health to the mining industry a real sincerity must ring through the phrases used. As a corporate body their life was bound up with the prosperity of that industry, and their proceedings had always its welfare in view. They were chemists, metallurgists and miners, and it was right to mention what each branch was doing, or had recently done, to further the cause they all had at heart. Their chemists have solved the problem of destroying the contaminating cyanide which prevented the immediate return of the residual sand direct to the mine, and this to such effect that one mine was now returning to its stopes every particle of sand derived from the rock mined a week before. They had also provided entirely new and most delicate tests to secure the absolute safety of this process ; they had devised simple means for determining noxious gases in underground working places, and were busy perfecting methods for determining exactly the amount of floating dust in mine air and so facilitating its reduction. Their share in promoting the health of the industry in the most literal sense was a most important one. Their mining members were now considering the best means for reducing the number of accidents in mines, and the discussion of this subject has produced many useful suggestions. The mining exhibition recently held by the society was replete with safety appliances, which were fully explained in the lectures given on their use, and in connection therewith the Council had decided to offer a valuable trophy for competition among St. John Ambulance workers. Special prizes were also being offered to students for papers on mining and other subjects. The problems of mine ventilation had received special consideration, and especially in this direction the society felt that the impetus given to progress was largely due to the efforts of its members.

What the metallurgist had done for the industry might best be expressed by the fact that without the cyanide process and its developments on these fields four out of every five of our mines would be working at a loss or else closed down entirely. The metallurgist was still up and doing ; new and successful means for separation and collection of clean sand, evolved in their midst, were being introduced on many of the mines, and ideas from beyond the seas were being turned into valuable account in the treatment of slime and of gold solutions, wherever conditions were suitable. They were cutting down the area of copper required for amalgamation ; they were crushing more finely and yet more cheaply, so that soon one pennyweight of gold, about one part in half a million, would represent the cost of its extraction, added to the value of the residue. Yet in spite of these facts attempts had been made to treat their profession as one the mechanical engineer might pick up in his odd moments, if indeed any effort at all were required. In the annual report of one of the newer mines, where crushing had recently commenced, he found that to a mechanical engineer the entire credit for the design as well as for the erection of the reduction plant was given. Although he would be the last to detract from the credit due for the most excellent and skilful engineering displayed in that mine, he ventured to surmise that its successful operation, at least to some extent, depended upon some metallurgist whose fame was there unsung, though it was certainly not unknown to them. Now, this was only typical, and he was sure that not five per cent of the annual meetings reported in the local press had any reason given them to imagine that such a profession as that of the metallurgist had any connection with the success of their mines. Again, with what ease did a great many people fall into the error that the troubles—out of which their Far East giant was having a happy issue—were metallurgical only. Now, they all knew that trouble did not lie in that direction at all.

This is an age of specialisation—evolution is from the general to the particular—and the time has gone by when the general knowledge of all branches of science connected with mining, which is necessary for the successful control of this industry, is in itself sufficient for the practical working of all the processes required. Each branch demands a special study, and the advance of knowledge in all directions has become so great that in one branch alone can any man, however brilliant, lay claim to be an expert. Disastrous failure may accompany any ignoring of these important facts, and this has been illustrated more than once in this country. Gentlemen, we are proud of our industry, and have every right to be so, and, alas! every need to emphasise the fact on such suitable occasions as this. If Satan came to the Transvaal he would probably get a wire from the South warning him not to let his court make a premature acquaintance with Johannesburg lest their morals might be contaminated. When reading the political debates one often feels inclined to wonder if the phrase "unctuous rectitude" has not too soon been forgotten in Parliamentary circles. The desire for the health of our industry is often found on the lips of Ministers, but theirs is the old-fashioned remedy of "bleeding" the patient. We believe that our way is best, and we will continue to serve this great mining industry of ours to the utmost of our ability. (Applause.)

Mr. John Munro (*Vice-President Chamber of Mines*): We have all heard read with regret the telegram from Mr. Hamilton, President of the Chamber of Mines, that in consequence of his ill-health he was unable to be present to-night. I, in a special manner regret his absence, because this morning I had a telephone message from him saying he was unable to be present and asking me if I would reply for the Mining Industry, instead of him. I may say that it was a surprise to me that I should be asked to reply for the industry because I have nothing in particular to say. Like yourselves I am a working man and do not think very much on the larger questions, but I must say that, from the remarks Mr. White has made, the Mining Industry acknowledges the work that is done by your Society in enabling the industry to work at a profit, because I am quite sure a very large proportion—perhaps not so large a proportion as he has stated—of our mining propositions would not be able to be worked at a profit unless it were for the research of the metallurgists of the Transvaal.—(Hear, hear.)

At a recent meeting of a kindred society the whole position of the Mining Industry was very clearly placed before the public by my friend Sir George Albu, so that I need not go over the whole position again, but, since that meeting, there are two matters that have cropped up prominently in front of the Transvaal population. One is the Miners' Phthisis Bill and the other is the Estate Duty Bill. So far as the Miners' Phthisis Bill is concerned I can only say that from the mining industry point of view we have been most anxious, ever since the knowledge that the dust of the mines was detrimental to the health of the workers in the mines, to alleviate the possibility of there being danger to health —(Hear, hear,)—and from that point when we ascertained that this dust was detrimental to health every effort has been made by the Mining Industry to alleviate the trouble in connection with it. We have not had very much assistance from any other source. Our managers are perpetually informing us that by doing this that and the other we would help to make the trouble less. We have been trying and we now feel we are in a position to so alleviate the trouble that it will not be what is called an occupational disease but we will entirely dispense with it.—(Hear, hear.) We are quite prepared to take our responsibilities, but we call upon the Government also to take their responsibilities—(Hear, hear)—because we have all been at fault.

The next question is death duties Well, I do not know that it will interest you very much, but it will interest the foreign shareholders very much. There has been a great to do about the duties, and I think the Government is floundering and its leaders do not know where they are. I do not want to go into the whole question, but I should like to point out what is the effect of frightening away foreign capital. I will just put one case before you—probably not one of the most important in these fields, but still it is one of importance. The mining companies with which I am particularly connected* paid for the year 1911 dividends amounting to £524,254. That was a small proportion of the gold extracted from the mines. A very large amount in excess of that was paid in working expenses, as anybody studying the statistics will know. In contrast to that there was £655,367 paid by the firm with which I am connected in developing properties. Not a penny was received back in connection with that expenditure. That is only one year's work. There have been several years before that in which somewhat similar amounts have been paid; there will be several years yet when probably larger amounts will have to be paid. I am only giving you one instance of what it means if the Government frightens away the foreign capitalist from investing his money in this country. Take another view of the case, £130,000 more money is spent in developing the country than is received in dividends by all the shareholders connected with our organisation, and yet they say we must tax the foreign capitalist. I think the proposition that the Government now want to make out is most outrageous.—(Hear hear.)

In thanking you for proposing the prosperity of the mining industry I can only say that we always rely on the advice of our technical advisers and that your society is one that plays a very great part in the decision at which we arrive as to whether we shall go on with work or whether we shall discontinue trying to find out whether we can make the proposition pay or not, because you have after all the final word.

South Africa.

Mr. Alex Aiken: Ever since I was told I had to propose this toast I have been wondering why. There is a popular belief, I suppose there is a myth about it, that I know something about figures. If you expect me to tell you how many square miles there are in South Africa I can only say I have no idea. If you want me to give you other figures, I heard Mr. Munro talking about millions by the yard; you had better apply to him. Similarly I cannot say anything about the agricultural or the chemical possibilities of South Africa. Mr. Andrew F. Crosse I believe has made some very interesting experiments in that line with regard to the chemical constituents—I believe that is the proper term—of the soil of South Africa. That I know nothing about. Under these circumstances I am I think in a favourable position with regard to the toast. There is no man so unbiassed as the man who says "I know nothing about it."

There is an old wheeze that South Africa is very like another place; it has a very bad lack of water and too much bad company. There is a lot of truth in that. I will say nothing about the lack of water. That is one of the things you can deal with or at all events try to deal with; it is one of those propositions that the Chairman has been telling us about and it is up to you gentlemen to find a solution. But, with regard to bad company, there I think we hit the nail on the head. So far as I can see all we can want of any country is that it should provide for humanity an opportunity, with a certain amount of hope of success, just a sphere for a man to exercise what intelligence he has got and what perseverance he has. South Africa will give you all that. You may actually despair about it but still get your chance and so far as you have your chance I see no reason for any one of you to ask any more. It is up to you. I am in the position in which I can lecture you. There is no occasion for me to take my own lessons to heart. I think I ought to have done what Mr. White did, seeing that I have to go through it somehow. All tnat I can say really about South Africa is nothing, but what I will say—(Laughter)—is this: not about South Africa, but about the people in it. When I was younger and probably better I was taught that one of the wisest of men preached a good deal about three great virtues as being the supreme virtues. One was Faith, another was Hope and a third was Charity. With regard to Faith and with regard to Hope I know of no community that will beat South Africa. I have only to illustrate that by the remarks of Mr. Wagner to-night with regard to assays—(Laughter)—and the sort of thing we hear of at meetings of companies and the wonderful prognostications of politicians; they never come true but that has nothing to do with the case; the hope is perennial. Also think of the company prospectuses when you desire an illustration of Faith.

But gentlemen, while that is so, there is I think no place in the world where there is so very little charity—(Hear hear),—the charity that thinketh no evil, the charity that is rather willing to believe the best than the worst about people. That, I tell you frankly, there is a sad lack of in South Africa so far as I have seen. We have got our troubles and difficulties but both are as a matter of fact indispensable to any country because it is only troubles and difficulties that give men like you an opportunity of doing something.—(Hear hear.) They are the hammer, as the country is the anvil on which you have to mould your future.

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* The Barnato Group of Mines.

I have great pleasure in asking you to drink the toast of South Africa.

Mr. H. J. Hofmeyr (*President, Rand Pioneers*): Gentlemen, I am suffering from an occupational disease—(Laughter). I do not think there is any man in South Africa who has made more speeches about South Africa during the last 18 months than I have done and I think I was quite entitled when I came here to look forward to some compensation—(Laughter). But what have you done with me to-night? You asked me as a guest and yet you make me pay my footing—(Hear, hear). You ask me to come here and enjoy myself and you place me between two of the most mournful individuals possible—(Laughter)—as lugubrious as undertakers. That is the hospitality that I receive at your hands.

Then take Mr. Aiken, who preceded me. I was looking out for something I could criticise in Mr. Aiken but, with that frugality which is the national characteristic of Scotland he has only told me that we are not charitable. I quite agree with him. I am quite certain you are not charitable, as I have already told you.

Mr. Aiken is in doubts as to why he proposed the toast of South Africa. I entirely share those doubts with him—(Laughter). I go further and extend the doubt to myself. I can only say if you ask me why it reminds me of the gentlemen of Hebrew persuasion who went to a store and asked for some cocoa; the assistant asked "Vi Cocoa?" to which he replied "Vy not?"—(Laughter). That is my only excuse for being here likewise; we do not know why we should not be here.

Gentlemen, it has been said I cannot be serious. That is a distinct slur on my character. I am now going to be serious. I think my friend was perfectly correct when he said when we have the toast of South Africa before us we are too apt to forget it is not South Africa we have to deal with but it is ourselves as individuals. If each one of us put his shoulder to the wheel and left off talking about South Africa in this vague general way I think things would be much better for us in this country. There is no doubt about it; we are inclined to think things will come all right by themselves. Eighteen months ago, when we entered upon Union we thought the millenium had come and we were just like children who expected to be coddled and lifted over all the rocks and stones in the river beds that came in our way, without any effort on our part. Gentlemen, if that were so we would soon become atrophied; we would soon become an entirely worthless nation, a byword to the other virile nations of the world. We have to expect troubles and we have to expect difficulties. Sir Owen Lister was accustomed to say to his students "Gentlemen, when you meet a difficulty be sure you are on the eve of a discovery" and I entirely endorse what Mr. Aiken has said in that respect, that difficulties are made only to be overcome. It is not for us to whine and squeal but when difficulties come we have to fight them, find a way out, and overcome them.—(Hear, hear.)

I do not know why your training should not lead you to help us in this fight. Surely you are the men who have often stood before what appeared to be insoluble problems, you have investigated the matter and after boundless research the end of it has been that what appeared to be a great difficulty has dissolved before your view and before your work. We are in such a terrific hurry to do everything at the same time. Have you ever found a scientist who dabbled in six problems at the same time? Certainly not because he would fail at the whole half dozen. We are all trying to pull everything right in five minutes; it is impossible for us to do it. There is only one example of that sort of thing and that is the exception which proves the rule—the suffragettes. They have only one thing in view and they are going to get it, but look at the fury and the frenzy they employ in doing it.

There is no nation that has arrived at any greatness of pre-eminence that has not tackled one great problem at a time and if we are not going to concentrate our forces we shall be weakened; we must tackle one problem at a time. Our public jumps from one thing to another; there is no patient persistent effort in unravelling difficulties and getting to the solutions of the problems before us.

I remember hearing one judge say to another judge who was particular about his appearance: "Is there anything ridiculous in this wig?" The reply was: "Nothing but the head."—(Laughter.) This is a fine country but there is something wrong about our heads. The only thing ridiculous in this country is going to be our heads, our brain power, our power to sit down and clearly reason things out. We have had examples—many of them We have during the last few weeks had a very difficult problem put before us and of all the people who have joined in the fray how many can throw any enlightenment on the subject. It is for you as citizens of this country, to bring your training to bear, even on questions of that sort. If you do not mind my saying so I want you to act more as citizens of this country than you have done in the past. It is not we alone who are in this room who ought to be appealed to ; this is a general appeal to all men who have made this country their home. When it comes to doing work which is somewhat outside of your own particular groove who is to be the man to come forward? When anyone steps into the breach, only too often friends wink their eye and say "There's another fool coming along. What does he mean by making an ass of himself by going on to the Town Council or some other body ; he is just ripening for the lunatic asylum."

South Africa wants everyone of us. We hear what a wonderful country Canada is ; what a wonderful place New Zealand is and of the wonderful possibilities of Australia. I do not know any other country in the world like South Africa, with all its opportunities and advantages. Do not let us throw up our hands at the first reverse we meet with ; let us go forward and make this country what it deserves to be.

There was a preacher who said once to his audience "My friends, there is no one amongst us who is without sin. If there is anyone amongst you without sin let him get up and let us see him." Nobody rose. Then he proceeded to say "I knew there would be nobody without sin amongst you, but is there anyone who has seen anybody without sin?" There was no response. Then he grew emboldened by his success and he said "I go a step further and I ask is there anybody amongst you who has ever heard of anybody without sin?" Then there was a rustle in the gallery as a lady got up. The preacher was somewhat taken aback and he said "My dear sister, you misunderstood me. What I said was "Was there anybody here who had ever heard of anybody without sin?" She said "I quite understood. I have heard of somebody without sin." He said "For goodness sake tell us who it was?" She said "It was my husband's first wife."—(Laughter.)

You may talk as much as you like about other countries ; it may have been your first wife ; it may have been a place you have only heard about ; but this country is quite good enough for you—(Hear, hear)—as it is quite good enough for me. Let us see the thing through. I hope to see people coming forward to take part in the government of the country. Those are the men that work. What are we doing, we who cannot get out of the country, who have no escape door like you find in a theatre, to be used in case of fire ; we have to put the fire out.—(Cheers.)

Gentlemen, I thank you very much for the way in which you have received this toast.—(Cheers.)

OUR GUESTS AND KINDRED SOCIETIES.

Mr. E. J. Laschinger (*Vice-President*) I am glad, as I am called upon to make a speech, that this is one of the last and it can therefore be short. It is regrettable to find the Government of South Africa taking so little interest in the affairs of our technical societies as is generally evidenced by the attendance accorded to invitations. Most of the members of the Government are of course in Cape Town at the present time but in enumerating our guests we have with us Mr. Frank Brown, M.L.C., whom we are very glad to welcome here.—(Hear hear.)

We all welcome our sister Societies. We have here the President of the S.A. Institution of Engineers. Some people think the engineer a necessary evil. He certainly is necessary. If the engineer goes away from the mine for a time there is sure to be trouble. Then we have the representative of the allied society the S.A. Institute of Electrical Engineers, whom we wish every success. With regard to the medical profession there is no doubt it has its influence on thousands of workers and that here on the Rand its members have a splendid opportunity for study and work. Mention of the medical profession naturally brings one to the Society of Pharmacists who are also represented here tonight. We only hope they will dispense their prescriptions with increasing knowledge and care. We have also as guests members of the Geological Society. Their work is certainly useful and it is probable that it is more scientific to some extent than that of most of our technical societies. Although the geologist cannot predict where gold is to be found he propounds very elaborate theories upon how the gold has occurred in deposits which have already been exploited. Certainly also within the last few years geologists have been able to trace and to indicate in some degree at least the extent of payable fields from the information which they have gathered from the work which has been done by them. By means of correlating data all over the world they are able to indicate where the mineral wealth of any country may possibly be found and we wish them every success in their work.—(Hear hear.)

We have here a representative of the Mine Managers who some people consider are also a necessary evil. We wish them prosperity in their careers.

The Society of Accountants is also represented —these men who handle the finances of the fields —on paper, weird entries in books, on both sides thereof, and in the end make them balance. I am sure there is a great deal of science displayed in their operations. How they do it is a mystery which is only known to themselves. I think they keep it a great secret. Certainly if we have accountants it is better that they should be chartered and recognised authorities on the subjects to which they give their attention.—(Hear, hear.)

We have also to-night representatives of the Rand Pioneers. The name carries us back to what has been done either by ourselves or by our forefathers in pioneering work in various parts of the world and it is but meet that we should recognise the work of the early pioneers. We do so as far as we can by asking them to be our guests at our Annual Dinners and we will all gladly drink long life to the Pioneers of the Rand.—(Hear, hear.)

We have also representatives of the Chamber of Mines, various mining houses, the School of Mines and besides these I have enumerated there are others. No one will think I have put the guests in order of merit or made any invidious distinctions with regard to them.

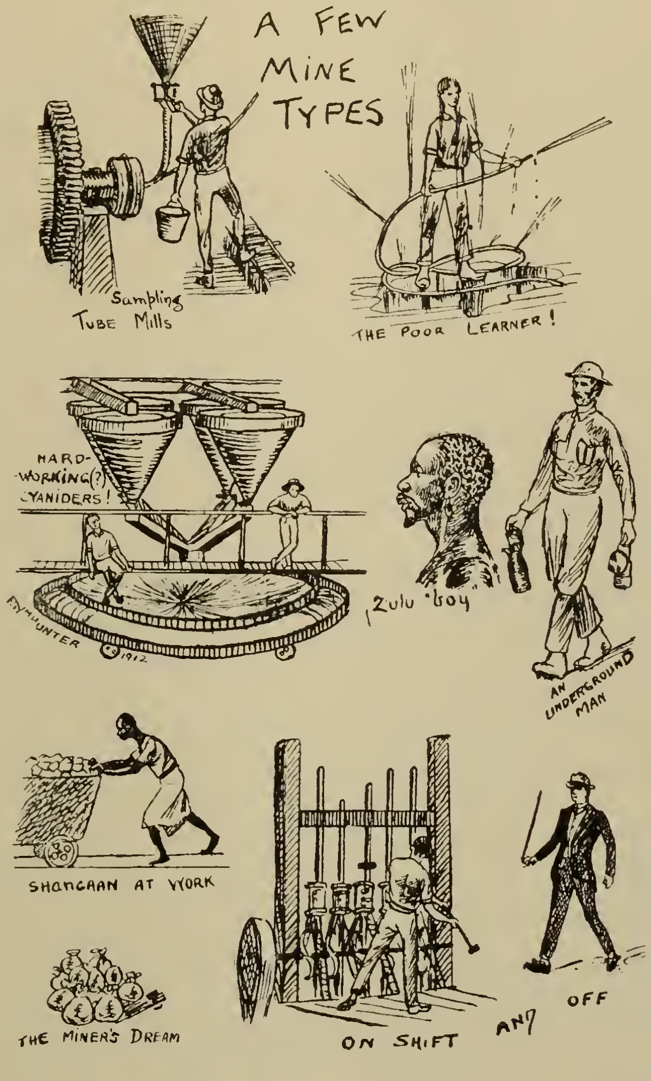
It is I think a significient sign to have active technical bodies in our midst in the Transvaal. They are not only an indication that progress is being made, but they also make for progress, and it is the duty of every technical man to identify himself as much as possible with the society interested in his work. One of the best ways in which a man can be a help to his profession is to belong to a society that represents his profession—(Hear, hear)—so that the solutions of problems affecting it can be disseminated amongst men who are best able to understand his work and propagate the benefits which may be derived from it.—(Hear, hear.)

In this connection the quaint words of Francis Bacon apply :

> "I hold every man a debtor to his profession ; "from the which, as men of course seek to receive "countenance and profit, so ought they of duty "to endeavour themselves by way of amends to "be a help and ornament thereunto."

Mr. Dale Lace : I rise to thank you for the hearty way in which you have received the toast so ably presented to you by Mr. Laschinger. It is a grand thing to see so many representatives of our various societies gathered together and to hear the note of optimism when there has been so much of pessimism. It is only the optimist who makes a country.—(Hear, hear.) We have no room for the pessimist. This is a country which will hold its own in the future—probably in the very near future—with any other of our colonies.—(Hear, hear). We have our talents and we are going to develop them and we have our talented men capable of developing them. We may make mistakes, no man is any good who does not make mistakes and he has at any rate the satisfaction of putting them right. We need not be startled by what we hear goes on in Cape Town and elsewhere ; we have our industry, and who has made it? The Chemical, Metallurgical and Mining Society.—(Cheers). I speak as a pioneer. That is one of the Societies that I have to answer for. I saw the introduction of cyanide and without cyanide and without science we should have been nowhere. Your Society has amongst it men who can give a lead to any similar Association anywhere else in the world, and I say it is up to you to see that things are put straight in this country. If mines are not payable they must be made payable.—(Hear, hear.) We must not wish to go too fast ; we must walk before we can run ; but that this country is going to be a big country and a prosperous country goes without saying. What was done in cyanide may be done in other things. No one yet has touched upon the progress that is going on in the sense of the better use of power ; I refer to the small drills and the effect that that is going to have on the affairs of the industry. The science of the age will alter the circumstances of the country and we are not going to be beaten by the native ; we are going to beat the native with science and do without him. The great remedy for the ills of this country would be to see it flooded by immigration but that is a thing that some will not face. If our Government will not face it, then we have to face it in another way for ourselves. That is the only remedy. (Hear, hear).

Mr. R. C. Warriner (*President Association of Mine Managers*) : I am very sensible of the honour done me in asking me to respond for the institutions represented here to-night. We are all much indebted to you for the good cheer and the hospitality you have extended to us and the expressions of welcome you have given us. I do not think that there is much left for me to say except to express a vote of thanks to Mr. Dale Lace for the way in which he has covered the whole speech. There are however a few points I should like to mention. You have here represented to-night as your guests the S.A. Institution of Engineers, the Association of Electrical Engineers, the Geological Society, the Pharmaceutical Society, the Medical Society, and other societies which are not distinctly of a scientific nature, such as the Mine Managers Association. It may be thought by some that it is difficult to see how the Chemical, Metallurgical and Mining Society is kindred with all these different societies, but I am sure you will agree with me in saying not only are these societies all akin in working for the best interests of the industry, as well as for the promotion of scientific knowledge, but they are practically as it were first cousins. Each association is or should be working out its own branch of exact knowledge. No society can be otherwise than strengthened by the light thrown upon its labours by some similar research in other directions. A great many years ago—1660, or some thing like that—the Royal Society of London was started. It is still alive and flourishing. The original members comprised several medical men, an astronomer, a mathemetician, various other seekers after knowledge, and a doctor who afterwards became a bishop. Although your society has not been in existence for many years, comparatively, I notice you are already extending your scope and taking in a great many subjects other than those with which you started to deal. I may say that it is only a little while ago that you added to the original Chemical and Metallurgical Society the additional title Mining, and I presume in a few years more you will take on a medical society, it may be the astronomical society will follow, and, judging from the suggestions I have heard to-night possibly Agricultural may also be added to the name! I take it they are all going to be submerged because you are extending your reach and taking in various sub-



jects which, to begin with, you did not attempt to deal with. It seems to me that your field of research in the direction of metallurgy and chemistry is a very wide one indeed and if you are going to cover that one branch really well you must devote your whole attention to it. That is if you are going to be specialists in chemistry and metallurgy why not devote your whole time and energy to proficiency in that line and extend your operations in that line, so as to make the chemistry and metallurgy of gold on these fields the foremost in the world. We represent the largest portion of the world's output and accordingly that field is an exceedingly wide one. May I express the hope that in the future you may not continue to widen your field but instead to concentrate your efforts in the direction of those lines in which you are really specialists and bring the Chemical and Metallurgical Society of South Africa to be what it should be the foremost in the world of scientific societies dealing with the metallurgy and chemistry of gold.—(Hear, hear.)

Sir George Albu: Mr. Dale Lace's desire is to be fulfilled, and I am given an opportunity of saying a few words to you in asking you to drink the health of your Chairman.—(Hear, hear.) I was under the impression that your science was an exact science. When I find from the reports that extraction is 95%, then I think it is an exact science; when I find that the extraction is 105%, then I don't know where I am! Then I believe it is an abstract science. I find on the back of your menu card some interesting illustrations, varying from the learner who does the work to the manager who goes off to the Club.—(Laughter.) Or it may be the amalgamator on the mash.

The Chairman suitably responded, and the National Anthem brought an enjoyable function to an end.

Reviews.

"Text Book of Rand Metallurgical Practice," Vol. I. (Chas. Griffin & Co., Ltd.)

"The object of the authors has been to describe metallurgical practice, both mechanical and chemical, as actually carried out on the Rand at the time of writing." Thus says the preface of this volume and readers will, we feel sure, agree that the authors have performed their task in a very satisfactory manner and that the book is the embodiment of a good idea well carried out.

It is not very many years since the cyanide process was an infant industry and the world looked on with surprise at Messrs. MacArthur & Forrest's early achievements with a solvent which was destined to revolutionize the metallurgy of gold ores. In the interim many books have been produced dealing not only with the wonderful progress of cyanidation, but the metallurgy of gold generally. These works were commendable inasmuch that they recorded progress, but it is now some years since any serious attempt was made to deal with Rand metallurgy. Meanwhile great strides have been made and our practice, both milling and cyaniding, has been so developed that we wonder whether any further material improvement is possible and ask whether finality as regards design and practice is not nearly attained. So rapidly has improvement followed improvement in the design of plant that it is no exaggeration to say that not infrequently sections of new equipment are of obsolete pattern even before they are placed in working commission, a fact which, whilst it doubtless worries our financial controllers, is nevertheless indicative of a move in the right direction and in the best interests of the industry.

To accomplish the difficult task of producing a description of to-day's metallurgical practice on the Rand a number of authors have collaborated, each confining his attention to that section of the subject in which he is a specialist. Thus we have, one might say, a volume written by working specialists who, in presenting their work, have carefully kept in mind the idea of benefitting their co-workers by giving them a clear, detailed and reliable description of the latest practice and line of thought. No metallurgist on the Rand can afford to be without it and the value of the volume has been augmented by a very complete bibliography and a good index. The authors, Messrs. Stokes, Thomas, Smart, Dowling, White, Johnson, Caldecott, McA. Johnston and Schmitt—all prominent members of the Chemical, Metallurgical and Mining Society of South Africa—are to be congratulated on producing a work which is at once masterful and lucid, displays much care, thought and research, and withal is written in a high style which makes it pleasurable reading for men of all sides of the mining and metallurgical profession.

The work is presented in two volumes, the first of which is to hand.

It opens with a brief history of the industry, written by Ralph Stokes, in his usual interesting and lucid manner, and in the course of his remarks he deals with the geology of the district and the possible life of the mines. In discussing equipment, he claims that the policy of centralisation must be largely credited with the immense reductions in capital outlay and working costs which have been effected in recent years.

In Section 2, J. E. Thomas deals with Sorting and Breaking and discusses, among other things, the sphere of action of Breakers, Stamps and Tube Mills. We could wish this section extended.

Next we are treated to an excellent contribution by the late G. O. Smart, whose work on these fields well merited the esteem in which he was held. After enlarging on Thomas's interesting analysis of the sphere of utility of the different crushing machines and dealing with the question of best size of feed, he passes on and reviews the details of mill construction and practice, giving much useful information, and closes his contribution by a good description of clean-up appliances and methods.

W. R. Dowling takes up Tube Milling and the rapid headway being made in this department of our metallurgical practice will ensure this section being read with care. The Rand adopted tube mills reluctantly; we were sceptical regarding their alleged advantages, but doubts have vanished before the light of hard facts, and probably nowhere in the mines of the world is the tube mill now receiving such keen attention as on the Rand, where it has already seriously trespassed on the old undisputed domain of the stamp and promises to trespass still further. The Denny Bros. and J. R. Williams were the pioneers who introduced these secondary crushers into Rand metallurgy, thereby contributing materially to the welfare of the industry. Dowling ably deals with the subject and also with cone classifiers, and gives capital diagrammatic sketches showing the different methods of arranging the units of our crushing, classifying, and amalgamating plant.

Besides writing on Sorting and Breaking, J. E. Thomas is the author of "Treatment of Sand," and herein is found much valuable material including classification, sand pumps, the collection of sand, and the disposal of residues; the last covers the returning of the sand underground and the packing of stopes, etc., therewith—a practice which will be heard of more in the future than in the past. Revolving vacuum tables for the collection of sand are here described and illustrated, and the great incidence on plant capacity of this and other up-to-date methods of sand collection is shown.

The subject of Slime Treatment has been taken in hand by Messrs. H. A. White and W. R. Dowling, and is divided into two sections. Much may be learnt from a perusal of these pages in which the matter is skilfully and thoroughly dealt with. Many changes have been effected during recent years in the treatment of slime, and in conjunction with tube milling it is being rapidly developed. The days when slime was *non-desideratum* are passed and to-day we favour its production and would be pleased to see a greater percentage of it. There is a steady trend in the direction of its increase, and in view of its present importance and future possibilities, slime treatment is of considerable moment and its description has been placed in good hands. After discussing the determination of slime in pulp, settlement and the effect of electrolytes and heat on it, a valuable mathematical review of the process is presented, including many excellent tables. The actual routine work of slime treatment is described in a clear manner.

Precipitation, Clean Up and Smelting is from the able pen of E. H. Johnson who deals with extractor box work and sets forth the conditions on which efficient precipitation is dependent. A clear and comprehensive review of the clean up and smelting is given.

A. McA. Johnston is responsible for the Assaying and Testing Section which is quite a notable feature of the volume, and he places before us a particularly good monograph which will be greatly appreciated by all workers.

The next pages, "Chemistry of Banket Ore Treatment," are by Dr. Caldecott, whose thoughful contribution on this now little discussed subject reminds us of the early days of the cyanide process when controversy was keen and Feldtmann, James and many others still on the Rand participated in the unravelling of the tangled skein of cyanide reactions. It is a matter for regret that this section is not longer than it is.

At the end of the volume are some judiciously selected tables by Caldecott and White which should prove useful about the works.

There are one or two things, zinc dust precipitation and vacuum filtration for example, the recent rapid development of which has not permitted of treatment in true perspective, and there would also be a greater abundance of illustrations in the volume were it not for the fact that Vol. 2 will deal with the Design and Construction of Plants.

Within its sphere there is nothing in contemporary literature which approaches this book, and Messrs. Griffin & Co., the publishers, are supplying it in a very presentable guise. (J. Y.)

"TEXT BOOK OF CYANIDE PRACTICE," by H. W. MacFarren, p.p. 291. (McGraw Hill Book Company, New York & London.) Price 3 dollars nett.

The purpose of this work is to furnish students with a practical and technical exposition of the principles and practice applicable to cyaniding in general. It is mainly intended for beginners and to that end simple and homely methods of explanation are adopted. For those who wish to pursue matters further a complete and most excellent bibliography is appended.

The chapters on the early history of the process, on the properties of cyanides and on the dissolution of gold and silver are somewhat meagre, but those on the actual treatment of sand and slime are more full of detail. In especial the handling of slime is well described and all the well known filtration methods are explained with full illustrations. The author has evidently had unfortunate experience with the decantation process for he describes the results at Homestake, where he gives costs at 1s. per ton and extraction at 90% as something not to be approached by any other treatment. The zinc dust process is illustrated and described, and its merits and disadvantages briefly discussed.

The book is very well got up and the type and illustrations are excellent; slime, sand, concentrate and similar words appear without the final (and redundant) "s" and with the exception of "niter" the spelling is English throughout. Some very loose writing is occasionally met with, for example "The settlement of slime by lime or other alkali, an acid, or a neutral salt is on the theory that particles of any kind when suspended in a liquid are electrostatically charged." That kind of theory should certainly "hold water." (H. A. W.)

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THE DETERMINATION OF TOTAL SULPHUR IN INDIA RUBBER.—"A comparative study was made of different methods for determining the percentage of total sulphur in a sample of rubber. The conditions were modified in many ways, the endeavour being, primarily, to effect the complete oxidation of the sulphur to sulphuric acid. It was shown that the method of Henriques may give slightly low results, probably owing to a loss of some of the free sulphur. By this method the rubber is decomposed by digestion with nitric acid; the acid is then evaporated off on the steam-bath and the residue ignited with a mixture of soda and saltpetre. The hot aqueous extract of the melt is acidified and the sulphuric acid precipitated by adding barium chloride.

In order to avoid the loss of free sulphur the authors decomposed the rubber with concentrated nitric acid saturated with bromine. Using pure, powdered sulphur, it was shown that the oxidation was complete, while with nitric acid alone a considerable part of the sulphur was not attacked, and was lost in the subsequent ignition. Blank determinations run with measured volumes of carefully standardised sulphuric acid showed that the ignition occasions no loss of sulphur trioxide."—C. E. WATERS and J. B. TUTTLE, Notes from the American Bureau of Standards, *Journal Franklin Institute*, March, 1912, p. 295. (J. G.)

DETERMINATION OF ALKALIS IN SILICATES BY FUSION WITH CALCIUM CHLORIDE.—"The powdered silicate (0.5 gm.) is mixed with 7 gms. of pure dry calcium chloride in a platinum crucible which is fitted into a hole in a sheet of asbestos, so that only the lower half may be in contact with the flame, and is heated, at first gently, until all the chloride is melted, and then to full redness (though not with the blowpipe). After 25—30 min. heating, when the mass has become more or less solid, the crucible is cooled, and the contents are digested with water in a platinum dish. The insoluble residue is then filtered off and the calcium precipitated by adding ammonia and ammonium carbonate to the filtrate. The precipitate is re-dissolved in hot dilute hydrochloric acid and the calcium again precipitated in the same manner, after which the united filtrates are evaporated to dryness, ammonium salts are expelled, and the residue is dissolved in the smallest possible quantity of water. The last traces of calcium are then precipitated by adding ammonia and ammonium carbonate and allowing to stand for some hours, and finally, after filtering and evaporating, the ammonium salts are expelled, and the residual alkali chlorides weighed."—E. MÄKINEN, *Zeitschrift Anorg. Chem.*, 1912, 74, 74-78., *Journal of the Society of Chemical Industry*, vol. 31, p. 185. (J. G.)

DETERMINATION OF FERROUS OXIDE IN SILICATE ROCKS.—"In accordance with the results of previous investigations (*Journal of the Society of Chemical Industry*, 1910, 1054; 1911, 694, 1209), the determination is made as follows:—0·8—1·0 gm. of the rock, which is not too finely powdered (with the addition of 0·5 gm. of powdered quartz, in the case of basic rocks) is mixed with 2 cc. of concentrated sulphuric acid in a platinum crucible of 30—40 cc. capacity, the mixture being well stirred with a platinum spatula, 8 cc. of hydrofluoric acid are added (quickly, in one portion), and the mixture is at once heated on the water-bath until solution is complete, which often occurs in a few minutes. An atmosphere of carbon dioxide is preferably maintained during heating by passing the gas into an inverted funnel placed over the crucible, the stirrer passing through the short stem of the funnel. The hot contents of the crucible are then added to about 100 cc. of water, containing 10 gm. of pure powdered silica and. preferably, 20—25 gm. of potassium sulphate, to which 2 cc. of strong sulphuric acid have been added, and, the crucible having been rinsed with boiled water, the whole, is, as quickly as possible, titrated with $N/20$ permanganate."—M. DITTRICH and A. LEONHARD, *Z.anorg. Chem.*. 1912, 74, 21-32. —*Journal of the Society of Chemical Industry*, vol. 31, p. 185-86. (J. G.)

SIMPLE METHOD OF DETERMINING COPPER IN STEEL.—"The proposed method depends on the fact that on treatment with sulphuric acid in absence of air, iron is completely dissolved while copper remains unattacked. 3—5 gm. of the metal are treated with dilute sulphuric acid in an atmosphere of carbon dioxide, using a flask holding about 750 cc. the contents of which are warmed until all the iron has gone into solution, and the copper remains as a dark graphitic mass. After allowing the liquid to cool in presence of carbon dioxide, it is filtered, and the residue well washed with water. Finally, the copper is pressed between filter paper, ignited to constant weight in a tared porcelain crucible, and the amount of copper originally present in the sample calculated from the weight of cupric oxide thus obtained. The iron solution may contain traces of copper, but the amount passing into solution is negligible for ordinary technical work."—S. Zinberg, *Z. anal. Chem.*, 1912, 51, 19-20.—*Journal of the Society of Chemical Industry*, vol. 31, p. 234. (J. G.)

Determination of Vanadium in Ferro-Vanadium.—' The following method, whilst sufficiently accurate for ordinary work is very much more rapid than those requiring the previous removal of the iron and more convenient than the fusion methods. 0·5 gm. of the alloy is dissolved in 15—20 cc. of nitric acid of sp. gr. 1·20, the solution is evaporated to dryness, the residue baked for a short time, taken up with a little strong hydrochloric acid, and evaporated to dryness again, these operations being repeated two or three times; after which the residue is taken up with hydrochloric acid, 30 cc. of 1 : 2 sulphuric acid added, and the mixture evaporatd till fumes of sulphuric acid escape. After cooling, adding water, and warming, the clear solution is transferred to a large beaker or conical flask, 30 cc. of phosphoric acid of sp. gr. of 1·30 added, the solution diluted to 500—700 cc., and titrated with permanganate ($V_2O_4 + O = V_2O_5$)."—P. Slavik, *Chem.-Zeit.*, 1912, 36, 171.—*Journal of the Society of Chemical Industry*, vol. 31, p. 234. (J. G.)

Determination of Gold, Silver and Platinum.—" The following method gives results within 0·5 per thousand, or closer in experienced hands. 0-5 gm. of bullion, if finer than 250 per thousand, or 1 gm., if baser, is mixed with pure silver in the proportion of about 10 of silver to 1 of gold and platinum. 3 to 15 gm. of lead are added, and the whole cupelled. Allowing for added silver, the weight of the button gives silver, gold, and platinum together. The silver loss during cupellation is best determined by control assays. The addition of so much silver results in the entire removal of lead from the obtained button. No appreciable amount of gold or platinum is lost. The button is parted, without flattening, in 25 cc. of concentrated sulphuric acid, nearly boiling, for 30 minutes. The residue is coherent, and is washed, and dissolved in *aqua regia*, and the solution evaporated to small bulk, diluted, and the silver chloride filtered off. To the filtrate is added 15 cc. of hydrochloric acid, and 1 gm. of hydrazine hydrochloride. After one hour, with occasional stirring, the gold is filtered off, ignited, and weighed. As it often contains some platinum, it is cupelled with 5 to 8 parts of silver, and the button parted in nitric acid. Silver and platinum dissolve. The pure gold is weighed. Platinum is precipitated from solution by an excess of ammonia or potassium hydroxide, ignited, and weighed."—Trenkner, *Metallurgie*, 1912, 9, 103-105.—*Journal of the Society of Chemical Industry*, vol. 31, p. 234. (J. G.)

The Coagulation of Colloidal Copper.—"§ 9. *General Conclusions.*—(i) The 'square' law which is found to hold for the coagulative process, as showing the relation between the rate of coagulation and the concentration of the colloid, is in accordance with the principle of *mass action*, and indicates that coagulation is brought about directly by the mutual attraction of the particles. It is inconsistent with any form of condensation of the particles round some other nuclei such as the ions.

The question arises as to what is happening in the solution during the initial stage of the process before precipitation sets in. There are two possibilities: either the time is spent in the absorption of the ions of the electrolyte—coagulation commencing when that absorption has reached some particular value; or else, the absorption taking place more or less rapidly, the particles begin to associate practically at once, but do not attain the size of visible ' coagulated ' particles till that association has reached some particular stage, after which precipitation sets in.

The evidence is entirely in favour of the second explanation. The comparison of the coagulation curves of § 7 shows that the duration of this initial period is exactly proportional to the duration of any other stage in the coagulation, as though the processes were of the same nature in each. Further, time experiments on absorption, show that this process is completed almost instantaneously,* whereas in the coagulation experiments, the initial quiescent period extended sometimes for several days.

Again, the increase in size of the particles has been found to occur before precipitation sets in. Linder and Picton † observed that the addition of a few drops of very dilute sodium chloride solution to a filterable arsenious sulphide solution rendered it non-filterable, without bringing about coagulation. They also found ‡ that the rate of diffusion of arsenious sulphide solution was sensibly diminished by the addition of an amount of ammonium chloride quite insufficient for coagulation.

They further observed * the process of coagulation under the microscope, a drop of salt solution being brought into contact with the edge of a drop of arsenious sulphide solution under a cover slip. First of all a clear yellow field in the microscope was seen, and then the spreading along of the wave of coagulation. 'In the wave front itself the genesis of the hydrogel from the hydrosol was clearly visible; first appeared a dim cloud of vibrating particles too small to be followed with certainty; these rapidly increased in size with proportionately retarded movement, until at a certain critical moment, the granules began to attach themselves to the free surface of the hydrogel and to one another, and all movement ceased.'

The actual precipitation, therefore, is associated with the later stages of the agglutination of the particles. But since all the combinations take place at a rate in accordance with the square law, it follows that the rate at which some particular stage in the process, namely precipitation, is reached will also be in accordance with this law—the concentration m in the formula $(dm/dt) = km^2$ referring to the concentration of the colloid still in solution, for the precipitated portion has been shown experimentally to be inactive (§ 8).

It has already been stated that on account of the treatment of the solutions too much importance could not be attached to the results of individual curves. When, however, the curves were examined, it was found that there was a general tendency on the part of many of them to fit in with the above relation—that is, to satisfy the equation $m(t-\beta) = constant$, deducted therefrom. It seems probable therefore that a more accurate method of experiment would show a closer agreement.

(ii) The explanation of the power rule for the variation of the rate of coagulation with the concentration of the electrolyte does not seem to be at all clear. The expression therefore is for the present merely empirical. It agrees with the experimental results very closely, and for a wide range of velocities of reaction. Viewed at in another way however, the range appears limited. Thus the rate of reaction is proportional to the sixth power of the concentration in the solution, that is, to the thirty-sixth power of the concentration on the colloid particles. In order to increase the rate of coagulation 3,000 times, it is only necessary to increase the amount absorbed by the colloid to 1¼ times its previous value.

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* Davis, *J. C. S. Trans.* 1907 (ii.), p. 1666.

† *J. C. S. Trans.* 1895, p. 63.

‡ *J. C. S. Trans.* 1905, p. 1906.

An interesting coincidence appears in the power rule, for the value of the index p (in $R \propto c^p$) is the same within experimental limits as the index n in absorption law $(y = \alpha . c^{\frac{1}{n}})$.

The fact that p has the same value for coagulation with ions of different valency (monad and dyad) goes to confirm Freundlich's explanation of the behaviour of the different ions. When equivalent quantities of the ions are absorbed, the particles behave similarly so far as their coming together in coagulation is concerned.

The complex particles so formed, however, have different properties in these various cases, for those coagulated with monovalent ions can be dispersed again, while those coagulated with polyvalent ions show no such tendency. That this difference lies in the nature of these complexes, rather than in the manner in which they were built up, is shown by the following experiment. The precipitate obtained after coagulating a copper solution with sodium sulphate (which shows no tendency to redissolve on washing with water) was digested once or twice with sodium chloride solution (when the absorbed sulphate ions are replaced by the chloride ones). On again washing with distilled water several times, the coagulum is dispersed, exactly as though the precipitation had been brought about directly by sodium chloride. In a similar way the chloride-precipitated coagulum is rendered quite insoluble if first washed with sodium sulphate solution.

The difference in behaviour may, perhaps, be brought about through the possibility of greater concentrations of electric charges in the case of polyvalent ions, whereby the electric forces binding particles together (if indeed such binding forces be electrical) may become more intense than is the case with monovalent ions.

(iii) These experiments with colloidal copper are interesting as demonstrating that coagulation is a definite time-process, which, in some cases, can be traced quantitatively in its various stages. In so far as the properties of inorganic colloids show marked similarities, it is probable that the coagulation of many of them resembles that of colloidal copper, though the appearance of the precipitate may be very different. The method of experiment described in this paper, a method based not so much on the study of individual curves, as on a comparison of these curves over similar stages in the coagulation, would lead one to expect similar results in other cases, especially as these results are fairly simple.

§ 10. *Summary.* The following is a summary of the most important results obtained in the study of the coagulation of colloidal copper :

1. There is an initial period in which the solution remains clear, and no precipitation takes place.

2. Using colloidal solutions of various concentrations, the rate of coagulation is proportional to the square of the inital concentration ("mass action").

3. For varying amounts of the electrolyte, the rate of coagulation is proportional to some power of the concentration of the anion *in the solution ; i.e.* a proportionate increase in this concentration results in a proportionate increase in the rate of coagulation."—H. H. Paine, M.A., B.Sc., *Proceedings of the Cambridge Philosophical Society*, Michaelmas 1911, p. 452. (H. A. W.).

METALLURGY.

Tungsten and its Uses.—"Germany offers the largest market for tungsten ore in the world, although the domestic use of the metal is exceedingly limited as contrasted with its uses in England and the United States, which altogether absorb probably two-fifths of the total production. The metal itself is employed chiefly for hardening steel, and its varied applications are increasing rapidly.

Hamburg importers are in a position to sell offerings of ore practically in unlimited quantities: The price is based on the content of tungstic acid (WO_3), the usual content running from 60% to 65%. Ores under 60% are not in demand. The ordinary commercial terms are Hamburg weight, analysis by the State laboratory, shipments in bags of 50 kg. (1 kg. = 2·2046 lb.), price per ton of mineral, and percentage of tungstic acid. One thousand kg. of 60% ore should produce about 400 kg. of metal, worth (January 1912) 5·75 marks $1·37 per kilogram, containing 97% of metal, 0·1% of carbon, and free from impurities.

The metal is obtained by reducing the oxides or chlorides in a current of hydrogen, or by means of carbon in an electric oven. According to German patent No. 141811, the raw material is melted at a temperature of 300°C, with sodium bisulphate, after which the temperature is raised to about 800°, and lime or lime salts and alkaline chlorides are added. Sodium bisulphate must always be contained in the mixture in excess of other material. If the method is thoroughly carried out, all tungsten is obtained in the form of tungstic acid, or sodium tungstate. The acid is melted and reduced to metallic tungsten.

The metallic tungsten, used in the form of threads in incandescent lamps is obtained from wolframite. Tungstic acid (tungsten trioxide, WO_3) is imported into the United States for the manufacture of these threads. WO_3 is obtained from wolframite by heating together finely ground wolframite and HCl (hydrochloric acid) ; thus oxide of tungsten and soluble chlorides of Fe (iron) and Mn (manganese) are formed. The three products are carefully washed about a dozen times with water. They are then allowed to settle, when the water is again drawn off and fresh water added. The precipitate of tungsten oxide is dissolved in ammonia, whereby ammonium tungstate is obtained. This is filtered and heated and finally boiled together with HNO_3 (nitric acid) to remove the iron and manganese.

Tungstic acid is obtained from thoroughly washed ammonium tungstate by heating it to a glowing point. This result originated in the manufacture of metallic thread, according to Auer's method (Osram lamp). He removes the oxygen and thus obtains metallic tungsten, which is mixed with an agglutinant and forced through fine diamond matrices. Just and Hananan apply the method of substitution. Kuzel applies the colloid process, according to which the metallic tungsten is alternately treated with strong boiling acids and alkalies. It is, of course, thoroughly washed each time until it is converted into a colloid. The colloid is then precipitated, the liquid drawn off, and the residue, which is gelatinous, is forced through the diamond matrix. Lamps furnished with these threads are called Kuzel lamps.

Another method for the production of colloidal tungsten is to send an electric current under water through two electrodes made of metallic tungsten. In this way small particles of tungsten are thrown off from the electrodes and form colloidal tungsten. The products of the three methods are of equal value. The tungsten thread coming out of the matrix is dried in an oven and then heated in an electric oven up to 2000°C. to drive out the volatile ingredients. Then an electric current is allowed to pass through the thread in an atmosphere of indifferent gas, and the thread for the lamp is thus obtained. The current consumption of the tungsten lamp is from 1 to 1·3 watts per Hefner candle-power, whereas the carbon-thread lamps consume 2·3. The disadvantages are the higher price and greater fragility of the thread. Against this, tungsten thread has a longer life (1,000 burning hours against 450 of the carbon-thread lamp), and it is also much less sensitive to current fluctuations. The tungsten lamp also gives a better light."—ROBERT P. SKINNER. *Mining and Scientific Press*, March 9, 1912, p. 379. (H. A. W.)

CYANIDING WEATHERED PYRITIC TAILING. "The tailing on the dump of the Lady Gladys Co., Mulline, Western Australia, contained marcasite, blende, arsenical pyrite, and the decomposition products of these minerals. It had been exposed to the weather for three or four years and was exceedingly acid, giving off a sulphurous smell which was apparent for some distance from the dump. Although it had been sampled several times and pronounced quite untreatable except at a loss, it yielded a fair profit to my firm although we had to pay 10s. per ton for the dump. The dump contained 2,300 tons of this material assaying 43s. 6d.; after treatment the residue assayed 5s. 4d. The bullion recovered realised, after paying bank and mint charges, £4,377, or a little over 38s. per ton, practically all in gold, the silver being negligible.

Tailing Highly Acid.—The average soluble acidity was 12·2 lb. CaO, and the total acidity 35·2 lb. per ton. The soluble acidity contained ferrous iron equal to 3·3 lb. $FeSO_4$ per ton. A screen test gave the following: On 30 mesh, 2·8%; on 60 mesh, 29·4%; on 80 mesh, 13·1%; on 100 mesh, 8·6; on 120 mesh, 7·1%; on 150 mesh, 4·7%; through 150 mesh, 33·4%. The portion passing 150 mesh was mostly a true clayey slime and contained a larger portion of acid and soluble iron than the sand portion. Had the tonnage been greater, classification and separate sand and slime treatment would have been better.

Owing to the limited tonnage, the plant had to be as simple and cheap as possible, but as long treatment was necessary the capacity could not be too small. Two leaching vats 40 ft. diameter, by 6 ft. deep, were erected each capable of containing 200 tons. The ground was first prepared for the bottoms by gradually sloping it from the circumference to the centre, the fall being 10 in. From the centre a large pipe conveyed the solution to the extractors. Light duck, sewed so as to form a circular surface, 41 ft. 6 in. in diameter, was thoroughly tarred and laid on the prepared ground thus forming the bottom. The sides were of 22-gauge galvanised corrugated iron.

The bottom duck was turned up round a board, fastened to the bottom of the iron sides, to prevent escape of solution. The sides were stiffened by posts placed at intervals of 8 ft. the whole being encircled by three bands of old wire rope. The filter-bed support was formed of slats $\frac{5}{8}$ in. square, placed $1\frac{1}{8}$ in. apart, centre to centre, and resting directly on the tarred duck. These were kept in position by a $1\frac{1}{4} \times 1\frac{3}{4}$ in. laths, tacked on top of the slats. The support was made in sections 5 ft. square, which were put in the vat with the slats pointing toward the centre. The bottom was then covered with Hessian (burlap) and finally with 6 in. of rather coarse sand.

With such vats, which were mere filter beds with retaining walls, it was not possible to check the flow of solution, which was great at times, yet the precipitation averaged 65%. No solution was lost by leakage. These vats were afterwards used for treating another parcel of ore and at the end of two years' use showed absolutely no deterioration as far as the duck bottom and filter bed were concerned, although the iron sides were eaten through in many places. Such a plant, however, could only be justified by the special circumstances.

Preliminary Wash not Profitable.—Tests showed that although so much soluble acidity was present little was to be gained by a preliminary wash. The extraction was more rapid at first, but soon fell off and the total time of treatment was not greatly shortened. Moreover, a preliminary wash involved the use of soda for subsequent neutralisation of the 'latent' acidity, as lime could not be mixed sufficiently intimately with the wet sand. In any case it necessitated another handling. The use of soda was too costly, and was objectionable for other reasons, as will be seen later.

The main feature in the successful treatment consisted in first adding about 50 lb. of slaked lime per ton, which had to be thoroughly mixed, and then passing the mixture through a $\frac{1}{2}$ in. screen, all slime lumps being broken up. After charging the vat with this mixture the next step was to wash with water or barren solution. The first wash always came off slightly alkaline. The original sand contained about 3·3 lb. $FeSO_4$ per ton, which was now converted into ferrous hydrate. It was necessary to thoroughly oxidise this to the ferric state, otherwise the destruction of 8·1 lb. of KCN would result. Oxidation was accomplished by giving a succession of washes and draining between each wash. All wash water was well aerated, air being admitted into the suction of the centrifugal pump used, but the greater part of the oxygen supplied was contained in the drained sand, which was for the most part forced downward through the charge by each succeeding wash. It was estimated that a wash of 31 tons would contain about 0·4 lb. oxygen in solution, but the drained sand would contain about 14 lb. oxygen most of which would be forced downwards through the charge. The washing occupied four to five days, about $\frac{1}{2}$ ton of solution per ton of sand being used. In some cases 1 lb. of lead acetate per ton of sand was added to the wash water, but without marked benefit.

A 40-day Treatment Given. The sand now being well oxidised, about 50 to 80 tons of 0·05% cyanide solution were pumped on the 400 tons of sand daily. A 40-day treatment was generally required, the total solution used amounted to 6 tons per ton of sand, exclusive of wash water for oxidising.

The solution carried gold in amounts varying from 18s. to 20s. per ton at the commencement to 1s. or less at the end of the treatment. The cyanide consumption was a little under 4 lb. NaCN per ton. Notwithstanding the long time occupied in treatment the cost was not unduly high, as after the vats had been filled only one man was required to attend to the pumping, testing and assaying of solutions.

Before the treatment described was adopted the first charges treated threatened total failure. To these vats only 16 lb. of lime were added, the mixture was imperfect, and it was not screened. The idea was to wash out some of the soluble acidity and then complete the neutralisation with caustic soda. The first wash came off showing soluble acidity equal to about 7·6 lb Na_2O per ton of solution ; this with succeeding washes was thrown away until the acidity had fallen to 1·5 lb. Na_2O. The wash was then neutralised with lime, soda added and the solution returned to the vats. A large amount of pumping was required ; much time was expended, an additional 34 lb. of lime and 34 lb. of caustic soda were consumed per ton, bringing the total amount of lime up to 50 lb. and of soda 34 lb., which was more than twice the amount called for by the usual 'total acidity' test.

Upon investigation it was found that on this class of ore the 'total acidity' test (shaking a weighed quantity of ore with a measured volume of alkaline solution of known strength and noting the amount of alkali consumed), gave varying results, depending on the quantity or strength of alkali used and also on the time of contact. The accompanying examples show the effect of long contact in increasing the alkali consumption. The tests were made on ore from both vats at different depths which had already been partly neutralised. Samples of 112 gm. each were shaken with 200 cc. of 1% Na_2O, and the consumption of alkali noted after one hour's contact and after standing 15 days.

TESTS FOR CONSUMPTION OF ALKALI.

"A" Vat Depth.	Pounds of Na_2O Consumed per Ton.	
	After 1 hr.	After 15 days.
0 to 1 ft.	3·76	17·92
1 to 2 ft.	3·04	37·76
2 to 3 ft.	15·84	33·20
3 to 4 ft.	24·40	37·72
4 to 5 ft.	26·64	36·64
Average	14·7	32·6
"B" Vat Depth.		
0 to 1 ft.	1·24	15·04
1 to 2 ft.	0·96	13·84
2 to 3 ft.	2·00	16·88
3 to 4 ft.	10·80	24·48
4 to 5 ft.	20·80	34·40
Average	7·2	20·9

These tests show clearly that much alkali is consumed beyond that required to neutralise the sand if sufficient time be given, one of the chief reasons being the solution of considerable amounts of sulphur.

The presence of alkaline sulphides was never detected in the solutions coming from the vats but thiosulphates were present in quantity. Probably alkaline sulphides were first formed and were immediately oxidised to thiosulphates, etc. Alkaline sulphides could be detected in the above bottle tests that had stood for 15 days, as there was little chance for oxidation.

Several 112 gm. samples of 'Gladys' sand were each agitated with 200 cc. of a solution of either caustic or carbonate of soda at a strength equivalent to 1% Na_2O. The consumption with the caustic solution after 1 hr. was 18·8 lb. per ton ; after 20 hr., 20·96 lb. ; after 72 hr., 25·6 lb. With the carbonate solution the consumption at the end of 1 hr. was 9·44 lb. ; after 20 hr., 9·44 lb. ; after 72 hr., 11·6 lb. per ton.

The caustic gave a coloured solution and settled well ; the solution contained alkaline sulphide. The carbonate gave a colourless solution and did not settle so rapidly : the solution contained no alkaline sulphide. These tests seem to show that when soda must be used the carbonate is better than the caustic alkali.

It was noted that with neutralisation with caustic soda there was apparently a much greater oxidation of the ore than was the case when lime was used. When the treated ore was discharged from the vats its colour was distinctly changed from yellowish to red or warm brown. This did not happen, at least to anything like the same extent, with lime neutralised sand. The colour imparted to the solution was also characteristic of caustic-soda neutralisation, which became a dingy brown and which persisted throughout the whole cyanide treatment of that particular parcel.

According to Julian* 'pyrite is much more soluble in cyanide solutions when a large excess of oxygen is present ; with an excess of oxygen the pyrite caused the solution to become coloured varying in shade from amber to muddy brown ; the brown solution was found to have lost much of its dissolving power on gold and silver but the power was easily restored, by the use of desulphurising and oxidising agents.' From this it would be inferred that the solution of the pyrite with consequent colouring of the solution and loss of dissolving power was caused by the action of KCN solution in presence of excess of oxygen, whereas I found it to be caused by the action of an excess of NaOH in the presence of an excess of oxygen and that the same ore if neutralised with lime gave no such solution.

Constituents of Brown Solution.—The brown solution contained 0·36% $Na_2S_2O_3+5H_2O$, some thiocyanate and the merest trace of ferrocyanide. After aeration it was as effective as colourless solution in dissolving gold, but the addition of permanganate or ferricyanide decidedly impaired its solvent activity for gold. Why this should be so was not clearly established. It was found that the addition of sodium thiosulphate to a pure cyanide solution increased its solvent activity, and of course the addition of permanganate or ferricyanide would destroy the thiosulphate, but this explanation is not altogether satisfactory.

While the addition of 'hypo' to a cyanide solution gave increased solvent activity on immersed strips of gold, yet its advantage in practical work is at least doubtful. Leaching and percolation tests were made on other ores with and without the addition of 'hypo' but the results were discordant. In the leaching tests there was no great difference, but the plain cyanide gave rather better results ; in bottle tests the results varied. The tests were made on ores containing copper as carbonate and sulphide.

Julian shows† the great loss of dissolving power if the same solution is successively passed over several vats of ore containing pyrite and marcasite, and I noticed the same effect when the brown solution was passed several times over the ore without running it through the zinc boxes. Zinc apparently has some power of restoring the solvent activity which mere aeration has not. I have no explanation to offer but the effect was very marked.

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*"Cyaniding Gold and Silver Ores." First Edition, pp. 72 and 104.

†"Cyaniding Gold and Silver Ores," Chap. 10.

After the initial trouble with soda, there was no difficulty in getting neutralisation with an intimate mixture of lime and as already stated the amount of hypo formed increased with the amount of lime added, but the best work was done if the lime was not in large excess.

Methods of Testing.—The ordinary test for reducing power, which consists of adding K_2MnO_4 to an acidified solution, is of no practical use for this type of solution. Solutions showing a high reducing power, owing to the presence of thiosulphates, may actually have a higher solvent activity than pure cyanide solutions of similar strength. The method adopted was to pass a current of air for two or three hours through the solution, testing it with permanganate in the usual way, and also testing solution which had not been so aerated, the difference giving the 'active reducing power' which as a rule was exceedingly small or nil.

At the time this work was going on I knew of no published method for the estimation of thiosulphate in the presence of cyanide, so the following was adopted: Excess of lime was added to remove certain organic matter if present; 10 cc. of the filtered solution was then taken and diluted with 50 cc. of water and a slight excess of H_2SO_4 added (generally 2 cc. of a mixture of 1 volume of strong acid with 2 volumes of water); this was then titrated with N/10 iodine, 1 cc. of iodine equalling 0·0248 gm. of crystalline sodium thiosulphate.

Tests were made on 'model' solutions containing 'hypo,' cyanide, thiocyanate, and ferro-cyanide, and the test was found to be accurate to within 3% of the total 'hypo,' (more 'hypo' being shown than was actually present) which was near enough for the purpose. If the solution were not sufficiently diluted the error was greater. Since that time Clennell in his 'Cyanide Handbook' has given an almost identical method."—HERBERT R. EDWARDS, *Engineering and Mining Journal*, March 23, 1912, p. 605. (A. R.)

CYANIDE REGENERATION.—"*With the introduction of the cyanide process it appeared that finalty in gold-extraction processes had nearly been achieved, but the early application of the process to many ores was accompanied by such unsatisfactory results that eventually only those which contained no refractory elements were selected for cyanidation. In accumulated dumps the difficulty lies chiefly in the high consumption of cyanide, owing to the fact that other metals than gold generally go into solution readily, and the baser metals have sometimes to be fairly well leached out before the gold is attacked. This is particularly the case where copper is present in small quantities, and in the cyanidation of aurocupriferous ores many variations of the cyanide process have been tried, mainly aiming at effecting a reduction in the chemical consumption. No plant operating on aurocupriferous ores has yet overcome copper troubles, and so long as copper is readily soluble in a cyanide solution, in no such case will the initial consumption of cyanide be very materially reduced. The presence of copper in solution gives rise to precipitation difficulties, and these have so effectually resisted correction that precipitation on zinc-shavings had to be abandoned in favour of other methods. In the early operation of the process the only other form of precipitation practised was the Siemens and Halske electrical process, with precipitation-boxes about 4 by 8 by 30 ft., consisting of about six compartments, each 4 by 8 by 5 ft., and each compartment containing anodes of iron enclosed in Hessian cloth and cathodes of lead-foil. The distance between anodes and cathodes was about 6 in., and a current density of 0·02 amperes per square foot of anode surface was generally employed. Low as this current density was, it was sufficient to cause the anodes to decompose rapidly, with consequent formation of ferro-cyanides and ultimate loss of cyanide during the passage of the solutions through the precipitation plant. The only advantage that the process offered over zinc precipitation was that more metal was recovered, and the solutions were returned to the vats in a fairly clean state. The chief difficulty, the high consumption of cyanide, had not been remedied; on the contrary, the cyanide consumption had slightly increased, and the advantages obtained scarcely compensated for the increase.

In 1903 a great improvement in electrical precipitation was effected by Charles Butters, of California. Butters used anodes of lead sheets peroxidised by electrolysis in a solution of permanganate of potassium, and sulphuric acid. These anodes resisted oxidation to a much greater degree than iron anodes, and withstood a current density of 0·5 amperes per square foot without apparent decomposition, and, owing to the use of a greater current density, the size of the precipitation plant was greatly reduced. Butters erected at Virginia City an electrical precipitation plant consisting of boxes 29 ft. long, 10 ft. wide, and 4 ft. deep, and with V-shaped bottoms. Each box had 12 compartments, and each compartment contained 18 anodes and 17 cathodes, spaced 1 in. apart. The anodes consisted of 0·185-in. lead plates, covered with lead peroxide, and not enclosed in any way. The cathodes were plates of ordinary commercial tinned steel, and fitted with an iron strip along the tops to keep them rigid and to connect them together. In using such a high-current density, the gold is precipitated as a pulverulent deposit on the tin, and is periodically wiped off, falling to the bottom of the box, where it remains until cleaned up. In the operation of the Butters plant a regeneration of cyanide was observed, and this was increased as the decomposition of anodes was reduced. It also increased with the increased precipitation of copper and the consequent liberation of cyanogen, which, in the presence of excess of alkali, reformed active gold solvents. It was not, however, until this plant had been in operation for some time that the possibility of a recovery of much cyanide was fully appreciated, and, by carefully regulating current density, alkalinity, and other conditions, Butters was able to regenerate sufficient cyanide to defray the cost of running the precipitation plant.

In September 1905 a patent, entitled 'Method for recovering precious metals from solution,' was granted to Isaac Anderson and Michael Scanlan, metallurgical chemists, of Arizona. In this process it was claimed that the silver and gold could be precipitated in such a manner that the solution could be readily restored to its original activity with but little loss of the original solution and at a small cost for regeneration. This process is based upon the fact that when sulphuric acid is added to a cyanide solution of the precious metals in which sulphides and chlorides are present, the sulphuric acid so added will combine with the potassium of the potassium cyanide to decompose the double cyanides formed by the action of the potassium cyanide on the precious metals, and the metals freed.

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*Abstract from *The Mining and Engineering Review.*

from combination with the potassium cyanide will then combine the sulphur and chlorine present to form insoluble sulphides and chlorides, which will be precipitated, leaving the cyanogen in the liquid in the form of hydrocyanic acid, or in combination with the radicals with which the sulphur and chloride have been combined. The precious metals having been precipitated as chlorides or sulphides, the supernatant liquor can be decanted off, and the potassium cyanide may be regenerated by decomposing the potassium sulphate formed by the action of the sulphuric acid from the solution. The decomposition of the potassium sulphate and the removal of the sulphuric acid may be readily accomplished by introducing lime into the solution and decanting off the precipitated gold and silver, the lime so added causing the decomposition of the potassium sulphate with the formation of insoluble calcium sulphate and simultaneous regeneration of potassium cyanide. In the operation of this process it was claimed that a regeneration of 90 % of cyanide could be effected. At the time of the introduction of this process there was little demand for the advantages which it offered, and there is no record of its operation on a working scale, except on the works controlled by the inventor, where, apparently, it was satisfactory.

In 1908 W. H. Wheelock, an American chemist, in experimenting with the action of sulphuric acid on cyanide solutions, practically rediscovered the process previously operated by Anderson, and contributed an article to the *Mining and Scientific Press* (December 18, 1909) setting forth the results of his experiments. After perusing Wheelock's article I then conducted a number of experiments at the works of the Mitchell's Creek Gold Recovery Co., Bodangora, N.S.W., on the lines laid down by Wheelock, and the following are the particulars of one of these tests: 160 lb. KCN solution containing 0·23 % free cyanide, 1·5 lb. lime, and about 5 lb. copper per ton of solution was taken from the tops of the precipitation boxes and put into a clean cyanide case for treatment. Sulphuric acid was then added (the solution being gently stirred during the addition of the acid) till a precipitate ceased to form. It took 2 lb. of acid to complete the reaction. After standing for two hours, a solution from 5 lb. of lime was added (the portion of lime that did not dissolve in water being discarded); lime solution was added only until the acidity was neutralised. After stirring and allowing the precipitate to settle, the solution was again tested for KCN and it was found to have risen to 0·695 % or a gain in KCN of 10 lb. per ton of solution.

These results indicate that under careful manipulation the process of precipitation by sulphuric acid and regeneration of the cyanide is commercially possible when the following general conditions are observed: The solution to be treated should contain very little free cyanide, because if free cyanide be present a fair amount of acid will be consumed in neutralising it before precipitation actually commences. This cyanide would be totally lost, and so would the acid used to neutralise it. The solution should be as nearly saturated with copper as possible; this can be obtained by passing the solution through successive vats until it will pick up no more metal. Acid should be added until the further addition of acid ceases to make the solution turbid. An excess of acid will prevent a good regeneration and cause a subsequently greater consumption of lime to restore alkalinity and regenerate the KCN. The precipitate formed on the addition of acid should be removed as quickly as possible from the solution and lime added before much HCN could have time to escape.

Anderson claims that a regeneration of over 80 % of the original cyanide can be obtained in the treatment of solutions carrying 0·5 % of copper, and that the precipitate will contain 50 % of copper; but, of course, the results must depend upon other conditions than only the amount of copper present.

At the Mitchell's Creek works several hundred tons of solution were profitably treated by acid regeneration, but lack of vat accommodation, the large amount of precipitate to handle, and the absence of conveniences to systematically continue this method placed it at a disadvantage when compared with improved electrical precipitation. The latter plant will operate continuously with very little attention, and with only a small power-cost: while facilities for acid regeneration would require large storage capacity for solutions and a special smelting plant to deal with the precipitate. On the same works an experimental electrical precipitation plant on the Butters principle was installed, and this plant consisted of a box 10 ft. long by 12 in. wide and 12 in. deep at the sides, and tapering to 18 in. deep in the middle, and containing eight compartments. About 120 sq. ft. of cathode surface was employed, and from 3 to 8 tons of solution passed through the box in 24 hours. The current density rate of flow and other conditions were occasionally varied, but the average of daily returns of this plant during about three months' run is given below:—

Rate of Flow: Tons per 24 hr. 3·5	Top of box.	Tail of box.
Au (dwt.)	1·6	0·4
Cu (lb. per ton)	4·68	3·83
KCN (%)	0·05	0·125

Current Density: Amperes per sq. ft. of anode surface. 0·5.

This average is from the total returns recorded. During the trials two samples were taken daily, each sample being from a continuous siphon, and included in the returns are a number of samples taken while the current was off, and during which period a loss of cyanide was recorded. As a result of these experiments a large electrical precipitation plant was installed, but this has not been long enough in operation to furnish any valuable data. Since this plant started it has effected a regeneration of from ½ to 2 lb. KCN per ton of solution treated, but better results than these are expected when the most suitable condition of the solutions for precipitation has been determined.

Since the installation of the above plant a patent has been granted for a method of electrical precipitation wherein rods of Acheson graphite, each ½ in. by 24 in. are placed side by side to form a plate, and cathodes of corrugated or waterproofed felt or cardboard, which has been made superficially electrically conductive by treatment with an adhesive compound and Acheson graphite powder. Acheson graphite, by reason of its purity and high conductivity, will withstand a very high current density as an anode without injury, and the fine graphite is found to impart a highly conductive surface to the treated cardboard. Cathodes treated in this way cost about £10 per 5,000 sq. ft., and this area is sufficient to obtain good precipitation from 200 tons of solution per 24 hours. The arrangement of electrodes in this plant is such that the greatest possible number of AuCN and CuCN molecules will be brought into contact with the cathodes during the passage of the solutions through the box. In electrical precipitation more depends on diffusion of the molecules than on variations of other conditions, and increasing the cathode area increases the percentage of precipitation more than does increased current density, as decomposition of the metal-bearing molecules only occurs when they actually make contact with the cathode. By the use of anodes, consisting of round rods placed side by side to form a plate, and cathodes of finely corrugated, electrically conductive card-board or water-proofed felt, the area of electrode surface in a box of given size is more than 100% greater, and the solutions in passing between the uneven surfaces of the electrodes will be more diffused and less liable to track than is the case where the electrodes are ordinary flat sheets. The cathodes require to be made sufficiently strong to prevent warping, and this can be done by running a wire around the sides and bottom, the ends of the wire being bent over so as to dip into the mercury grooves to make electrical contact. An additional advantage in having the cathodes corrugated is that the precipitated metal will not peel off and touch the anodes, causing short circuits. If the cathodes are quite rigid the distance between anodes and cathodes can be reduced to about $\frac{3}{8}$ in., the clearance required being only enough to prevent the accumulation of precipitate between the plates. The rods composing anodes are fixed together firmly. Kerosene is poured into the box to form a layer about $\frac{1}{2}$ in. deep on the top of the solution, to prevent the accumulation of froth and the escape of hydrocyanic acid.

Precipitation and subsequent regeneration of KCN depends upon the fact that when the metal-bearing molecules in their passage through the box come in contact with the cathode they are immediately decomposed, the metal adhering to the cathode or falling to the bottom of the box, where it remains until the clean-up. The energy imparted to the liberated cyanogen is expended partly in forming compounds with free alkali, and partly in the decomposition of water and formation of hydrocyanic acid. Where the anode used has not a high resistance to oxidation, most of the liberated cyanogen would be lost in the formation of compounds with the anode material.

The total cost of a plant to treat 300 tons of solution per 24 hours would be about £500, the attention required would be negligible, and about 12 b.h.p. would be required to furnish the necessary current. A small recovery in cyanide and the higher extraction from cleaned solutions would soon return the cost of the plant. Attendance, power costs, and cathode renewals would not be as great as the cost of attendance, zinc, etc., in a zinc precipitation plant. In the former plant the copper also would be recovered.

In some cases it would appear to be an advantage to use a combination of zinc and electrical precipitation, the solutions passing first through the zinc boxes, where the gold would be saved, the function of the electrical boxes being to purify and regenerate the solutions before returning them to the vats. On account of the difficulty in getting complete precipitation of the metals without excessive cathode surface in an electrical precipitation plant, a judicious combination of zinc and electrical precipitation in the cyanidation of auro-cupriferous ores is almost certain to show greater economy than either method alone.

The latest process claiming to effect a saving in cyanide and a good extraction from raw sulphide or sulpho-telluride ores has recently been patented by J. C. Clancy and is now in operation at the 100 ton plant of the Ajax Gold Mining Co. at Victor, Colorado. This process depends upon the constant formation by electrolytic action of cyanogen iodide in the pulp during agitaton. Mr. Clancy claims to have reduced the consumption of cyanide to mechanical loss only. I have conducted a number of experiments in accordance with Clancy's directions, but up to the present time the results have not been uniform, and trials of the process on a larger scale are now being carried out. In Australian mining at the present time more depends on the discovery of some process which will profitably extract the metals from poor sulphide ores than on the discovery of new mining fields, and the success of the Clancy process is greatly to be desired. Coöperation between metallurgists exploiting the electro-chemical processes would probably bring success where isolated experiments are certain to fail."—B. GEORGE NICHOLL, *Mining and Scientific Press*, March 16, 1912, p. 426. (H. A. W.)

MEASURING AIR QUANTITIES.—"The uses of compressed air in connection with treatment plants are many, and it is necessary in view of power economy, to be able to estimate the quantities employed in different processes, in order to make comparisons with other methods of doing similar work. Meters can be obtained for measuring fairly large quantities of air at considerable pressures or velocities, also delicate and expensive instruments for dealing with small quantities and low pressures, but what is required is a cheap, simple and strong apparatus suitable for actual working conditions. In some cases, where the only obstruction to the free passage of air is a fluid of definite density, from calculations based on pressures and densities of fluids, sufficiently reliable results can be obtained. But often the obstructions are numerous and of unknown quantities, and to meet such a case (in connection with slime pulp aeration experiments), the writer evolved the method described below, and set out diagramatically in the following sketch, viz., that of substituting temporarily an equal flow of air under conditions which allow an easy measurement, in place of the ordinary supply. This method has been in use here for several months, and would, with slight modifications, measure any quantity of air flowing, provided that the supply pressure is constant during the test, and the flow regular for the few seconds necessary to obtain the reading of the first gauge (B), which flow is then measured at leisure. The only disadvantage is that the flow of air must be diverted to obtain the quantity measurements but half a minute is usually sufficient for this purpose. In one case, where it was desirable to keep a constant supply of air in a vat, a duplicate pipe was run from the point J to K, with a stop-cock so placed that this pipe supplied the vat only while the regular supply was being measured. The arrangement, which is simply the addition of a few fittings of the same size as the ordinary supply pipe to it, is as follows :—

A is a valve regulating the air to the required working pressure.

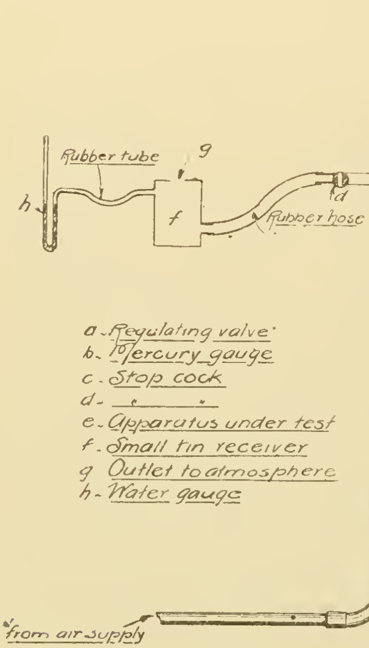
B is a mercury gauge a few feet from A.

C and D are stop-cocks, E the apparatus under test ;

F a small tin receiver connected by rubber hose to D ;

G a round orifice in a tin plate for air outlet, and H a water gauge with a range of about 12 in. to allow for occasional rushes of air while manipulating the cock D.

A low pressure supply being required at 'e' (about 5 lb. per sq. ft.) and the cock 'c' being opened and 'd' closed, the regulating valve 'a' is opened until the gauge 'b' indicates the required pressure. (It is important not to touch the valve 'a' again during the test.) This point on 'b' is marked, and to measure the quantity flowing at this instant, the cock 'd' is opened and 'c' closed; 'd' is then gradually closed until the gauge 'b' reads the same as before, when the total substituted resistance to the free flow of air is equal to that of the apparatus being tested, the pipe 'a' to 'i' in both cases acting as an intermediate receiver with constant in and out flow as proved by the gauges. The height of the water gauge is now taken, and the area of the outlet orifice noted; the cock 'e' is opened and 'd' closed, the gauge 'b' reading the same as before. This is a check on the constancy of flow during the testing period. The velocity through the orifice 'g' is that due to a 'head' of air equivalent to the pressure on the water gauge and is



A.—Diagram illustrating Method of Measuring Low Pressure Air Quantities.

[*] $\sqrt{2\ g\ h}$ = feet per sec.

'h' being the height of the motive column (head) in feet. (For air, the formula holds good only for small differences of pressure on the two sides of the orifice 'g.' Reducing to inches of water and simplifying, this becomes

[*] $\sqrt{\text{in. water}} \times 66{\cdot}1$ = feet per sec.

The co-efficient of flow for such an orifice being 0·64 owing to the contraction of the flow near the edges the actual mean velocity over the fall size of the opening becomes

$\sqrt{\text{in. water}} \times 42{\cdot}3$ = feet per sec.

In the experiments referred to, the effective velocities for an orifice of 1 sq. in. area, at pressures of up to 4 in. water gauge were tabulated, with the corresponding discharges in cubic feet per minute (free air). All subsequent readings for different sizes of orifice were simply referred to this table. Removable screwed caps were used on the receiver with orifices of 0·125, 0·25, and 0·5 sq. in area, these being simple fractions of 1 sq. in. These different sizes were used to keep the water gauge readings within reasonable limits, and covered all measurements up to 7 cub. ft. per min., with ⅜ in. pipe and fittings. Pipe sizes do not come into the calculations provided that they are large enough to carry the quantity of air, and the receiver F need only be of sufficient size to reduce the kinetic action of the air to a negligible quantity, and give a true reading on the water gauge. In the above case it was a tin of about 2 pints capacity, and for air lifts, consuming about 50 cub. ft. per min., an ordinary kerosene tin, with interchangeable orifice plates with openings of 0·5, 1, 2 and 4 sq. in. area, was used, pipe fittings in this case being 1 in. In all cases, the compression in the receiver is so slight that all calculations are as for free air. With the apparatus described above no difficulty was found in measuring quantities anywhere between 0·5 and 60 cub. ft. per min. By substituting a high pressure gauge at B, readings can be taken at any pressure, and in any case a little experience soon gives the most suitable size of orifice to use."—G. S. Weymouth, *Monthly Journal of the Chamber of Mines of Western Australia*, Jan. 31, 1912, p. 351. (H. A. W.)

MINING.

Accidents in Coal Mines.—"In reply to a question asking how many workman were killed or injured in the coal mines of the United Kingdom during each of the past ten years, the Home Secretary has furnished the following figures:—

Year.	Number killed.	Number injured as reported to the Inspectors.	Number injured and disabled for more than 7 days.
1902	1,024	3,745	—
1903	1,072	3,822	—
1904	1,055	3,754	—
1905	1,159	3,646	—
1906	1,142	3,839	—
1907	1,245	5,892	—
1908	1,308	5,860	141,851
1909	1,453	5,859	153,306
1910	1,775	5,737	159,042
1911	1,259	(Not yet available.)	

The figures in the last column were obtained for the first time in 1908 under the provisions of the Notice of Accidents Act, 1906, and cannot be given for the preceeding years. The table speaks eloquently of the terrible dangers of the collier's life. On an average some four men are killed every day, and five hundred disabled for more than seven days."—*Journal of the Royal Society of Arts*, March 22, 1912, p. 498. (J. G.)

Estimation of Oxygen in Mine Air.—"This is a method for the estimation of oxygen in a fiery

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[*] The weight of 1 cub. ft. of air at 62 deg. F. = 0·0671 lb.
,, ,, 1 ,, water at 62 deg. F. = 62·5 lb.
∴ the ratio of water column to air column is

$$1 : \frac{62{\cdot}5}{0{\cdot}0761} = 1 \text{ to } 821$$

mine where the naked flame method cannot be used.

The writer has found that when the percentage of carbon dioxide in the air is known, the oxygen can be calculated in the following manner:—

From the normal percentage of oxygen in the air is deducted twice the percentage of carbon dioxide estimated in the mine sample. The normal percentage of oxygen being taken as 20·9.

It will be seen from the following results that the agreement between direct analysis of oxygen, and percentage calculated from percentage of CO_2 is reasonably close.

	CO_2.	O_2 (Analysis).	O_2 (Calculated).
1	·30	20·28	20·3
2	·16	20·55	20·58
3	·37	20·05	20·16
4	·27	20·32	20·36
5	·05	20·80	20·80
6	·40	20·19	20·10
7	·50	20·06	19·90
8	·32	20·33	20·36
9	·34	20·22	20·22
10	·58	19·70	19·74
11	1·32	18·55	18·26
12	1·84	17·87	17·22

—M. E., *Colliery Guardian*, March 8, 1912, No. 2671, p. 480. (T. D.).

RECORD SHAFT SINKING.—"The south-east shaft of the Government Gold Mining Areas (Modderfontein) Consolidated was sunk 233 ft. during March, which is certainly a record for the Rand, and we believe the world's record. Great credit attaches to Mr. Martin H. Coombe, the manager, and to Messrs. Lowson, Readhead, and Robar, the sinkers, for this remarkable achievement. The previous record was 213 ft. at the Modder Deep, beating that of the south shaft of Brakpan Mines in July 1907; these shafts are of three and seven compartments respectively. During the quarter 583 ft. were sunk at the State Mines, which we believe is also a record; at any rate, it compares with 532 ft. at the Brakpan Mines. It is usually considered that record sinking can only be attained in the Kimberley slate, and it has been said that a monthly record depends largely on the date at which the Kimberley slate is entered so that the greatest possible distance during the calendar period can be sunk in that formation. This was, however, not the case at the State Mines, less than 80 ft. being in the slate shales.

The shaft in which the calendar month's record sinking has been made is of seven compartments, the excavations being 45 ft. by 10 ft., equal to 450 square ft. area, or 37·5 tons to the foot of shaft in the solid. The total tons tallied for the month as hoisted for the 233 ft. sunk were 8,155, equal to 35 tons to the foot, which comes very close to the estimate. The sinking was done through quartzite and slate shale, 155 ft. of quartzite and 78 ft. of shale being pierced during the month. It is noteworthy that only one-third of the distance was in the Kimberley shale; previous records have been almost entirely in that formation. The record was made from 1,738 ft. to 1,971 ft. Natives do all the hand-drilling and shovelling, 82 natives with a leading sinker and a white assistant forming the shift's sinking crew. Eight-hour shifts are worked seven days a week. The majority of the boys drill double hand, using 7-lb. hammers, the end benches only having 'single-hand' boys at work on them. No particular trouble or care is given to a 'sink.' The shaft being laid out with its long axis on the dip, advantage is taken of the bedding planes, the great majority of the holes being inclined to the north against the strata.

Sinking with a skip installation, one of the main objects striven for is a flat bottom, an ideal skip sinking shaft bottom having a bench only at each end. The reason of this is the greater facility for loading the broken rock into the skips and getting each skip in its turn into the bottom at approximately the same level. It is obvious that if the 'benching' system were in use in a skip shaft with the engine running double and the skips landing at the bottom at different levels innumerable difficulties would be encountered such as tipping difficulties, risk of kinking the rope, either running the skip away empty or taxing the boys unduly by shovelling the rock three or four feet higher into the skip; in fact, many unforeseen troubles would be encountered which would cause delay and probably accidents. Blasting takes place every eight hours. From 40 to 45 holes completes a 'round' for a shift and 'squares' the shaft. These holes are drilled from 3 ft. 6 in. to 5 ft. in depth. Four are drilled 'abreast' for the length of the shaft. The side holes are given from a plumb line hung from the bottom sett. Gelatine is used $\frac{7}{8}$ inch diameter; $66\frac{2}{3}$ per cent. used is of Cape Explosives Company manufacture and $33\frac{1}{3}$ per cent. of Modderfontein. Bickford Smith's '27 Special' fuse is used, and Edgar Allen and Company's Green Label $\frac{7}{8}$ inch steel. No pump is installed in the shaft below 600 ft., the water, 30 skips a shift, being handled with the sinking skips from 'rings' at various points.

Four timbermen are employed in each of the four shafts which are being sunk on the State Mines. Two men and six boys being 'bottom sett' men, one man and four boys are on the hitches, bearers and permanent guides, and one man and one boy on the brattice. All timbermen work morning shift only from 7 a.m. to 3 p.m., and only for very urgent reasons is overtime allowed. After the sinking shift is in the bottom and hoisting men has commenced, the two bottom sett men go down with their lines, stage planks, blocks, wedges and 'boys.' Their first duty is to block the next to the 'bottom sett,' using the bottom sett to carry their stage planks. They are not allowed to interfere with the skips once they have reached their place of work, until the sinker has finished 'cleaning up,' had his drills down, and started his 'boys' drilling. Then the engines are handed over to the timbermen, who take them to 'hang' the next sett. From the time the banksmen shackles on the first pair of wall plates until the sett is hung, filled in, tightened up and temporarily blocked, and the timbermen are up to surface with their stage planks and 'boys,' is an hour to an hour and a half. Timbermen are not allowed to deal with 'tight' ground on the timber. The sinker is responsible and is compelled to take it down. Knowing this he is careful with his lines and little is ever heard of ground being 'on.' If a sett of permanent guides is required or a sett of bearers to be taken down, as soon as the timbermen have hung their wall plates and had their 'filling in' timber sent down they hand the engine over to the other man who goes on with that particular work. 57 skips, or 114 tons of rock, have been hoisted from the bottom, one sett blocked, another hung and filled, four 30 ft. guides hung and bolted up, the shaft drilled over in the usual manner, and blasted a quarter of an hour before the end of the shift. All hitches are cut, bearers installed and timber dropped whilst the shift is at work in the bottom. The shafts carry six 8 in. and 4 in. penthouses, two, one above another in each 'dummy' compartment. Running four skips in a seven-compartment shaft gives each end and the centre compartment clear. In these the penthouses are placed.

There are 132 cubic ft. of pitch pine in each sett. The timbers are 7 ft. centres, 6 ft. 3 in. between; wall plates and end pieces, 9 in. by 9 in.; centre studdle over the joint, 10 in. by 7 in.; corner studdles, 8 in. by 8 in.; intermediate studdles, 10 in. by 4 in.; dividers, 9 in by 7 in. The guides are 8 in. by 4 in. pitch pine carried on channel brackets bolted to the dividers. One inch 'joggling' is used. Lagging, when any is used, is 12 in. by 12 in. pitch pine. Bearer setts are put in approximately every 120 ft., each bearer being made up of three pieces. Under the end pieces three 9 in. by 9 in., making a bearer 27 in. by 9 in.; under the dividers three 9 in. by 7 in., making a bearer 27 in. by 7 in Hitches are cut to a minimum of 4 in. in the solid surface. Wire plumb lines are dropped once a month to check the timber and timbermen's lines. All the framing, guides, ladders, rings, etc., are got out for the four shafts by three men by hand. To date of writing, in this shaft, 64 setts of 7 ft. centre timber has been put in position in 61 consecutive days. This in itself constitutes a record, and is one that will be difficult to beat. Six hanging bolts a-side are required, three for each half plate. 'Waverley' 1⅛ in. iron is used for the bolts. The wall plates are in two halves, and are simply butt-jointed on a divider, no plating or bolting being used. The length of the two wall plate halves is respectively 22 ft. 9½ in. and 20 ft. 8½ in., making a total length of 43 ft. 6 in. The dividers are 6 ft. 6 in, giving a shaft width over the timber of 8 feet.

Compartment sizes are as follows; The south or pump compartment is 8 ft. 6 in. by 6 ft. 6 in. in the clear. The other six compartments are each 6 ft. 6 in. by 5 ft. in the clear. The pump compartment is divided up as follows: 2 ft. 8 in. ladderway, 3 ft. 6 in. for repairing cage, the 'runners' 4 in. and 4½ in. being bolted to the wall plates, and 2 ft. 4 in. for pipe bearers and columns. The power cables are carried down in the space between the wall plates and the repairing cage ends. All lights used in the shaft are 'carbide.'

No fan installation or pipes are in use. A wooden chimney is taken up above the headgear the entire area of the pump compartment. A ⅞ in. brattice between the pump compartment dividers and 'scribed' to the ground, is carried down with the timber. This gives a natural draught of from 20,000 to 25,000 cubic ft. of air per minute passing over shaft bottom. Ten minutes after blasting the bottom is clear of smoke and fumes.

The shaft is equipped with a modern steel headgear, 110 ft. from collar to platform, covering of course all the compartments. Two double drum engines are in use, one a geared Robey 10½ in. by 33 in. cylinder, one a Fraser and Chalmers direct 22 in. by 48 in. cylinder. All the drums are 8 ft. diameter. The boilers are Babcock and Wilcox Dutch oven type. A 50 drill Fullerton compressor is also erected, but it is not being used. The winding ropes are 1⅛ in. diameter. Sinking skips of 45 cubic ft. capacity are used. White and native change houses, workshops, etc., are of the most modern design and construction. Massive concrete collars have been put in, built up from the solid rock to from 5 ft. to 6 ft. above the surface. This shaft is one of four of the same plan. Sinking on the property, which probably constitutes a record for the world; the average sinking for March in the combined shafts was 694 ft., an average of 173 ft. 6 in. for each shaft. The total tonnage hoisted was 24,290.

The officials in direct touch with the shaft are mine captain, engineer and foreman boilermaker. No shift bosses or foremen timbermen are employed. The opinion is held that a shift boss hinders and acts as a stumbling block in the way of the man who is doing the work. The men are of the best type and work on the bonus system. The greater the footage the greater the cheque and the cheaper the shaft per foot sunk All men directly connected with the sinking join in the bonus list. Engine drivers are eight hours shift, six to a shaft. One and all know and observe the system installed, the whole thing works like clockwork and the result is efficiency of the highest order and good sinking. The State Mines is breaking, hoisting, and putting on the dump from the bottom of the sink-shafts rock at a cheaper price per ton than the great majority of the producing mines can stope and hoist it. That is the difference in efficiency, though concentration of work has something to do with it. The highest tonnage broken per hole averaged over the month was 2·23, the gelatine consumption being ·457 lb. The average number of natives per shift was 82."—*S. A. Mining Review*, April 1912, pp. 69-70. (A. R.)

MISCELLANEOUS.

THE PRODUCTION OF OXYGEN IN THE WORKSHOP.—"The precise nature of bleaching powder (chloride of lime) has not been satisfactorily demonstrated; but it is generally regarded as a chemical combination of calcium hypochlorite and calcium chloride, which may be expressed by the chemical formula: $CaOCl_2$. It also contains small quantities of basic calcium hypochlorite $CaOCl.OH$ and basic chloride $CaCl.OH$, and there is always present an excess of lime and a certain quantity of water. It is manufactured by electrolytical processes in various strengths containing anywhere between 20% and 40% of available chlorine.

That bleaching powder could be used for the production of oxygen has been known since 1829, when Dingler first obtained this gas by heating chloride of lime with certain metallic oxides (cobalt, nickel and copper). Later the wet process of decomposing bleaching powder was resorted to and various methods were subsequently proposed by Mitcherlich, Stalba, Winkler, Fleitmann, and lately by Jaubert.

If a clear and concentrated solution of the powder, to which has been added, a small quantity (0·1 = 1%) of the oxides of cobalt, nickel, copper or iron (or the salts of such oxides), is gently heated to 70° to 80° C., 158° to 176° Fahr., oxygen gas is given off in a slow and regular stream. It is not necessary to heat any further, since when once started the reaction continues until all the oxygen from the hypochlorite has been disengaged.

Instead of a solution, a paste made of bleaching powder and water may be used. In that case, however, it is necessary to add some paraffin oil, or a piece of paraffin. The oil or the molten paraffin, floating on top of the pasty liquid, prevents froth ing of the mixture, which otherwise would occur. The oxygen escapes with effervescence in quantities equivalent to the active chlorine.

Although a number of salts and oxides (ferric hydrate or manganese dioxide may also be used) may serve as a contact substance, it seems that cobaltic oxide (cobalt peroxide) is the one which is most prompt and energetic in its action. and that relatively small quantities of it are required for the reaction.

The oxygen obtained is quite pure; the free lime which is always present in the bleaching powder sufficiently prevents the disengagement of chlorine.

A modification of this process is the one proposed by Fleitmann. A suitable vessel is filled with milk of lime to which has been added some cobaltic oxide. A constant stream of chlorine is then allowed to pass through the fluid, causing the disengagement of a like amount of oxygen. The latter will be found to be almost free from chlorine, even if the gas should be introduced in a rapid stream. Only when the calcium hypochlorite is entirely decomposed will the oxygen be mixed with chlorine, which can easily be removed by passing it through a solution of caustic soda. This method can be used to advantage only where manganese dioxide and muriatic acid (the essential chemicals for generating chlorine) can be obtained cheaply enough.

The production of oxygen from bleaching powder has at various times been the subject of attempts to adapt it to industrial requirements or to the requirements of the individual consumer of oxygen. A simple generator for that purpose had been proposed fifty years ago by Varrentrapp. An ordinary acid-carboy is partly filled with the chloride of lime mixture and is then closed up by a tight-fitting plug, having an exit pipe of sufficient width. This carboy —basket and all—is placed into a wash kettle or the like, and the water is heated to boiling. The contents of the carboy will yield 1,000 to 1,200 liters (35 to 43 cu. ft.) of oxygen. Fleitmann recommended the use of steam-jacketed boilers, where the gas can be obtained under a higher pressure and also in large quantities.

Jaubert recommends the following: 100 parts of the dry bleaching powder are mixed with 1,000 parts of cold water and allowed to settle. The clear liquid is decanted into the generator. To this is added, while stirring, a concentrated solution of 8 parts of crystallised cobaltic chloride. The reaction is said to last five minutes, while it would take from 1 to 2 hours if but 2 parts of cobalt salt was used. The only objection that could be raised against this method would be the rather high cost of the catalytic substance.

It may also be mentioned that in most of these methods some of the calcium hypochlorite is liable to be transformed into the chlorate, which, of course, lessens the output of oxygen. Three pounds of bleaching powder titrating 35% of active chlorine should yield about 2½ cu. ft. of gas.

In 1903 Geo. F. Jaubert succeeded in finding new and extremely active catalytic substances which were much cheaper than cobalt salts and which cause a rapid generation of oxygen even in the cold. Such substances are the sulphates of iron and copper when used in combination; if used alone the decomposition is an extremely slow one, even if heat is employed.

The proportions as proposed by Jaubert are 60 parts of bleaching powder mixed in a suitable vessel with 350 parts of water. This mixture is poured into the generator. To start the reaction a solution of 12 parts of sulphate of iron (or manganese) and 3 parts of sulphate of copper in 50 parts of water is slowly added, while stirring. It is probably better to mix the copper sulphate solution with the bleaching powder and add the iron salt solution when required.

Several welding shops in this country have adopted this or at least similar methods for producing oxygen. As a modification of this process the dry bleaching powder (220 lb.) may be mixed with dry sulphate of iron (33 lb.) and sulphate of copper (11 lb.). This powder evolves oxygen when placed in water. "Epurite" is a trade name for such a combination. The gas can be generated in an apparatus similar to calcium carbide generators. The generators should be provided with suitable stirring devices (paddles) and suitable apertures for introducing the chemicals and removing the residue. The exit tubing is connected with a scrubber to remove any impurities, and from these leads to the gas holder.

Where a larger supply of oxygen is required it would be preferable to use cast-iron generators, topped by a dome or a "gooseneck," to which is attached the exit pipe; or a lead-lined double-jacketed copper kettle may be used. Heating, although not necessary, hastens generation; the temperature should, however, not exceed 80° C. The feed pipe for the solution of sulphates should reach more than half way down the "milk" of bleaching powder; it will then, at the same time, act as a safety pipe.

It may also be mentioned that the catalytic agent may be suspended in cylinders made of porous material which are immersed in the liquid. This modification had first been proposed by Jaubert, but a patent on a generator of this type has only recently been granted to M. Mauran and J. H. MacMahon (U. S. Pat. 1,001,873, Aug., 1911). In the latter case the metallic oxides are mixed with Portland cement or baked clay and are used in connection with "lavoisite."

The oxygen may also be produced under pressure, as the latter does not prevent decomposition of the bleaching powder. E. H. Kirkpatrick, who has experimented along these lines, found that evolution of oxygen takes place at a pressure of 10 and even 15 atmospheres (150 to 225 lb.)

If the dry bleaching powder is mixed with small quantities of slaked lime, and heated to redness, oxygen is also given off (H. Deville).

Bleaching powder has no dangerous properties, therefore large quantities may be stored. All the work connected with the generation of oxygen can be done by unskilled labour. The chloride of lime usually titrates 33 to 35% of "available" chlorine. It is sold in barrels, drums or casks at about 1½c. per lb. The cost of oxygen does not exceed 2c. per cubic foot; this process, therefore, compares favourably with the chlorate process.

It may be added that instead of bleaching powder any other alkali hypochlorite may be used, or the solutions of these hypochlorites (Javelle water, Labarraque water)."—Aug. S. Neumark, *Metallurgical and Chemical Engineering*, February, 1912, p. 109. (F. W. W.)

Electric Light for Sterilising Milk.—"Attention has recently been called in Holland to the effect of ultra-violet beams on bacteria, and to the fact that such beams are abundantly developed by mercury incandescent lamps; moreover, according to the American Consul at Amsterdam, it has been stated that through this medium milk may now be sterilised in a few minutes. An apparatus has been constructed, it is explained, whereby the milk flows in a thin stream along an electric light. Demonstrations were first made with water infected with different kinds of bacteria, and it is said that the water was purified in a few minutes without appreciably increasing its temperature. The result is attributed to the quality of the ozone formed under the influence of the light, but the demonstrations must be conducted where there is sufficient room for the light to burn freely."—ANON, *Journal Royal Society of Arts,* March 8, 1912, p. 467. (J. G.)

COPYING OLD MINE PLANS.—"It occasionally happens in connection with the re-survey of an old mine, or in development work involving the examination of old plans, that copies of plans are required the originals of which are so faint and the paper on which they are drawn is so deteriorated by age that the task of copying is a most tedious or almost impossible one.

In such a case the following method was adopted some few years ago to reproduce a large and intricate mine plan which was 50 years old and a copy of which was required. This plan had been badly kept and as a result mildew and mould had played havoc with it. The ink lines were so faint and obscure that they could not be seen through the best tracing cloth or paper that could be obtained, and it therefore, at first appeared that to obtain a copy of the plan was impossible.

The method finally adopted, however, and one which was successful was to spread over the plan a sheet of transparent celluloid so that the whole plan was covered with a layer of this material. On this sheet all the details were readily traced, as the sheets were absolutely transparent, by using Higgin's waterproof blue ink. The celluloid sheet was then removed from the plan and placed on white paper, and the drawing so obtained enabled a finished tracing on cloth to be made with ease."—*Engineering and Mining Journal,* March 23, 1912, p. 589. (A. R.)

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Abstracts of Patent Applications.

(C.) 675/11. Wilbur Alson Hendryx. Improvements in dewatering devices or means for separating solids and liquids. 28.12.11.

This application refers to improvements in dewatering devices or means for separating solids and liquids. It comprises in combination a settling vessel, a pair of oppositely rotatable screws placed in a common trough or casing and having no dividing wall or partition between them, the said trough or casing being in communication with the bottom of the vessel. The lower end of the screw comprises double threaded sections spaced apart and a plurality of abutting single pitch segments secured upon a shaft. The vessel comprises also a feeding screw and means for preventing accumulations of the settled material, comprising rotatable arms connected together, the arms or the means connecting them being provided with plough shares or other equivalent for the purpose above specified. Means are also provided whereby the feeding screw can be driven at different rates of speed.

(C.) 668/11. Josef Carl Steinebach. Device for automatically stopping electric hoists. 18.12.11.

This specification relates to an invention for automatically stopping electric hoists when the speed of such hoists during the slowing down periods differs from the normal speeds at that period of the operation.

The device consists in the use of a small 3-phase generator, driven in any convenient manner from the main winding motor, so arranged that the current from this small generator energises an electro magnet which breaks the circuit of the no-voltage release coil on the main 3-phase winder switch.

A resistance is fixed in the circuit of the small generator and is controlled, either by the cages or by means of the travelling nut of the depth indication apparatus, so that as the cages reach the stopping places the resistance is automatically cut out, which would energise the electro magnet opening the no-voltage release circuit, if, at the same time, the small auxiliary generator was not slowed down to a speed sufficient to prevent the current in the control circuit being large enough to operate the said magnet.

Means are provided by means of a throw-over switch in the stator circuit of the small auxiliary generator to ensure that the voltage of such generator is always in the right direction, and a hand operating switch is included for putting an auxiliary resistance in the control circuit when rock or men are being hauled.

(C.) 74/12. Henry James Rees (1), Henry Livingstone Sulman (2). Improvements in or relating to the deposition of metallic salts from solutions containing them. 2.2.12.

This is a process for depositing metallic salts, especially of copper and zinc, by passing a thin film of a solution of these salts over heated baffle plates, preferably placed in a tower. The heat is supplied by a coke stove.

(C.) 90/12. James Millar Neil. Process and apparatus for atomizing metals, minerals and the like. 9.2.12.

This specification describes a process of atomizing metals, minerals, or the like, consisting in subjecting a jet of the molten material to the action of a blast or jet of an atomizing agent, such as air, steam or gas, heated to a temperature approximately equal to or above that of the melting point of the material being treated, for the purpose specified.

(C.) 135/12. Hans Charles Behr. Improvements relating to percussive apparatus. 2.3.12.

This application refers to a new method of operating a pneumatic stamp for crushing ore.

The stamp piston is actuated by compressed air supplied by a separate piston, termed the operating piston, and driven by a belt.

An air reservoir and a number of valves are introduced to govern the distribution of the air and secure the proper travel of the stamp piston.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 165/12. Oscar Birger Carlson. Improvements in the method of manufacturing combustibles for explosives. 22 3.12.

(C.) 166/12. Ernst Ritter. Improvements relating to the construction of hollow beams and the like. 22.3.12.

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(C.) 170/12. T. P. Jordeson & Co. (1), August Marius Kjaersgaard (2). A refrigerating machine combined with an ice cupboard. 22/3/12.

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(C.) 180/12. John George Robinson. Improvements in and relating to steel superheaters for locomotives and other boilers, (transverse compartment header). 26.3.12.

(C.) 181/12. John George Robinson. Improvements in and relating to steam superheaters for locomotive and like tubular boilers, (two compartment header.) 26.3.12.

(P.) 182/12. Harry Templeton Villiers. Improvements in distempers and the like. 27.3.12.

(P.) 183/12. Edgar Percy Hedley. Improvements in torches suitable for fuse lighting. 28.3.12.

(P.) 184/12. Edgar Percy Hedley. Improvements relating to torches suitable for fuse lighting. 28.3.12.

(P.) 185/12. Archibald Thorburn. A mechanical roaster for a complete roasting off of sulphur and other refractory elements from pyritic ores. 28.3.12.

(C.) 186/12. Botho Schwerin. Improved process for treating finely sub-divided substances by electric osmose. 29.3.12.

(C.) 187/12. Henry Squarebrigs Mackay. Process of extracting metals from ores. 29.3.12.

(C.) 189/12. W. M. Still & Sons Ltd., (1) Andrew George Adamson (2), Arthur James Douglas Kitson, (3). Improvements in vapour burning lamps. 29.3.12.

(P.) 192/12. Charles Harold Taylor (1), Maurice Edward Marsden, (2). Improved drill bar and bracket. 29.3.12.

(P.) 194/12. C. H. Rotherham. Improvements in tube mill linings 2.4.12.

(P.) 195/12. Ernest Holdroyd Martin. Improvements in pebble feeders for tube mills. 2.4.12.

(P.) 196/12. Oswald Victor Whittle, (1), James Houston (2). Improvements in electrical signalling systems. 2.4.12.

(P.) 197/12. Andrew Harvey. Improvements in tube mill linings. 2 4.12.

(C.) 198/12. Benjamin Raffael (1), James Kerr Fulton (2). A new or improved composition or preparation for lubricating gear wheels and the like. 3.4.12.

(P.) 199/12. Rollo Bowman Ballantine. Cleaning or sooting device applicable to tubular boilers and the like. 4.4.12.

(P.) 200/12. Albert Holdsworth. A new or improved turbine driven tube cleaner. 4.4.12.

(C.) 201/12. John Galileo Barclay (1), Archibald Newton Barclay (2). Improvements in connection with the raising and lowering of pit cages.

(C.) 202/12. John Lang (1), John Johnston (2). Improved roller supports. 6 4.12.

(P.) 203/12. Daniel Henry Essler. Wet process for the separation and extraction of the gold contained in antimony ores. 6.4.12.

(C.) 205/12. Michael Setter (1), Alexander Alsworth Keith (2), Edward Arthur Mellinger (3), Talbot Grant Martin (4). Improvements in or relating to telephone systems. 6.4.12.

(P.) 206/12. James Dunlop Wilson. Improvements in fuse lighters. 9.4.12.

(P.) 207/12. John Foye (1), Henry Egbert Moore (2) and Robert Boyle (3). Improvements in the treatment of refractory ores. 9.4.12.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Alston, R. A., *l/o* London ; Vogelstruis G. M. Co., Ltd., P. O. Box 115, Roodepoort.

Anderson, P. von M., *l/o* Johannesburg ; Princess Estate and G. M. Co., P. O. Box 112, Roodepoort.

Ball, H. S., *l/o* Johannesburg ; Modder "B" G. M. Co., Ltd., P. O. Box 14, Brakpan.

Brown, G. D., *to* New Fortuna G. M. Co., Ltd., P. O. Box 124, Barberton.

Cindel, F. W., *l/o* Cleveland ; Bantjes Consolidated G. M. Co., Ltd., P. O. Box 2, Florida.

Clennell, J. E., *to* c/o Campania Minera, El Barreno y Anexas, San Luis Potosi, Mexico.

Cockburn, P. A., *to* King Edward Claims, via Susakaas, N. Rhodesia.

Dowsett, C. W., *l/o* Germiston ; Brakpan Mines, P. O. Box 3, Benoni.

Gerber, O. A., *to* P. O. Box 103, Crown Mines.

Howe, Ben, *l/o* London ; Gwalia Consolidated Mines, Murchison District, W. Australia.

Linton, R., *l/o* Mexico ; 1,005 Trust and Savings Buildings, Los Angeles, Cal., U.S.A.

Mackey, A., *to* Thornhill, 9th Avenue, Mayfair, Johannesburg.

Mather, T. W., *l/o* New York ; 308, Chestnut Street, Pacific Grove, California, U.S.A.

Moir, Dr. J., *l/o* Johannesburg ; c/o Mr. John Hall Barron, Barrister, 45, Parliament Street, London, S. W.

Neal, W., *to* c/o El Favor Mining Co., Hostotipaquillo, Jalisco, Mexico.

Oliver, E. L., *to* 503, Market Street, San Francisco, Cal., U.S.A.

Patrick, W., *l/o* Sabie ; Elandsdrift Mine, P. O. Rose Hill.

Pollard, B. E., *l/o* Knights ; Quest G. M. and Development Co . Ltd., P. O. Box 113, Klerksdorp.

Quince, W. A., *l/o* Germiston ; Sub-Nigel, P. O. Box 50, Nigel.

Smith, W. J. W., *to* Wanderer Mine, Selukwe, Rhodesia.

Stephens, A. H., *l/o* Benoni ; 46, Alma Road, Rosebank, C.P.

Summers, E., *l/o* Johannesburg ; Witwatersrand G. M. Co., P. O. Box 1, Knights.

Thompson, R. R., *l/o* England ; Bongwelli Camp, Delimi River, Naraguta, N. Nigeria.

Thorpe, W. E., *l/o* Lincoln, England ; Princess Estate G. M. Co , P. O. Box 112, Roodepoort.

Watson, F. W., *l/o* Germiston ; c/o Mr. Harold Watson, 121a, St. Vincent Street, Glasgow, Scotland.

Webb, W. M., *l/o* Roodepoort ; P. O. Box 5, Kingville. Ontario, Canada.

White, F., *l/o* Rhodesia ; c/o Messrs. Percy Tarbutt & Co., 18, St. Swithens Lane, London, E.C.

Willcox, A., *l/o* Rietfontein ; Ginsberg G. M. Co., P. O. Box 16, Knights.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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Vol. XII. JUNE, 1912. No. 12.

Proceedings
AT
Annual General Meeting, June 22nd, 1912.

The Annual General Meeting of the Society was held in the South African School of Mines Building, on Saturday, June 22nd, 1912, Lt.-Col. C. B. Saner (President) in the chair. There were also present:—

48 Members: Messrs. W. R. Dowling, E. J. Laschinger, H. A. White, R. Allen, R. G. Bevington, W. A. Caldecott, A. F. Crosse, J. Gray, A. McA. Johnston, A. Richardson, Prof. G. H. Stanley, J. E. Thomas, J. R. Williams (Members of Council), H. R. Adam, S. Beaton, A. J. Bowness, J. Q. Braidwood, J. M. Campbell, J. Chilton, R. V. Crake, J. M. Dixon, J. Donaldson, L. L. Ellis, R. Gascoyne, W. W. Gordon, A. J. Johnson, G. A. Lawson, W. Lea, L. Marks, S. Newton, J. M. Phillips, F. D. Phillips, W. S. V. Price, J. F. Pyles, G. A. Robertson, A. Schwarz, A. L. Spoor, A. R. Stacpoole, S. H. Steels, R. Stokes, J. A. Taylor, W. A. C. Taylor, A. Thomas, W. E. Thorpe, C. Toombe, F. Wartenweiler, P. White and A. Wilkinson.

28 Associates and Students: Messrs. A. R. Adams, N. S. Arnot, E. G. Baskett, W. M. Coulter, J. Cronin, J. Davidson, M. Dennehy, I. Dicks, O. A. Gerber, M. Green, W. J. R. Hunter, W. Human, A. King, H. L. Krause, L. Y. Leyson, G. F. Matthews, E. C. Maunsell, F. Millar, D. Ordman, G. S. Paterson, F. J. Pooler, V. E. Robinson, H. Stadler, A. M. Thomas, D. J. Thomas, H. J. Ward, W. Waters and A. Willcox.

13 Visitors, and Fred. Rowland, Secretary.

MINUTES

The minutes of the May meeting as published in the May *Journal* were confirmed.

ANNUAL REPORT OF COUNCIL.

The Secretary read the Annual Report of the Council and submitted the statement of Accounts and Balance Sheet together with the Auditors' certificate:—

The Council has pleasure in submitting the following report upon the work of the Society during the past year.

It is regretted that the financial statement is not quite so satisfactory as last year. This may be put down to the lack of attention paid by many members to the payment of their subscriptions, as had last year's revenue been maintained, this statement would have been an excellent one. It appears necessary to continually remind some members that their subscriptions are in arrear. This should not be necessary as it is unfair to those who pay and causes a needless cost to the Society. The reason for the greater part of the decrease of over £292 in the revenue compared with that of last year may be ascribed to this.

Turning to the expenditure side of the account, the ordinary expenditure, including depreciation, again shows a decrease—this year about £50 compared with £75 last year. Several grants have been made—one of £10 10s. to the Memorial which is being raised in Amsterdam to the memory of the late Van 't Hoff, the eminent chemist, while another of £50 has been made to the Seymour Memorial Library for scientific books. It is unnecessary to again allude to the usefulness which this technical library is to the members of the Society, but it is satisfactory to learn that the Government has seen the necessity of coming to its assistance to the extent of voting £150 to its upkeep. Dr. Macaulay, "our member," is to be congratulated upon his efforts in the House of Assembly, and the Trustees also upon their work to that end in other directions.

The *Journal* shows a decrease in cost of about £180 due chiefly to there being not quite so many pages and illustrations printed, and to an increased revenue from advertisements of nearly £100. The practice adopted of writing off the value of the printed transactions in stock has been continued, and in another two years there will be no need for the large amounts to be written off annually for this purpose The statements before you show that over £163 has been written off on that account. This, together with the grants made during the year, in conjunction with the decreased revenue, more than accounts for the deficiency shown on the year's working.

The total membership at present is 1,229, an increase of 52 over that reported last year, of which the usual progressive table is given herewith.

TABLE I.

	June, 1895.	June, 1896.	June, 1897.	June, 1898.	June, 1899.	June, 1902.	June, 1903.	June, 1904.	June, 1905.	June, 1906.	June, 1907.	June, 1908.	June, 1909.	June, 1910.	June, 1911.	June, 1912.
Hon members	5	5	5	8	14	9	13	18	18	16	17	18	21	22	20	23
Life members	—	1	1	1	1	1	1	2	3	4	7	7	8	9	9	9
Members ...	49	81	101	197	281	291	441	639	680	680	683	618	630	731	711	733
Associates ...	—	—	—	180	238	243	281	338	321	314	342	357	342	402	380	442
Students ...	—	—	—	—	—	—	—	4	8	4	7	18	20	60	57	22
	54	87	107	386	534	544	736	1,001	1,030	1,018	1,056	1,018	1,021	1,224	1,177	1,229

It will be seen from the Balance Sheet that there is a slight decrease compared with last year in the cash at Bank and in hand, and a slight increase in the amount on fixed deposit (with accrued interest), which now stands at £722 12s. 7d., making the total cash assets £867 7s. 2d., which amply cover the liabilities of £377 9s. 9d.

Outstanding subscriptions, which are never brought into the accounts, amount to £811 for the current year, but since the closing of the books £131 5s. of this has been received.

Sixty-six Members, 82 Associates, and 23 Students were elected and admitted during the year, three of the members having been transferred from the Associate Roll and 10 of the Associates having previously been registered as Students.

The loss by death was less this year, six Members and two Associates having passed away. Their names are Messrs. G. O. Smart (Member of Council) whose loss was greatly felt not only by his fellow Members of Council, but also by his numerous friends along the Reef, E. E. A. Blume, James Gaze, J. N. de Jong, A. L. Mills. and Sir Julius Wernher, Bart. (Members), and H. Jansen and C. R. Johnson (Associates). Thirty-four Members, 19 Associates, and one Student have resigned, or have not confirmed their election by the payment of their fees. In view of the stringent measures taken last year in striking off the roll the names of those in arrears for two years and over, the Council considered its action would this year have been an inducement to members to pay their subscriptions without undue delay and without the expense of sending out so many reminders.

Twelve ordinary general meetings of the Society have been held during the past year, and the papers and notes submitted thereat were as follows :—

July, 1911.—"Notes on the Treatment of Mill Concentrate," by Mr. R. Lindsay.
"Experiments in the Treatment of Accumulated Ore Slime by Air Lift Agitation," by Mr. J. E. R. Adendorff.

August, 1911.—"An Electro-Magnetic Separator" (Note), by Mr. A. F. Crosse.
"A New Grading Machine" (Note), by Mr. B. V. Blundun.
"Sinking Operations at the Norma Shaft, Kattowitz, Upper Silesia," by Mr. B. C. Gullachsen.

September, 1911.—"Note on the Use of Certain Cyanicides in the Estimation of Undissolved Gold," by Mr. H. A. White.
"The Assay of Gold Bearing Cyanide Solutions by Electrolysis," by Mr. C. Crichton.

October, 1911.—"Zinc Dust as a Precipitant" (Note), by Mr. M. Thornton Murray.
"Example of Dangers in Mining" (Note), by Major A. B. Denne.
"Notes on High Duty Gravity Stamp Mills," by Mr. P. N. Nissen.

November, 1911.—"Accidents in Transvaal Mines (with special reference to Falls of Ground)," by Mr. James Chilton.
"Practical Applications of the Specific Gravity or Moisture Flask," by Mr. H. Stadler.

December, 1911.—"Notes on the Absorption of Gold by Amalgamated Copper Plates, Cast Iron Retorts, and Amalgam Trays," by Prof. G. H. Stanley, and Mr. M. Thornton Murray.

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

(INCORPORATED MAY, 1907.)

REVENUE ACCOUNT for year ending 1st June, 1912.

Dr

1910-11. £ s. d.	EXPENDITURE.	1911-12. £ s. d.	£ s. d.
	To CHARGES (including Postages, Bank		
(229 3 9	,, Charges, Insurance, etc.) ...	214 11 7	
212 14 0	,, PRINTING AND STATIONERY ...	176 5 0	
18 6 3	,, RENT AND LIGHTING	22 5 1	
423 15 0)	,, SALARIES...	440 0 0	
883 19 0			853 1 8
(24 13 8	,, DEPRECIATION ON FURNITURE ..	20 0 0	
177 8 2)	,, Do. TRANSACTIONS ...	163 10 3	
202 1 10			183 10 3
	,, GRANTS: Seymour Memorial Library	50 0 0	
	Van 't Hoff Memorial Fund	10 10 0	
	Evening Class Prizes ...	9 0 0	
			69 10 0
	,, SOCIETY'S JOURNAL—		
	Cost of Reporting, Printing		
(1,359 11 1	and Sundries	1,275 7 3	
361 1 1)	*less* Received for Subs. and Advts. after deducting commissions on		
998 10 0	Advts. and Collections ...	456 16 8	
			818 10 7
£2,084 10 10			£1,924 12 6

Cr.

1910-11. £ s. d.	REVENUE.	1911-12. £ s. d.	£ s. d.
	By SUBSCRIPTIONS—		
(1,252 13 0	Members	1,057 7 0	
486 18 9	Associates	447 14 6	
29 2 9)	Students	8 18 6	
			1,514 0 0
1,768 18 9			
107 2 0	,, ENTRANCE FEES		57 15 0
13 8 4	,, INTEREST ON FIXED DEPOSITS ...		23 2 3
195 6 0	,, BALANCE, deficit on year's working		329 15 3
£2,084 10 10			£1,924 12 6

BALANCE SHEET, as at 1st June, 1912.

Dr.

1910-11 £ s. d.		1911-12. £ s. d.	£ s. d.
(215 11 5	To SUNDRY CREDITORS	337 6 3	
26 15 6	,, SUBSCRIPTIONS paid in advance ...	35 3 6	
6 0 0	,, *Prize Fund*		
20 0 0)	,, SUBSCRIPTIONS FOR DECENNIAL INDEX	5 0 0	
			377 9 9
268 6 11			
	,, REVENUE ACCOUNT—		
(1,536 2 2	Balance at June 1, 1909 ...	1,340 16 2	
	Less deficit as per Revenue		
−195 6 0)	Account, June 1, 1912 ...	329 15 3	
			1,011 0 11
1,340 16 2			
£1,609 3 1			£1,388 10 8

Cr.

1910-11. £ s. d.		1911-12. £ s. d.	£ s. d.
	By CASH AT BANK—		
(150 8 5	Johannesburg	55 18 0	
30 1 4)	London	12 2 4	
180 9 9	,, Cash in Hand	76 14 3	
			144 14 7
713 8 4	,, Amount on Fixed Deposit and in Post Office Savings Bank, with accrued Interest		722 12 7
120 0 0	,, OFFICE FURNITURE		100 0 0
530 0 0	,, TRANSACTIONS, VOLS. I. TO XII.— Stock in hand		350 0 0
(67 7 9	,, SUNDRY DEBTORS...	73 6 3	
2 2 9)	*less* Provision for Doubtful Debts ...	2 2 9	
65 5 0			71 3 6
£1,609 3 1			£1,388 10 8

We have examined the above Balance Sheet and accompanying Revenue and Expenditure Account with the books and vouchers of the Society, and have obtained all the information and explanations we have required. In our opinion such Balance Sheet is properly drawn up so as to exhibit a true and correct view of the affairs of the Society according to the best of our information and the explanations given to us, and as shown by the Books of the Society.

Sgd. ALEX. AIKEN and CARTER, } *Hon. Auditors*

JOHANNESBURG, 21st JUNE, 1912.

(Signed) JAMES LITTLEJOHN, *Hon. Treasurer.*

(Signed) FRED. ROWLAND, *Secretary,*
Fellow Chartered Inst. Secretaries.

January, 1912.—"Determination of Carbon Dioxide" (Note), by Dr. J. Moir.

"Some Observations of Ancient Mine Workings in the Transvaal," by Mr. T. G. Trevor.

February, 1912.—"Poisoning of Cows by Cyanide" (Note), by Mr. A. Whitby.

"Presidential Address," by Mr. C. B. Saner (President).

"Zinc Dust Tests," by Dr. W. J. Sharwood.

March, 1912.—Accidents in Stamp Mills and Cyanide Works" (Note), by Mr. G. A. Robertson.

"The 'Luminator' Purification Treatment of Water for Steam Boilers," by Mr. W. Cullen.

April, 1912.—"Notes on Coal from Wankie Colliery, S. Rhodesia" (Note), by Mr. James Gray.

"Determination of Carbon Dioxide" (Note), by Mr. E. H. Croghan.

"Regeneration of Cyanide Solutions" (Note), by Mr. W. D. Williamson.

"The Action of Mineral Sulphates and Arsenates on Cyanide Solutions," by Mr. Andrew F. Crosse.

May, 1912.—"Note on Dust Determination by Filtration through Sugar," by Mr. A. McArthur Johnston.

"New Apparatus for Sampling Air for Dust," by Mr. E. J. Laschinger.

"A System of Keeping Underground Costs and Records," by Mr. G. Hildick Smith.

June, 1912.—"Estimation of Carbon" (Note), by Mr. A. McArthur Johnston.

"The Surface Workers on the Rand, and their Technical Education," by Mr. F. J. Pooler.

"Mine Tributing in Rhodesia,' by Mr. W. Anderson.

It will be noted that there is a decrease in the number of papers submitted compared with previous years, but with one exception their reading and the discussions thereon, have fully occupied the whole of the usual time alloted to the monthly meetings. It would be of great assistance to the Council and of value to the members if those who intend reading papers would notify the Secretary in order that convenient arrangements may be made for their submission to the Society. It would be of considerable advantage if a syllabus could be printed in advance for the year or even for six months, and the co-operation of members in this direction is cordially invited.

The Council would bring to the notice of members submitting papers or contributions to discussions the fact that the use of personalities —whether against individuals, classes or sections —is strongly deprecated, and it would recommend that this should be always borne in mind when submitting contributions.

An innovation was inaugurated this year, on the suggestion of Mr. K. L. Graham, of including in the *Journal* each month the contents, not only of the current number but of previous issues. It is hoped it will prove of value to members.

It was only possible to make one excursion this year, namely the interesting visit to the Coalbrook Colliery in the Free State. The courtesy and kindness of the General Manager, Mr. Matthew Cullen, and the hospitality of the Directors, was gratefully acknowledged by those who took advantage of the invitation. Several interesting visits are contemplated during the coming year.

The Annual Dinner was again a successful function although the attendance was not so large as in former years. The Council has had under consideration the changing of the usual time—approximately the Society's birthday—of holding the Dinner, to a month less taken up with similar functions, and the matter will be again dealt with by the incoming Council.

The Society's prizes to Students at the School of Mines Evening Classes were again competed for last November, and will be distributed at the meeting this evening. The results were an improvement on the previous year and the successful Students are to be congratulated.

The Chemical Sub-Committee of the S.A. Engineering Standards Committee, most of the members of which are prominent members of this Society, published a few months ago the result of their labours on the "Standardisation of Sampling Coal" in a 12 pp. pamphlet, and have just issued another on the "Standardisation of Coal Analysis, Determination of Calorific Values, and Standardisation of Sizes of Coal." Other work is being taken in hand by the Committee which it is hoped will prove of value to the industry, to the community, and South Africa as a whole.

The Eighth International Congress of Applied Chemistry will be held on September 4th-13th in Washington and New York and Dr. James Moir, Past President, has undertaken to officially represent this Society, in addition to his representing the Union of S.A. in an official capacity. The Society will, no doubt, receive an interesting account of the proceedings upon Dr. Moir's return.

In response to a letter from the Royal Society of South Africa, the Council appointed a representative upon a Committee formed by that Society and the S. A. Association for the Advancement of Science to deal with a Research Fund established for the furtherance of Scientific Investigation in Southern Africa. It is understood that certain grants were made last year and that applications have been invited for grants this year, but as this Society has received no intimation, excepting the first above mentioned, the Council is unable to give members any further information on the subject.

It will be noted from Dr. McCrae's letter published last July, that the opportunity exists locally for those desirous of obtaining the Associateship or Fellowship of the Institute of Chemistry to do so. As a Society keenly interested in the advancement of Chemistry, the Council warmly welcomes the opportunity thus afforded to South African Students and it hopes it will be taken advantage of to the fullest extent.

The Council also welcomes a new association, The S.A. Association of Analytical Chemists, formed for the advancement and protection of professional interests which, due to the energy of some of our members, has recently come into being. The Council extends to the new body its best wishes for future success.

With a view to bringing to the notice of those interested in the industry and to the public generally, the many ingenious devices of the inventor, natural products of the country, and the various types of life-saving apparatus, the Council, acting upon the suggestion made independently by two of the Past Presidents (Messrs. Andrew F. Crosse and William Cullen) resolved to inaugurate a Mining Exhibition. The Council of the School of Mines very kindly placed at the disposal of the Society, the Assembly Hall of the School of Mines as a *venue* and the Transvaal Chamber of Mines generously made a grant to the Society towards the necessary expenditure. The exhibition was brought to the notice of members in November and was held during the three weeks ending on the 2nd March, being opened on the 9th February by Mr. H. O'K. Webber then President of the Chamber of Mines. Commercial firms came liberally to the support of the venture, the Press (particularly the *Star*) gave much publicity to the Exhibition, and there was a very fair response from inventors considering the limited notice given. It is estimated that between 16,000 and 17,000 people visited the Exhibition, and the opinion generally expressed was that it was a great success. During the time it was open many demonstrations were given showing the manner in which life-saving apparatus should be used, while practical demonstrations of the several rock drills with and without dust arrestors were given on three Saturdays. Financially, the Exhibition met all its expenses, and to commemorate the event the Council has voted a sum of thirty guineas to purchase a suitable trophy for competition amongst Mine Ambulance teams to take place under practical mining conditions underground. A Committee of the Council is at present working in conjunction with a Committee of the St. John Ambulance Association framing the conditions for the competition which it is hoped will take place in October of the present year, but ample notice will be given of the date on which it is to be held. In this connection it is also expected that in conjunction with the competition above mentioned, a further competition will be held for mine surface employees. The efforts of the Council to further the interests of underground workers in this respect will, it is known, meet with the full approval of members.

In connection with the success of the Exhibition, interest has revived in the proposal to form a permanent museum for Johannesburg. At present there are a few exhibits in the Geological Museum, and with a view to this Society doing its share towards the formation of a suitable museum for the City, the Museum Committee requested the Council to appoint two of its members to join it. Members of the Society will appreciate the value of such an institution and will no doubt support the movement.

The assistance of the corresponding members of Council has been greatly appreciated by the Council during the year under review.

Twelve Council meetings have been held at which the average attendance has been 12. The removal of the President to Pilgrims Rest after his return from England was a great blow to the Council who had looked forward to having the advantage of his presence with them during his year of office. His absence has been greatly felt, particularly by the mining section of our work, but his duties have been ably carried out by the Vice-Presidents to whom your thanks are due. Prof. Wilkinson, who unfortunately contracted enteric fever towards the end of the year, left for England as soon as he was convalescent and has recently married. He has our best wishes for his future happiness. Mr. K. L. Graham and Mr. F. W. Watson have also gone on long leave. Mr. F. L. Bosqui was appointed to fill the vacancy on the Council caused by the death of Mr. G. O. Smart. It is regretted that owing to ill health Mr. Tom Johnson was prevented from attending a number of Council meetings and we wish him a speedy recovery.

The usual table of attendance at Council Meetings is given hereunder :—

Mr. C. B. Saner[1] -	1	Mr. G. O. Smartt[7] -	1
,, W. R. Dowling	9	Prof. G. H. Stanley	9
,, E. J. Laschinger	8	Mr. J. E. Thomas -	10
,, H. A. White -	12	,, F. W. Watson -	10
,, J. Littlejohn -	6	,, A. Whitby - -	9
,, R. Allen[2] - -	6	Prof. J. A. Wilkinson[8]	5
,, F. L. Bosqui[6] -	4	Mr. R. G. Bevington	5
,, M. H. Coombe -	6	Dr. W. A. Caldecott	2
,, K. L. Graham[3] -	3	Mr. A. F. Crosse -	4
,, Jas. Gray[4] - -	8	,, W. Cullen - -	5
,, Tom Johnson[5] -	2	,, A. McA. Johnston	5
,, A. Richardson -	11	Dr. J. Moir - - -	4

[1] Granted leave of absence for 11 months.
[2] ,, ,, ,, 7 months.
[3] ,, ,, ,, 9 months.
[4] ,, ,, ,, 3 months.
[5] Absent from 4 meetings through illness.
[6] Elected alternate to Mr. Smart, July 1911, and to Mr. Graham, Oct. 1911 ; granted leave of absence for 2 meetings.
[7] Granted leave of absence for 3 months' ; died Nov. 1911.
[8] Granted leave of absence for 6 months.

The thanks of the Society are also due to the members of the various Committees, Editorial, Publications, Finance, Exhibition, Ambulance, etc., who have done much good work for the furtherance of the Society's objects.

With a view to furthering the interests of Student Members of the Society and to encourage them to submit papers and contributions to discussions, the Council in October last offered a prize of ten guineas for the best paper submitted and a prize of three guineas for the best contribution to discussions during the remainder of the financial year. It is much regretted that no student took advantage of the last named offer and that only one paper was received in response to the prize for a paper. Mr. Tom Johnson at the April Meeting gave a number of useful suggestions for the consideration of mining students, which by the way, could be borne in mind by the mining members also. The subject will be considered again by the incoming Council with the hope of better results next year.

The President : There is little for me to add in moving the adoption of the Report and Balance Sheet of the Council for the year.

I could bring to your notice especially how we are utilising the funds of the Society. The ten guineas to the Van't Hoff Memorial Fund, and £50 to the Seymour Memorial Library are big amounts out of our revenue, but are for excellent objects. A great many of our members take advantage of the Seymour Memorial Library and I am quite certain nobody will grudge those amounts being donated.

It will be noticed that the Report states that the number of papers has not been so large as in previous years, although their excellence has kept up to that high standard. In that connection I should like to appeal to members to prepare their papers beforehand so as to give the Council some chance of arranging an agenda a few months ahead. By this means the Secretary, or the President for the time being, would not be harassed in raking up his friends to get papers. Perhaps a conscientous man who when asked, at once sits down and burns the midnight oil, does not prepare as good a paper as he should do. It takes a long while to write a paper properly, if it is thought out well ahead (whether it is on a chemical, metallurgical or mining subject) and sent to the Council, they will be able to devote the necessary time to prepare it properly for publication. Those who have been on the Council realise it is no child's play, especially to the Editorial Committee. That Committee gets a tremendous lot of kicks, yet a good deal of hard work falls to their share, as I have known to my cost when a paper has been sent to me, and I have torn my hair in an endeavour to keep the sense and leave out the nonsense.

I am sorry to see that there was only one excursion to a mine. I had made one or two tentative arrangements for special excursions and I had made up my mind that my term of office would be noted for the number of excursions to the various mines. I only hope more will be arranged in future and I can assure you I will do my best to help the incoming President to carry out those excursions. I am a tremendous believer in seeing what has been done elsewhere. One can often learn how not to do things when one sees them actually in use.

I was glad to see that the Annual Dinner of the Society was such a marked success. I think that the Council is taking a step in the right direction by altering the date. By that means we will get the leading politicians to honour us, fresh and unharassed, and not jaded after interminable spars on the language and other questions they deal with " in another place."

The move that the Society has made in giving prizes to Students of the Society, as well as to the Evening Class Students, I think is an excellent one, and I hope next year when the scheme is more widely known, there will be any number of competitors. I would only give a little word of advice to those students. When I was serving my time I had not much opportunity of doing work by day, so I used to go to the night schools, and I think having an incentive in the shape of a prize at the end made me stick to the work.

There is another thing. I came here practically as a spectator to see the Exhibition that was held and I say that the Society deserves a great deal of credit. Especially is the credit due to our Past Presidents, Mr. Crosse and Mr. Cullen for initiating the idea. The Society is always in the forefront endeavouring to bring forward new ideas to help the mining industry and I hope it will continue to do so. I see that the Council intends to further the interests of the underground workers in this respect, by donating a shield for an Ambulance Competition for that section of the workers. Everyone is talking about the insecurity of tenure of mine billets and I think that this Society, through its discussions, has been the means of voicing that much troubled question to most of us.

In concluding my remarks I should like, specially on my own behalf, to offer Mr. Dowling and the other Vice-presidents my sincere thanks for taking the chair during my unfortunate absence. Many a time I thought I would try and come down, but it really meant being four days away, and as one is bumped about in a Cape cart or buckboard and covered with dust, it is no small or pleasant undertaking.

I specially wish to mention the good services of the Secretary throughout the year. He works many more than the 8 or 12 hours a day. A special vote of thanks should be accorded to him for his continual excellent and arduous work.

I have great pleasure in moving the adoption of the Annual Report and Accounts.

Mr. Andrew F. Crosse (*Past-President*): I have great pleasure in seconding the motion, and it has interested me very much to have heard the Report read this evening. Those of us who are Past-Presidents can look back with a certain amount of pride, and can congratulate the Society on its present position. Especially those gentlemen who are responsible for the *Journal* and its abstracts deserve particular praise and thanks.

The Reports and Accounts were then adopted and passed.

The Secretary read the Scrutineers' report and declaration of election, as follows:—

Johannesburg, 22nd June, 1912.

THE PRESIDENT,

CHEMICAL, METALLURGICAL AND MINING SOCIETY, JOHANNESBURG.

Dear Sir,—We have to report that we have this day examined 142 ballot papers handed to us by the Secretary.

There being no opposition, we have to declare as elected:—

PRESIDENT:

Mr. W. R. Dowling.

HON. TREASURER:

Mr. James Littlejohn.

The result of the ballot for the other offices is as follows:—

VICE-PRESIDENTS:

Mr. H. A. White.
Mr. K. L. Graham.
Mr. A. Richardson.

MEMBERS OF COUNCIL:

1 Prof. G. H. Stanley
2 Mr. E. J. Laschinger
3 ,, Tom Johnson
4 ,, A. Whitby
5 ,, J. E. Thomas
6 Prof. J. A. Wilkinson
7 Mr. Ralph Stokes
8 ,, Robt. Allen
9 ,, Jas. Gray
10 ,, F. W. Watson
11 ,, A. Salkinson
12 ,, M. H. Coombe

We are, Dear Sir,

Yours faithfully,

Sgd. S. NEWTON,
,, A. THOMAS,
,, R. LINDSAY,
} *Scrutineers.*

The President: The Society is to be heartily congratulated on the results of this election. Mr. Dowling has got to the top of the tree after many years of hard work. I am very pleased it is your unanimous vote. Also, with regard to the Vice-Presidents and Council, it shows the solid satisfaction that the Society has in the good work that the Council has been doing in the past year, by the re-election of every member of the old Council. That shows there cannot be much dissatisfaction with the running of the Society.

Mr. John R. Williams (*Past-President*): It gives me also great pleasure to congratulate the Society upon the election of our friend Mr. Dowling as President of the Society. Mr. Dowling is more of a pioneer than any one of us. He was an old stager when I came to Johannesburg in 1889 and this makes his 25th year of residence in the Transvaal. I think it is very fitting that on his silver wedding to the Transvaal he should be made President of the Society. He was our very first Secretary and I assure you he has worked very hard. I ask you during his year of office to loyally help him in every possible way. In taking on the Presidentship of the Society there is a large amount of work attached, but I know that Mr. Dowling has taken on that work in the same spirit that all of us have done, following the dictates of our own hearts and making it a labour of love. I know that Mr. Dowling is particularly fond of the Society and the Society may be congratulated on his election as President.

Mr. W. R. Dowling (*President-Elect*): This laudation is quite embarrassing. I wish to thank you for electing me to this office. It has been my aim to become one of your Presidents for a great number of years. I have been associated with the Society for a long time and I owe a great deal of the success that I have reached in life to my association with the Past-Presidents of the Society—my old chief Mr. Crosse, Mr. J. R. Williams and to Dr. Caldecott.

I wish to thank you for the honour you have conferred on me and the trust you have imposed in me. In the Council which you have elected I am sure I shall have all the support necessary to enable me to maintain the position of the Society and further its objects.

Mr. H. A. White (*Vice-President*): On behalf of the Vice Presidents and newly elected Councillors I beg to thank you for the honour you have done us and to assure you that we will do our best to deserve it. I also propose a hearty vote of thanks to the scrutineers for the work that they have done this afternoon.

Mr. C. Toombs (*Member*): As one of the unsuccessful candidates I beg to second the hearty vote of thanks to the scrutineers for their arduous labours.

Mr. S. Newton: Many thanks for the kind words expressed as to our arduous duties. We hope to do it again. We will do it willingly, and also assist the Society in any way that we can.

NEW MEMBERS.

Messrs. W. S. Price and C. Toombs were elected scrutineers, and after their scrutiny of the ballot papers, the President said I am glad to report that all the candidates for membership have been duly elected. A very good beginning for the new year.

Arnot, Edwin Percy, Simmer and Jack Proprietary Mines Ltd., P. O. Box 192, Germiston. Mine Surveyor.

Balfe, Maurice Francis, Simmer Deep Ltd., P. O. Box 178, Germiston. Mill Shiftsman.

Bruce, Christopher Yule, Simmer Deep Ltd., P. O. Box 178, Germiston. Mine Captain.

Crumpton, Harry, "C" Mine, Belingwe, S. Rhodesia. Assayer and Surveyor.

Gregg, Edwin Tredgar, Simmer Deep Ltd., P.O. Box 178, Germiston. Amalgamator.

Holman, Bernard Whelpton, Heymann's Laboratory, P. O. Box 3427, Johannesburg. Assayer. (*Transfer from Associate Roll.*)

Juby, George Henry, Simmer Deep Ltd., P. O. Box 178, Germiston. Mill Shiftsman.

Malpage, Theodore Albert, P. O. Box 178, Germiston. Assistant Manager.

Matheson, Dugald, Simmer Deep Ltd., P. O. Box 178, Germiston. Mill Shiftsman.

McAlpine, Hugh, P. O. Box 178, Germiston. Mill Shiftsman.

McNeill, William, Bacis Gold and Silver Mining Co., Ltd., Bacis, Durango, Mexico. (*Temporary address* 20, Tower Road West, St. Leonards-on-Sea, England.) Mining Engineer.

Welsh, Thomas Seth, P. O. Box 178, Germiston. Tube Mill Shiftsman.

The Secretary announced that since the last meeting 20 Associates and 2 Students had been elected:—

As Associates—

Adams, Alfred Charles, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

Bertram, Edwin Stuart, Rose Deep, Ltd., P. O. Box 6, Germiston. Cyanider.

Botha, Paul Michiel, Simmer Deep, Ltd., 17, High Road, Germiston. Mill Assistant.

Dakin, Norman Whitaker, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Assistant.

Flowers, Alfred, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

Glass, George Alfred, Simmer Deep, Ltd., P. O. Box 178, Germiston. Assistant Amalgamator.

Harger, Wilfred George, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mill Foreman.

Horsfield, James Ernest, Giant Mines, Gadzema, Rhodesia. Mechanical Engineer.

Hutton, James, Consolidated Langlaagte Mines, Ltd., P. O. Box 13, Langlaagte. Metallurgist.

Johnson, B. Angas, Simmer Deep, 178, Germiston. Reduction Works Learner.

Joubert Joshua, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

Laidler, Herbert James, Ivanhoe Gold Corporation, Ltd., Boulder, W. Australia. Assayer.

Lange, Pieter Jacob de, Simmer Deep, Ltd., 50, Lambert Street, Germiston. Cyanider.

Moulton, Hamilton, Simmer Deep Ltd., P. O. Box 178, Germiston. Mill Assistant.

Osborne, Harvey Glasson, Simmer Deep Ltd., P. O. Box 178, Germiston. Cyanider.

Roos, Isaac Franciscus, Simmer Deep, Ltd., 10, Highgate Street, Jeppestown. Assistant Amalgamator.

Stuart, William, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

Thorpe, Aubrey, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

Waldeck, George Victor, Simmer Deep, Ltd., 3rd Road, Malvern, Johannesburg. Cyanider.

Winters, Roelof, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Shiftsman.

As Students—

Maunsell, Errol Creagh, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mill Learner.

Osborne, Ernest, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanider.

General Business.

The President: I have the honour, on behalf of the Institution of Mining and Metallurgy, of making the presentation of the Consolidated Gold Fields medal, which has been awarded this year to Mr. W. R. Dowling. This particular paper ("The Amalgamation of Gold in Banket Ore"), as well as one previously read by Dr. Caldecott, was read by special arrangement both before this Society and before the Institution in London. I think it reflects great credit on the care that members take in the preparation of their papers that practically throughout the whole world our members should be able to capture this medal so often. It might be of interest to you to know that there is an arrangement made by the Institution of Mining and Metallurgy with the Chairman of the Consolidated Gold Fields of South Africa, Lord Harris, by which a gold medal with a premium is given annually for the best paper on gold mining and the treatment of gold ores. The Award is adjudged by a Committee of three gentlemen nominated by the Council of the Institution of Mining and Metallurgy. The first award was made in 1902, and altogether awards have been made to 17 gentlemen, either of the medal and premium or the medal or premium separately, and out of those 17, eight have been members or hon. members of this Society, who, of course, are also members of the Institution of Mining and Metallurgy.

I only hope that it will be an incentive to a great many of our members, who are not members of the sister Institution, to join that body and also do their best to try and keep up the standard of excellence set by others in winning this valuable prize. I have great pleasure in presenting Mr. Dowling with this very beautiful medal. I think it shows that the minds of the members of the Institution of Mining and Metallurgy are the same as ours. We have elected him to the top post. They have given him their best award.

The President then presented the prizes given by the Society to the successful students at the Examinations of the Evening Class of the School of Mines, the names of whom were published in the *Journal* for December last; and also the prize valued at ten guineas for the best paper from students of the Society which had been awarded to Mr. J. P. Beardwood for a paper on the "Sampling and Preparation of Coal and the Determination of its Calorific Power."

ESTIMATION OF CARBON.

Mr. A. MacArthur Johnston (*Past-President*): I wish to bring to your notice an adaptation which we have made for the determination of carbon, when, owing to the absence of gas, the ordinary combustion furnace has not been available.

This should especially appeal to chemists on mines or in outside districts, where gas is an unknown commodity, whilst on the other hand the muffle is a part of the outfit. The combustion tube is composed of a length of opaque silica tubing 42 in. long and $\frac{3}{4}$ in. inside measurement. This is bent to the shape of a U tube with limbs about 18 in. long and 6 in. apart, outside measurement. This size fits conveniently into a 'K' muffle.

The platinum boat, or piece of silica tubing open at both ends, containing the carbon to be burnt, is inserted into one limb, *e.g.*, the left hand one, care of course being taken that it be not jammed at the bend, whilst the right hand limb is connected to the potash bulbs. In this way the greatest heat is found at the bend after the carbon is burnt and complete combustion ensured. In connecting the ends of the tube to the absorption apparatus, sufficient length of capillary tubing must be used to allow for moving the combustion tube backward or forward, whilst capillary glass tubing may be employed that all the bulbs may be assembled on the same bench or table.

For combustion purposes the rounded end of the tube is gradually inserted into the muffle which has been brought to the desired temperature, the degree of heating of the tube being controlled roughly by the distance to which the tube is inserted. A thick piece of asbestos board with two openings cut therein to allow the tube to pass through serves as the muffle door. It is advisable to extend the ordinary muffle ledge in some manner so as to afford a level rest for the tube when withdrawn, whilst the two limbs may with advantage be clamped to each other.

THE SURFACE WORKERS ON THE RAND AND THEIR TECHNICAL EDUCATION.

By F. J. Pooler, B.Sc. (*Associate*).

Introduction.—For many months this Society has been discussing conditions of working on the mines of the Rand, special attention having been paid to the underground workers. Even in the discussion of accidents much of the information has concerned the miner and rightly so, since his lot stands more in need of improvement than that of the surface man. For this reason it occurred to the writer that a few notes on the surface worker and his prospects might be of value. One hears so much of the white labour cry that a reference to it seems to raise a political question at once, and yet any paper dealing with surface work conditions must of necessity touch on it. This is obvious when it is remembered that since January 1909, the Consolidated Goldfields Company has made an organised effort to employ young white men on its surface work. The writer is informed that this experiment has so far justified itself as to become a settled policy. Later on the Rand Mines, Ltd. also instituted a scheme on somewhat similar lines, a special White Labour Department being created to deal with the matter.

Scope of the Paper.—It is proposed in this paper to confine the discussion to the learners and apprentices who come under schemes such as those mentioned above or who are working on other mines where no special scheme is in existence. The fact that certain of the larger mining houses thought it advisable to introduce a scheme of this kind leads one to infer that some necessity to do so must have existed, otherwise the time involved in organisation would have been wasted, more or less. Doubtless the heads of the departments concerned felt it incumbent upon them to train men to take the place of the older hands who must naturally drop out with the efflux of time. The large number—the increasing number—of white youths leaving the High Schools and Colleges of this country every year offered a natural source of supply, a source promising, at least, a fair number of candidates who possessed a ground-work of education that would probably help them to attain to a fair degree of efficiency in a reasonable time. It is more than likely that the fact that the learners appointed having had a good basis of educational attainment, would be more willing to undertake special study with a view to fitting themselves for their particular work and more able to profit by that study, influenced the decision to give the system a fair trial. How far such a hope appears to have been realised, it is proposed to enquire later.

As illustrating the class of young men employed under one of the schemes mentioned above, the following figures are interesting :—

TABLE A.

No. of Learners at work	275
Abone Matriculation Standard ...	2·13%
Matriculation	22·34%
School Certificate (formerly School Higher)	32·55%
Under School Certificate	42·98%

"The learners are appointed wherever possible from among the matriculated students who leave the Schools and Colleges of South Africa annually."

Now in 1910, over 400 boys passed the Matriculation Examination. As the total number of successful candidates at the intermediate B.A., B.Sc., Mining and Law Examinations per year is in the neighbourhood of 300 and a considerable proportion of those who passed the Inter. B.A. are girls, it is obvious that there is a fairly large number of youths, probably about 200, who seek other employment than the professions. The year 1910 is chosen, as the number of matriculants was below the average in that year. From this, it is probable that we may reasonably look for an increasing number of applicants for these posts when once the conditions of service are better understood.

Conditions of Service.—The learners are informed before applying for the posts that the work they are called upon to do is hard and therefore good physique and grit are a necessity. From personal experience the writer feels he can say that this has not been fully realised by some of those appointed to the posts during the last 3 years. The point however is, that though work may commence in the batteries, wherever possible other surface work of a less arduous nature is found, for those who have done well, after a reasonable period.

A writer in the *Kingswood College Magazine* in June 1911, dealt fully with this point and *inter alia*, stated "scores of young colonials have started under these conditions . . . . and are making good progress but others have left, in some cases, through lack of physical power, in more through want of pluck and perseverance. The whole trouble lies in the fact that the reports of the life and prospects of the learners on the mines have been painted in colours too glowing, while the hardships, that are unavoidable, have been passed over too lightly." If, however, well-educated young men having the grit to face the rough life or what appears to them to be a rough life, can be drafted into the surface work, it is evident that no long time must elapse before the less well educated "rule-of thumb" man will have to make room or improve his qualifications. The latter is the obvious way out of the difficulty and the heads of departments on the mines can confidently look forward to increased efficiency, while the men can anticipate what is so necessary, increased security of tenure and stability of service.

Many of the "learners" object to the conditions of living on the mines, but it is fairly apparent to the casual observer that much has been done and we may reasonably expect will continue to be done to improve the housing accommodation provided. The aim of the groups concerned is to have separate quarters built on the mines for the accommodation of the Reduction Works staff.

Stability of Service.—With a view to illustrating how far the "learners" have accommodated themselves to the conditions, Dr. Caldecott kindly supplies the following figures :—

TABLE B.

From Jan. 1st, 1909 to Dec. 31st, 1911.

Total "learners" employed by the Consolidated Goldfields, Ltd.	220	
Still employed	105 =	47·73%
Employed in other Departments or Groups	28 =	12·73%
Left on own accord	73 =	33·18%
Dismissed as unsuitable ...	14 =	6·36%

Thus less than 40% have left or been dismissed in 3 years. This figure may seem rather high, but as some of these have certainly remained on the Rand in other employment, the number is perhaps a little too high.

Some dissatisfaction has been expressed as to the rate of wages paid to learners on different mines. From figures shown by one of the Groups to the writer this seems to be unfounded, the rates varying very little when length of service is taken into account, from mine to mine. The average rates of pay of over 100 learners is given as follows :—

Table C.

				s.	d.	
Employed	less than	6	months ...	8	8	per shift
,,	over	6	,, ...	10	1	,,
,,	,,	12	,, ...	10	1	,,
,,	,,	18	,, ..	12	3	,,
,,	,,	24	,, ...	15	2	,,
,,	,,	30	,, ...	14	8	,,

These figures show that this scheme offers a reasonable prospect of advancement to any youth who cares to do a fair day's work and sink his personal dislike to dirty hands, in his determination to succeed. The writer in the *Kingswood College Magazine* referred to above, says, however, in this connection, "There is no fixed standard of wages for learners on the various mines in the Group, men on one mine frequently being paid a higher wage than those doing corresponding work on another. Naturally this causes a great deal of dissatisfaction, etc." How far this is applicable the author cannot say, but his own opinion from figures shown, is stated above. A curious paragraph that seems worth a reference, occurs in this letter :—"Another great drawback is the fact that all learners are on a day's pay and may be dismissed on 24 hours' notice. This state of affairs may exist among the working classes, but it is unreasonable for any company to expect a man of the class such as the conditions of the scheme stipulate, to spend years of his life in this state of insecurity." Probably everyone would agree with this, if the first clause of the last sentence had been deleted.

Facilities for Technical Training.—Having very briefly summarised a few points with regard to the conditions of service, it is necessary to turn our attention to the facilities provided for the training of the learners, apprentices and students on the mines in the special subjects of which their work demands a knowledge. At the outset the author wishes to say that he does not intend to discuss the relative merits of the "technically trained" and the "practical" man, since in his opinion both theory and practice are absolutely necessary for the successful carrying out of any undertaking, But a member of the Council of this Society* stated recently "that there are too many technically trained men and not enough practically experienced men in charge of underground operations." As to this, the author is not able to express an opinion, but it is his conviction that on the Reduction Works their cannot be too many technically trained men, so long as these men are also conversant with the details of the practice of the processes they supervise. The writer of the above-mentioned remarks evidently is of a somewhat similar opinion as to the underground worker, as he advocates "three to five years practical work with the men whom one day he aspires to control." But the remarks when read hastily convey a somewhat scathing indictment of the technically-trained man who without practical experience is referred to as "useless as a drone and a source of irritation to men who are doing the work." Surely isolated cases that occur—as they do in every walk of life, are not so common as to require such a charge being laid at the door of the average man who has thought it worth while to add to his knowledge of the principles underlying the processes he controls.

Some years ago, Professor Wilkinson read before this Society a paper on the training it was proposed to give at the local Technical Institute (now School of Mines), but dealt specially with "Day" students.* He was criticised because he had neglected *the majority* for whom Technical Classes are required, viz., "Evening" students, so the author feels no apology is needed for examining, somewhat at length, the provisions made for and the advantage taken of those provisions by the learners, apprentices, etc. on surface work to-day.

Need for Scientific Training.—The *S. A. Mining Journal* of April 13th, 1912, drew attention to the regrettable fact that many of the more recent improvements in metallurgical practice had had their origin in America. Some remarkable exceptions were noted, especially those associated with MacArthur-Forrest and Dr. Caldecott. How far the indictment is true the author is not prepared to say, but in seeking for a possible explanation, it may be that our local younger generation is not taking as full advantage as might reasonably be expected, of facilities provided by local Institutions, for improving its knowledge of underlying principles of local mining and metallurgical practice. The Secretary of the White Labour Department of the Rand Mines informs the writer that of about 275 learners, "the number regularly attending lectures at the Transvaal School of Mines and

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*This *Journal*, Vol. XII., No. 8, p. 325.

*This *Journal*, Vol. VII., No. 1, p. 26 and seq.

Technology was very small during 1911, being only 30. The number of learners attending during the present term is, we think, a good deal larger." Also, "with regard to the apprentices working in the engineering shops, etc. of the mines of this Group, it is an understood thing that they attend evening classes, and periodical reports of their attendance are sent to the Engineer concerned." The author would like to suggest that similar reports to the Reduction officers, of other surface learners, might with advantage be made a feature of the scheme under consideration. "It is more than likely that, in future, regular attendance at one or other of the technical classes . . . will be made a *sine qua non* before a learner obtains promotion," is an extract from the conditions of service on the Rand Mines, Ltd. The writer is of opinion that the sooner the better for the industry and the learners."

How far the facilities provided are used.—The Principal of the S. A. School of Mines and Technology has kindly allowed the author to use the numbers given in Table D. These are taken from the books of the school, though any opinions expressed thereon are distinctly to be regarded as the personal opinions of the author who has no permanent position on the staff of this Institution, but has had some experience both as a lecturer and a student of the work it is doing.

In table D the figures No. 1 refer to total students, No. 2 to learners and apprentices who enjoy the privilege of half-fees. A satisfactory feature is the steady increase in the proportion of apprentices, etc., to total students, in the above mentioned classes, which are those most directly dealing with the surface work and are attended by cyaniders, assayers, battery men and samplers, principally.

Now, however, consider Table E., compiled from official figures issued by the Government Inspector of White Labour.

In table E the figures No. 1 refer to total employees on the Reef, and No. 2 to apprentices and learners.

TABLE D.

	No. of Students on Books.									
	June, 1910.		Nov., 1910.		June, 1911.		Nov., 1911.		March, 1912.	
	1	2	1	2	1	2	1	2	1	2
Chemistry I. ...	83	12	69	Not Stated.	80	46	50	31	54	35
Chemistry II. ...	27	4	20		24	7	16	8	20	6
Chemistry III. ...	No	Class	...		2	0	2	0	...	...
Metallurgy I. ...	9	2	4		29	14	20	8	14	6
Metallurgy of Gold	8	0	8		No	Class	...	...	19	5
Metallography ...	6	1	1		No	Class	...	...	...	...
Assaying I. ...	13	3	11		23	9	14	4	17	6
Assaying II. ...	1	0	1		5	1	5	1	7	4
TOTALS ...	147	22	114		163	77	107	52	131	62
% (2 : 1) ...	15·0		——		47·2		48·6		47·0	

TABLE E.

No. Employed.	June, 1910.		Dec., 1910.		June, 1911.		Dec., 1911.	
	1	2	1	2	1	2	1	2
Mill (excluding Fitters) ...	1318	228	1325	299	1480	317	1493	261
Cyanide (do)	1022	130	1006	157	1072	217	1137	223
Samplers (Technical Staff) ...	Not stated.		Not stated.		296		309	
Assayers and Assistants ...					174		174	

Comparison.—It is obvious that there are many men who are not taking advantage of the facilities provided. Naturally the heads of departments on the one hand and unskilled labourers on the other are not, as a rule, to be expected to do so. But these form comparatively a small proportion of the totals given. But the numbers of such are not stated above, as the author from personal experience, is aware of several cases of mill and cyanide foremen who attended the classes and are included in the members given in Table D. It is only fair to note, however, that besides the above classes, Preliminary Mathematics, Elementary Science and Metallurgy are also being held along the Reef. In this connection the Superintendent of Continuation Classes states that "between 130 and 140 mine apprentices" were attending Preliminary Classes of all kinds in the 1st term 1912. Considering that "295 youths left in Standards V.-VII. the schools of the Witwatersrand Central School Board" alone, during 1911, and no account is made of those leaving the East and West Rand Schools, the figures are not very encouraging. Also of the 130 or so mentioned above a number are not employed in surface work of the nature under discussion, being engineering apprentices. That these classes can be made a success, from a numerical point of view at least, is evident when one remembers that at a Preliminary Metallurgy Class conducted by two enthusiasts for technical training, both prominent members of the Council of this Society, some 80 students were fairly regular in attendance. It is to be regretted, however, that of these only 10 were classified as "learners," the rest as "men students."

These figures seem to indicate that the learners till comparatively recently, at least, have failed to appreciate the advantages provided by the more than splendid facilities available along the Reef.

Reasons for failure.—Is one of the main reasons for this failure to take advantage of educational facilities, not due perhaps to a distrust of scientific method and theory of any sort? A member of this Society has more than once stated to the author that "one theoretical man can keep a hundred of us practical men going." True, perhaps, but is it not also apparent that any one of these hundred might have been occupying a similar position to the one, if he had taken the trouble to attain to a greater theoretical efficiency in the work at which he is admittedly an expert in practice? Many inventions come from "practical" men, but how many more labour-saving devices might there have been, if more of those who have never risen from "among the ruck" had known how to apply, scientifically, the many useful hints acquired in the day's work?

This lack of interest in things scientific can scarcely be put down to the charge of the Heads of the Departments of our Industry. It would be interesting to know how many thousands sterling have gone to decrease dividends, in attempts to make labour-saving devices a practical success. The fact that so little response has been made to the offer of handsome prizes to student members of this Society for contributions to our discussions is not an encouraging sign for the future, though to the one who has come forward our heartiest congratulations are due. There were 57 students registered in the society's lists in June 1911, where are the greater number of them when discussion is invited? Neither can it be said that the Government is not fully alive to the need for a wider knowledge of technical processes, as is evidenced by the fact that a Conference on Technical Education was held under the chairmanship of the Minister of Education (and now of Mines) as recently as November last. One of the objects of this conference was "to stimulate and encourage the interest taken in industrial and technical education in this country." The Minister said what probably most of us will endorse, in dealing with certain aspects of the general question·—"there was a town side—where they had a large and growing number of children in large centres like Johannesburg, Capetown and elsewhere, with a bookish education, afraid to work, thinking it was "kaffirs work." The result was, for these men there was no room in South Africa; and South Africa was bound to suffer, unless they seriously set themselves to solve the problem." It is merely a phase of the eternal white labour problem of "what to do with our boys?"

Speaking from experience both here and in Kimberley, the author is inclined to think that more enthusiasm is shown for technical training by the engineering students. Admittedly at Kimberley and here, engineering students are expected and almost compelled to attend classes at nights. The South African College, Cape Town also seems to show this, since the evening class enrolment in 1910 was 195 and in 1911, rose to 424, many of their students, coming from the Salt River Locomotive Works. If this is so, it is interesting to remember a remark of Disraeli that "the prosperity of a country is to be measured by the state of its chemical trade." A very simple paraphrase will show the application to South African conditions, changing "chemical trade" to "cyanide and battery practice."

Perhaps a reason for some of the indifference is to be found, in a measure, in a quotation from the Syllabus of the local School of Mines. "It is necessary to impress upon students the necessity of following some sequence in their studies, otherwise their progress is bound to be slow and unsatisfactory. Usually a student employed at some particular trade or business wishes to commence studying at once in the class that deals most directly with the occupation in which he is engaged, so that he may begin immediately to learn something that will be directly helpful in his daily occupation. Such a short-sighted policy leads nowhere and generally ends in disappointment all round." It is a common failing of mankind to wish to get somewhere as quickly as possible, but one fails to see much profit in getting to a blank wall of inefficiency and lack of interest in one's daily task, when a little foresight and acceptance of facilities for improvement will so surely aid in mounting the wall.

But in fairness to many learners, it is necessary to add that many are doing their utmost to gain a sure footing on the ladder of success by a rigid self-discipline such as is necessary for a man who has a hard day's work to do and then must keep himself mentally and physically fit for a course of study and lectures in the evenings. That this is so, a reference to the letter already quoted from, will show. "The best plan that learners can adopt is to take up something which demands all their attention in spare time, whether sport or study. . . . . Many learners come up with a firm resolve to study. They start well, but manual labour somewhat militates against hard study and they lend too ready an ear to the arguments of fellow-workers whose words are more than their sense and advise that no theory is necessary." Hard words, perhaps, yet they evidently come from one who has been "through the mill" in a double sense and who has seen and chosen the better part.

Difficulties.—It must be granted that there are difficulties in the way, *e.g.* :

(*a*) Many learners are working "shifts" that make regular attendance impossible. But the School of Mines has duplicated its Chemistry and Metallurgy Classes especially to cope with this (*vide* Prospectus).

(*b*) All learners are mentally not fitted to benefit by the instruction. This may be so in some cases, but can scarcely apply in cases where learners have passed examinations of the standard referred to in Table A. Then why not compel, both in their own and in their employers' interests all who are capable, to attend regularly till a fair knowledge has been gained?

(*c*) A youth who has done a hard day's work is not physically and mentally fit to imbibe technical knowledge in the evening. Then what other knowledge does he imbibe, for very few come to the Rand without learning something fresh in a few months?

Also, perhaps a rigid enforcement of an eight hour's day would help. The exigencies of "cleaning-up" have, of course, to be considered, but complaints are sometimes heard of a youth being kept at a hard task for long stretches, almost beyond his physical limit. How far this is the case the author is not prepared to say.

(*d*) Many learners are not fitted to benefit because of deficiencies in their preliminary education. As stated before, this can only operate in some cases and for these, the Preliminary Classes are provided. In any case, this objection will yearly have less force, not only because more matriculants may be expected to come forward, but also that the Day School Code now contains a 3 year's course in Elementary Physics and Chemistry, compulsory for all boys in town schools of the Transvaal in Standards V. to VII.

Conclusion.—In bringing these somewhat cursory remarks to a close a brief summary may be permitted.

1. The exigencies of the Labour Market demand that white men, technically trained, shall be in charge of surface work on the mines.

2. There is a supply to hand of fairly well-educated youths. Can this supply be more effectively tapped than at present?

3. Some dissatisfaction is said to exist with certain conditions of living and pay under the schemes instituted. Enquiry scarcely reveals much cause for this.

4. Splendid facilities are provided for technical training. These facilities are not taken advantage of as fully as might reasonably be expected. Can nothing be done to bring about a change in this respect?

5. If compulsion is necessary, the question arises, should it come from the mines or from the Government? The author is not prepared to offer any opinion, but mentions that in Europe the opinion is daily gaining ground that compulsory attendance up to 18 years of age must be made the law of the land. After 18 years, which is about the age at which most of the learners start on the mines, if a youth has not got sense to see what is good for him, his employer on his own behalf, may reasonably compel him, within certain limits, of course.

The author wishes to express his best thanks to the Principal and Secretary of the School of Mines, to Dr. Caldecott of the Consolidated Gold Fields, to Mr. Courtenay Shaw of the Rand Mines, Ltd. and to various Government officials, for information supplied and permission, in certain cases, to publish the same.

The views, here expressed, are those of one who is outside the mining industry and therefore can have little more than an academic interest in it. But the schemes discussed seemed to him to be of such importance to the industry and there fore to the profession to which he belongs, as to be worth a little consideration by this Society.

Dr. W. A. Caldecott (*Past-President*): I have much pleasure in proposing a hearty vote of thanks to Mr. F. J. Pooler for his able paper upon one phase of an extremely important subject, and I would like to direct the attention of members to another aspect of training referred to later, which was recently discussed at the South African Institution of Engineers by Mr. William Cullen, one of our Past-Presidents, and by Mr. W. J. Horne, the Government Adviser in Technical Training.

Mr. Pooler's statistics show the regrettable fact that the facilities now available for evening class instruction on the Rand are by no means taken full advantage of by lads in the early stages of their working life on the mines. This is also evidenced in the small number of student members of this Society, as stated in the Annual Report which has just been read, as compared with the number of students of the Institution of Mining and Metallurgy and similar associations in the older countries. As a South African, I may be permitted to say that life over most of South Africa has been and is somewhat easy-going, and that the atmosphere of the modern competitive struggle for existence is largely absent.

In regard to the position of the lads entering life upon the mines, there is no doubt that many conditions are not unnaturally distasteful, particularly during the first year or two when engaged upon subordinate monotonous routine duties, involving hard and unaccustomed physical exertion. This period serves to eliminate those who through lack of aptitude or physique or industry or determination fail to adapt themselves to such conditions, but to those who survive the probation it may be pointed out that theoretical instruction and mental training in classes, even at the sacrifice of a little amusement, is a preparation for rising in the future by assisting them to prove themselves superior to their competitors, since average knowledge and energy here as elsewhere reap eventually only the average reward.

Position and pay at twenty years of age matter comparatively little, but at forty they are all-important, and hence the favourable conditions for acquiring knowledge and experience through study and varied training in the earlier years should be fully utilised. Adequate theoretical knowledge insures a complete survey of the field involved being taken, but only experience can give due relative weight to each factor concerned in arriving at a correct conclusion in each case. As the American saying puts it: "Theory alone is worth nothing, practice alone is worth five dollars per shift, and the combination may be worth anything." It cannot be contended that study and class instruction alone will turn an incompetent man into a capable one, but it certainly makes a good man better, and assists all who take the trouble to master such of it as pertains to their daily work.

In the past education in South Africa beyond the elementary stage has been mainly directed towards entrance into the law and civil service, or banks or offices, with occasional candidates for the professions of surveying, teaching, medicine, or the church. However necessary these occupations, the prosperity of this country depends upon its products, whether of mining, agriculture, or manufacture, and the greater the percentage of the population engaged in the actual production and exploitation of the natural resources of South Africa, the more prosperous will we become. Conditions of life here are inevitably changing and in this industrial community with its hard and strenuous competition, the worse equipped inevitably fall behind in the race. It should be the desire of all of us who have been brought up in the country, or who have thrown in our lot with it, to have as good men trained here as can be imported. Otherwise the deplorable state of affairs will continue whereby the bulk of our skilled artisans came from abroad, and yet the bulk of the lads here are growing up without being systematically trained for anything in particular, and consequently take an inferior position to those who can prove themselves to be of greater value to their employer. In addition, the competition in trades of coloured labour, already so serious at the Cape, is bound to increase with time and the supremacy of the white man is dependent upon his keeping ahead in skill as well as in sufficient numbers. The unskilled white man is in fact between the upper and the nether millstone of his more skilled competitor on the one hand, and on the other of the native who is far cheaper as a muscular machine.

"Blind alley" occupations, such as errand and office boys, taken up largely through lack of parental foresight and on account of the relatively high wages, leave young men adrift at twenty unskilled and too old to learn a trade. As the author suggests, vocational training up to a certain age in some craft or industry appears bound to become compulsory in time, just as is elementary scholastic instruction at present, if the white race is not eventually to be submerged. In this regard it may not be generally known that a Government Trades School has lately been started in Johannesburg, and it is to be sincerely hoped that it will not only prove successful but expand by converting lads, who are physically and mentally developed enough to fully profit by the instruction given, into young men who can eventually hold their own in their respective crafts.

Mr. Jas. E Thomas (*Member of Council*): I am afraid I cannot agree with Mr. Pooler regarding the question of learners being subject to 24 hours notice. In a great number of the large shipbuilding and engineering establishments on the Clyde it was found that the system of premium apprentices did not pay, and that these apprentices did not get the knowledge of their business that was expected of them. Apprentices were then signed on at a nominal wage 1¼d. per hour—in my own case—and were subject to 24 hours notice. It was also made a condition that apprentices should attend evening classes on subjects pertaining to their business, and the employers made a point of enquiring as to the progress and industry of each apprentice at class work.

Learners on the mines here are paid sufficient to enable them to live decently and also to afford to pay class fees. I do not know of any other place where this is done, and I cannot see any great hardship in their knowing, that unless they try to do the work allotted to them to the best of their ability, they must make room for someone who will. I have now had nearly four years experience of the learner system and I must say that it has turned out much better than I anticipated. I think the percentage of capable men we have secured is greater than even Dr. Caldecott expected when he introduced the system.

The most trying time for a new learner is undoubtedly his first month at the work. Those who "stick it out" for that time usually turn out quite useful fellows and take an interest in their work. The others generally find that they have mistaken their vocation and leave.

Mr. E. J. Laschinger (*Vice-President*): This question of technical education and training schools, means by which the younger generation can improve themselves and become fit members of society, in the social conditions of the future, is one which of course appeals to every man and one which is stirring South Africa to-day very deeply. As has been mentioned before, a paper has been read by Mr. Cullen before the S. A. Institution of Engineers, and we also had a contribution by Mr. Horne, one of our experts on technical education. This matter has been taken up very thoroughly by the Government, and is also being discussed in the public press, and I think it is very much to the fore amongst the public of South Africa.

It is impossible for us to adopt the systems of other countries at once—we have to adapt them, and if we do not find the encouraging results which we expect, I think we ought to give our boys and young men in South Africa generally a chance. I speak feelingly, having been a member of the Technical Education Commission under Lord Milner's Administration and have been very much disappointed at the results so far achieved on the Rand—but I cannot altogether blame the young men themselves. I would not like either to lay all of the blame on educational institutions. We are all groping our way, and I am sorry that we cannot get at the young men. The young men have not been educated up to the necessity of their education and it is naturally a matter which has been settled in older countries by the parents having instilled into the minds of their youngsters from the cradle upwards the necessity of their improving themselves and doing everything in their power to make them improve themselves. That condition we have not had in South Africa generally. If you take the young men that come from other parts of South Africa to improve themselves here you will find that they are the ones that get along, because they have realised that the education which they get and the training which is available for them is going to do them good. With regard to the young men who are native to the Rand, when we consider the cosmopolitan nature of our population, and the various surroundings under which they grow up, I say that the majority have not realised the seriousness of this matter or the necessity for education, and this is the important reason why we have not had the evident success in the educational problem of this country which I think a good many of us expected.

We must not look for too much in the beginning. We have to give the systems time for trial and for improvement. I am confident that the young man of South Africa has it in him when he realises the necessity of certain things, to come up to scratch. I think that is the hope of this country. If that hope were not existent there would be very little use in discussing the problems of the education of the youth of this country. I think if we give the youths of this country a fair opportunity to avail themselves of the educational facilities provided they will come out all right in the end. At the same time I quite appreciate the paper read to-night and the necessity of discussing this problem at the present time.

If we who have gone through the mill, who have had our troubles and difficulties, and who have been compelled to go in for education and improve ourselves in our technical knowledge to the best of our ability—if we do not keep the question alive who is going to?

Therefore I welcome the paper read to-night and all the other papers that have been discussed in other societies and in the press, and I welcome the interest the Government has taken in the problem. for the reason that if those of us who have come to years of discretion and forethought do not keep the question before the public there is very little hope indeed for the advancement of South Africa.

Mr. F. J. Pooler (*Associate*): Dr. Caldecott referred to Mr. Horne and so did Mr. Laschinger. I would like to take this opportunity of publicly recanting certain opinions that I held and advocated somewhat strongly in a journal peculiar to the Transvaal teachers. After Mr. Horne brought forward his scheme for Trades Schools at the Klerksdorp Conference of Teachers three years ago, I wrote a rather strong article against it. I thought he was entirely on the wrong lines. Having seen the local Trades School and the work it is doing, I have reason to think that I was entirely wrong, and I think perhaps this is a favourable opportunity of recanting.

Mr. S. Newton (*Member*): This paper is most acceptable just now.

About two months ago I had three appointments to make in which the rate of pay would be raised from £11 to £18 per month. Naturally those learners who were attending evening classes at the School of Mines would get preference. On asking the learners what classes they were taking, 8 out of 10 replies were to the effect that they were not doing anything this term or that they could not afford to take the classes. One that was attending evening classes was installed at once, and during the next 24 hours the other two vacancies were filled by others who had seen their way clear to join immediately on the news circulating. This shews the energy on the part of the learners! It is a real source of worry to cyanide managers and reduction officers to get these lads to try and help themselves so as to be able to take up a higher position when available and also to fulfil the obligation made when signing on as Reduction Works learners.

I think when the author's paper is freely circulated and discussed it may have the desired effect, and will instil into these youngsters some sort of energy to endeavour to improve themselves.

Mr. C. Toombs (*Member*): The author cannot say what effect the "clean-up" has on the attendance at evening classes by learners and other reduction works employees.

In the cyanide works there are two "clean-ups" per month, each of which in a large modern plant is spread over four days, *i.e.* eight days per month. At these times the learners and shiftsmen are mostly on 12 hour shifts—from 7 a.m. to 7 p.m.—so that practically every other week four 12 hour shifts are worked. Now these men are only human and they can hardly be blamed if continuation classes have little attraction for them after 12 hours at the acid vats or in the smelting house. Many of them have several miles to travel to get to the classes, so what time have they for a change and food?

The classes are started with the very best intentions, then after a little, a lecture is unavoidably missed every other week, which means that the individual loses touch with the class, which in a short time means loss of interest, and finally dropping it altogether. I speak from my own experience as some time back I tried the experiment of evening classes for metallurgical workers on the East Rand, and although the lectures were duplicated to allow for change of shift, the difficulty of the 12 hour shift every other week was a stumbling block which could not be overcome, as perhaps one week a man was on 12 hour shift either beginning or ending at 7 p.m., and the following week possibly on afternoon shift from 2 p.m. till 10 p.m.

My personal opinion is that the general system of learnership on the reduction works is bad. In what other country, or in what trade or profession do we find that a learner can become a full-blown craftsman earning a craftsman's wages inside a year? This I have known to occur frequently on the Rand. What is wanted is an indentured apprenticeship for a term of years, payment being progressive, making it a condition that classes be attended regularly during apprenticeship, efficiency in the works and at the class examinations to be considered when promotion falls due. If classes are to be attended then every facility for attending them must be given as is done in the big engineering works in the British Isles. Co-operation between the college authorities and the mining houses is necessary so as to give the apprentices no excuse. We hear a good deal about the rule-of-thumb man, but the rule-of-thumb methods of employing and advancing learners cannot turn out other than rule-of-thumb men.

The President: It is very encouraging that we should have this animated discussion on the paper and it augurs well for the discussion at subsequent meetings.

In the absence of the author, the following paper was read by Mr. A. Richardson:—

MINE TRIBUTING IN RHODESIA.

By W. Anderson (Associate).

Mines that are let on tribute may be divided into two classes:—I. Developed claims; II. Fully equipped producing properties.

Developed claims are generally held by companies who, owing to the disappointing nature of the development have decided that the erection of a reduction plant and the cost of running the mine on company lines is not warranted. To the tributer however, with his lower working expenses, such claims may offer a reasonable return on the capital expenditure incurred by the purchase of the necessary equipment. Generally, these propositions hold out the greatest chances of success to the man who is, or has been, working a claim tribute in the same district, and who, consequently, possesses an intimate knowledge of local conditions and a plant within easy reach. Producing mines may be let to tributers on account of their having proved unsuitable for company operation, or because of development having fallen rapidly in value, or because the mine is considered to be worked out.

The payment made for a tribute is on the basis of a percentage of the gross output of gold, usually $12\frac{1}{2}\%$, and all claim licences. Occasionally payment is based on the value of the ore mined, the percentage rising with a rise in value of the ore. The tribute agreement also, as a rule, stipulates for a certain minimum footage of development to be done, usually under the direction of the Company's engineer, any further development that the tributers may care to do being left to their own discretion. The lease may be for a period of months or years, the tributers generally having the right of renewal for a further term. In some cases the Company reserves the right to take back the mine before the expiry of the tribute upon the payment of a sum agreed upon.

The equipment of small mines varies considerably according to the nature of the ore. Gravity stamps are in the majority, but steam stamps are frequently used and pneumatic stamps occasionally. Rotary machines do good work with friable ores, but they are unsuitable for hard ores, unless used as an adjunct to the stamp mill. The Tremain steam stamp is greatly favoured by the small worker, its advantages lying in its low first cost, portability, and the ease with which it can be erected. Its disadvantages are, heavy consumption of power, stores and spares. This mill runs well at a steam pressure of 100-110 lb. per sq. in.; the number of blows per stem at 110 lb. steam pressure being about 210 to the minute, with a blow of 800 lb., at a maximum drop of $5\frac{1}{2}$-6 in. An efficient type of sight-feed lubricator must be fitted to ensure steady running, as the piston rings are liable to break if the mill runs dry. Its capacity on hard quartz ores is up to 250 tons per month. It is not generally advisable to run a Tremain on ore of a value much under 1 oz. to the ton, its capacity being limited and its running costs high. The Californian stamp mill is used in units of 2, 3 and 5 stamps, and is usually driven by steam power. Suction gas plants are now being largely used, the cost of power being considerably reduced thereby in districts where fuel is expensive. Where fuel is cheap, the locomotive type of boiler is used; the certainty of steam and the simplicity of its working appealing to the small man against the unknown tricks of an internal combustion engine. The boiler feed water is heated by passing the engine exhaust pipe through the feed tank, but this has the disadvantage of giving hot water to the feed pump. A better plan is to pump the feed water through a series of pipes surrounded by an outer casing (which can be made with a section of old smoke-stack), through which the exhaust from the engine passes. The boiler and steam-pipes are lagged with a mixture of mud and cow dung, which, whatever its disadvantages in other respects, is certainly cheap and excellent for the purpose. Wood fuel may cost anything from 8/6 upwards per cord delivered on the mine. A cord is equal to 128 cub. ft., as cut and stacked, say a pile 8 ft. × 4 ft. × 4 ft. A Tremain with mine pumps, hoist, an occasionally running crusher and cyanide pump, consumes about $4\frac{1}{2}$ cords of dry wood per 24 hours. Charcoal is delivered at 2/-, 2/6 per bag of 65-70 lb.

The cyanide equipment consists of sand treatment vats, collecting vats being replaced by pits. The vats are made of galvanised iron, lap jointed and double rivetted, the rim being strengthened by angle iron. They are placed directly on the ground, packed round with sand, and shovel filled from the sides. Their dimensions are up to 16 ft. diameter and 5 ft. deep, equivalent to a working charge of 30 tons. One solution duplex steam pump is all that is generally provided, but hand pumps are also sometimes installed. Extractor boxes are made of sheet iron with 6 compartments, each compartment being of 2 cu. ft. capacity. These are light, portable and strong. Headgears, bins, etc. are made of native timber. The headgear will be about 20 ft. high, and the engine a single drum hoist, using a $\frac{5}{8}$ in. rope over a 2 ft. 6 in. diameter sheave, and a $\frac{1}{2}$ ton skip or bucket.

The development of a small mine is started by sinking small shafts on the reef, the shaft following the changes of dip. In the case of ancient workings, the footwall of the old stope is followed down to the reef. The usual size of shaft is 5 ft. × 4 ft. or 6 ft. × 4 ft. the broken ground being handled by windlass or whip. These shafts have mostly to be timbered down to the water level; close cribbing being favoured as a good timber boy can cut the sets and place them in the shaft without white assistance. The first prospect level is driven at the water level, about 60 ft. down: and the shafts are continued down to 100 ft. and connected. Levels are usually 50 ft. apart, as the small man has almost invariably little or no development ahead of his mill, and is consequently in continual need of backs. Later on, if the development of the mine warrants it, a vertical shaft 8 ft. × 5 ft. is sunk to cut the reef at a depth from 100 ft to 250 ft.. and timbered in hauling and pump ways with 6 ft. sets of native wood with pole lagging, wire rope being used for hauling guides. The reefs have commonly a steep inclination, 65° being an ordinary dip, and therefore little cross-cutting has to be done to pick up the reef at the various levels. Development is slow work as the average hammer boy, in hard quartz, does not as a rule drill more than a 24 in. hole. 25 ft. to 30 ft. a month may be considered a fair rate of advance in drives in quartz; in winzes, if not troubled with water, the rate is slightly faster.

Stoping is generally underhand, boys drilling 24 in. to 36 in. holes; a 24 in. hole in a 4 ft. stope breaking 0·6 tons if the stope is well benched. In stoping, the want of development has frequently an injurious effect on breaking efficiency, as more boys are crowded on to a stope face than it should carry, making it very difficult to keep good benches. Pillars are seldom left, stulls and timber props taking their place. As the length of the payable ore shoots is never very great, 50 to 100 ft. being common, the areas demanding support are never extensive, and, unless the ground is faulted, there is thus no danger to be apprehended from this practice. The humble wheelbarrow is requisitioned to bring the broken ground to the shaft, the ore being dumped on a flat sheet and shovelled into the skip. In vertical shafts a track is laid, and the bucket is unhooked, run on a trolley to the working place, filled, run back over the shaft upon doors, the rope again hooked on and the bucket hoisted out. Shaft bins are seldom seen, though the advantages of such bins are so obvious that it is difficult to give satisfactory reasons for their omission. Probably the nearness of the levels and the limited strike of the shoots reduces the tonnage to a quantity for which it would not be worth while incurring the expense of a bin.

Development sampling is performed by taking one sample across the full width of the reef every day, and estimating the gold contents by panning. Assay offices are practically unknown, and as 7/6 is charged for each assay by private assayers, very few samples are assayed. In free milling ores, after a few trial pannings and assays a man can fairly well gauge by the show in the pan what the assay of a sample will go. Stope samples are seldom taken. Now and then a dishful of the fine going to the mill is panned, but the daily scrape is considered to be the most reliable recorder of values.

Water, in most mines, is struck at 60 ft. from the surface. If the ground is not fissured and cracked, so that the water does not follow down with the work, the shaft is ringed and the water led into a sump cut in the 1st level. Direct-acting duplex steam pumps, inside packed piston type, are favoured. They are cheap and, for a given capacity, small, and if pumping clean water do not require much attention. Any water that may accumulate below the 1st level is dealt with by hand pumps or small steam sinking pumps. For shaft sinking, a vertical sinking pump, preferably of the outside packed type, is used.

Very little sorting is done, the crusher boys merely throwing out any large lumps of country rock while feeding the crusher. Tremain mills have a three-way discharge mortar box and are fitted with lip plates and also an inside plate recessed into the back of the mortar box. The mortar box is cleaned out fortnightly, the contents being spread out on the table and well worked with the hands, with mercury if necessary. The iron chips are removed by a magnet and put aside until a sufficient quantity has accumulated. They are then pounded in a mortar to loosen the amalgam, sieved through a coarse screen and panned. The plates are dressed at intervals to suit the individual fancy, cyanide being commonly employed; a $\frac{1}{2}\%$ solution HNO_3, or HCl, also makes a good dressing. The amalgam, which generally contains from 30% to 60% Au, is retorted in a cast iron retort, fitted with condenser, placed over an open wood fire. When most of the mercury has come off the fire is increased by packing the wood around and over the retort and bringing it to a red heat. The operation takes from 1½ to 2 hours, and if it is carefully carried out the resulting gold is of a good colour.

The sand to be treated is shovelled out of the collecting pits and into the treatment vats by natives, two emptying the pit and two filling the vat. In treating current sand, no preliminary wash is given; a solution of ·2% to ·25% KCy is pumped on and kept in contact for from 12 to 24 hours. This is leached off and solution is then continually pumped on and leached, the final solution having a strength of about ·1% KCy. A water wash is then run on, leached, and the sand is then shovelled over the side, filled into trucks and dumped. The strength of the final cyanide solution used is kept high in order to minimise the formation of white precipitate in the extractor boxes. While the consumption of lime and cyanide naturally varies on different mines, the consumption of these on a typically clean quartz ore may be taken at 2 lb. CaO and ·8 lb. KCy per ton. The solutions, on a small mine, are run through one extractor box only, which is dressed fortnightly. A simple test to determine whether the extractor boxes are precipitating well or not is to fill a beaker with solution from the tail of the box and suspend a few threads of fresh zinc in it overnight. If the zinc is still bright in the morning the extraction is considered satisfactory. Colorimetric tests are often used to determine the value of the leaching solution, a sample being taken during the final washings. The percentage extraction is rarely ascertainable, an occasional charge and residue assay made, once a month or so, being about the amount of research in this direction. Assay fees are too high to permit the luxury of systematic sampling; fees, in Salisbury, are 13s. 6d. for a charge and 15s. for a residue. The clean-up is fortnightly or monthly, the procedure being as follows:—

Water is run into the head of the box to displace the KCy solution. The top compartment is syphoned down to within a few inches of the bottom, the tray with the zinc removed and the slime swabbed out. The zinc is washed in a tub, immediately replaced in the compartment and covered with solution. After each compartment has been cleaned up the collected gold slime is placed in the acid tub for treatment and allowed to settle. After settlement, most of the water is syphoned off, H_2SO_4 added, and the slime stirred with a pole. As the action dies down more H_2SO_4 is added until its addition ceases to give rise to further action. The tub is then filled with water, well stirred, alum being added to facilitate settlement, and allowed to settle for about 18 hours. It may be noted here that size is to be preferred to alum as a settler; in the above instance the time of settlement was reduced to 3 hours by the use of size. The next water wash settles in 3 hours, and in all about 4 to 8 washes are required. The washed slime is then filtered and dried. Calcination is performed on an iron tray over a wood fire; and the slime is then ground, fluxed, and smelted in a coke furnace. A handy furnace to burn charcoal can be made with a few fire bricks and some round iron rods for the bottom; the pot rests on a fire brick and a jet of air is blown into the fire by a small fan. More care and attention is required with charcoal, however, as the fire has to be constantly fed to keep up the temperature. Usually, the retorted gold is not smelted but simply consigned to the bank by registered post, parcelled up and put in a tea tin. The P. O. charges 1d. per ounce on the weight of the whole parcel; this charge paying postage and registration. The gold is paid for by the bank at the rate of £4 4s. per fine ounce, a small charge being made for smelting and assaying.

Native labour in Rhodesia is cheap, hammer boys being paid at the rate of 30s. to 50s. per month according to the district, the rate of pay being much higher in some parts of Rhodesia than others. Shovelling, tramming and surface boys are paid from 20s. to 30s., and special boys, such as boiler boys, boss-boys and police boys, about £3 per month. Although the rates of pay are, comparatively speaking, reasonable, native labour is often extremely difficult to obtain and the standard of efficiency is not high. The bulk of native labour is alien to Southern Rhodesia, coming as it does from N.W. and N.E. Rhodesia, Portuguese East Africa, and occasionally from Nyasaland. The local boy rarely works longer than 6 months, the majority signing on for 2 months only. His main object is to secure his hut tax, and this once obtained he departs for his kraal and does not appear until the necessity of earning the next hut tax drives him out. The Northern boy stays longer, 6 to 9 months, as it takes him from a fortnight to a month to come down to the mines from his kraal. Having served the agreed term he returns home a Crœsus in the eyes of his comrades and is rich enough to lead a life of leisure for the next five years.

The native labour required for a 2 stamp mill with, say, a 3 ft. stoping width, and keeping development slightly ahead of the mill, is in the neighbourhood of 80 to 100 boys. A 5 stamp mill will require about 150 boys for proper working. Very few tributers can afford to employ white workmen, the two or three partners generally sharing all the skilled work and supervision amongst themselves. The housing accommodation for natives consists of huts built of poles and dagga with thatched roof. The regulations provide for 250 cub. ft. of air space for each native, and a hut 12 ft. in diameter will accommodate 4 boys. These huts cost, with poles and grass near at hand, about £2 10s. each and they have a life of from 2 to 3 years, the thatch having to be renewed each year. When the probable life of the mine justifies the expense, a more permanent type of hut is built of unburnt bricks with thatch roof; the cost of these is about £8. The huts for the white men are very similar.

In comparison with conditions on the Rand, employees on the small mines of Rhodesia work very much longer hours for the same rate of pay. The usual shift is 12 hours, and the rate of pay per shift as follows:—

Miners	20s. to 25s.
Amalgamators ...	25s.

On the Rand if a man is a hand stoper his job is confined to that stope, but in Rhodesia a miner may have to stope, winze, drive, timber, and probably look after the pumps.

Working expenses on a basis of 2 stamps crushing 250 tons, pumping 3,000 gallons and developing 75 ft. per month, are about as follows :

	s.	d.
Stoping	5	10
Developing ...	4	10
Lashing and Hauling	2	10
Pumping	1	7
Milling	3	6
Cyaniding ...	2	6
Sundries	0	10
Total	21	11 per ton crushed.

With a Tremain mill the total cost per ton would be from 2s. to 3s. higher. No white wages are included in the above estimate, as a partnership is understood. If the mine is run with paid white employees the wages of two or three men will have to be allowed for additionally.

The man who wishes to start small working or tributing must be capable of doing every job on the mine from drill sharpening to amalgamating ; so that no portion of the work can come to a standstill through ignorance on his part. If there are two partners on a mine and one only understands, say, the pumps, then if that partner should become ill and the pumps get out of order, more damage may be done in a few hours through the drowning of the pumps and the consequent flooding of the mine than, possibly, can be rectified in as many weeks. The small worker, or tributer, must also be a man of resource, and he generally is, or soon learns to be if he wishes for success. As an instance of how difficulties are to be tackled, the following example may be of interest. A small worker developed his claims by means of a vertical shaft until the ore reserves warranted the erection of a mill. The plant was proceeded with and at the same time the shaft was continued in order to strike water. The mill was completed, but no water had been struck in the shaft, although the adjacent small mines had struck water at a much higher level than he had sunk to. He then decided to obtain the necessary water for milling from a small spruit 700 ft. from the mine. The problem was how to do it in the cheapest way. The distance was too far to take steam from his boilers, and the cost of buying a boiler and pumps to install at the river was prohibitive. He eventually solved the problem by utilising an old hand-power double-barrel bucket lift pump, driving it from the line shaft by a $\frac{3}{8}$ in. wire rope. A sheave was keyed on the end of the line shaft and the rope run over pulleys down to the pump. Wooden battens were clamped on to the hand wheel of the pump, forming a groove for the rope to work in. For an outlay of £50 (the cost of rope and pipe line), 1,200 gal. an hour were pumped against a head of 40 ft. A boiler and steam pump would have cost £1 per day to run, with the heavy first cost on the top of it.

Prospecting for gold is generally confined to the schist areas, and although in a few cases gold has been located in the granite, these occurrences are relatively unimportant. A prospector's outfit consists of a couple of pack-donkeys, a few boys, tools and stores. The donkeys can each carry a load of 100 lb. weight, the boys each 50 lb. His tools are some picks and shovels, a pan, sieve, and pestle and mortar. A cut down mercury bottle for a mortar, and a thumped up jumper for a pestle, is the lightest and most convenient combination for crushing with ; it also has the advantage of costing nothing. Flour, salt, coffee, and a few tinned meats, as a stand by if game is scarce, constitute the white man's food supply, and mealie meal that of the boys. The supply of meal is kept up by trading with the native kraals with limbo or salt, occasionally with cash. The best time of the year for prospecting is when the grass is burnt, usually about August. At other times of the year the height of the grass makes it extremely difficult to follow up indications of reef.

There are very few, if any, ancient workings not discovered at the present day, and the prospector has now to search for virgin reefs. The ancients went so thoroughly over Rhodesia that any prominent outcrop of quartz that has not been tested by them carries little or no gold. Reefs that have been exposed by denudation since the time of the ancients are rare. The prospector, therefore, generally finds his reef by picking up scattered pieces of gold-bearing quartz, and following these up the rise to points where the quartz is in the greatest quantity below him and very little or none above him. Several of these points can be located and a line joining them gives an idea of the strike of the reef. Trenching across this line for some distance above and below may lead to its exposure.

The climate of Rhodesia is good, and though fever is fairly prevalent in the outside districts it is not on the whole serious and can be minimised by care and good food. The man who tries to live cheaply, with his menu chiefly taken up with bully-beef and coffee, is certain to suffer, and he may have to pay more in one month for doctor's fees than he has saved in a year on food. The monthly cost for each, when two or three men are messing together, is about £8. There is no miners' phthisis to be feared, the small mines of Rhodesia being practically dustless.

The life in the less populated districts is, of course, a lonely one, and sometimes even months may pass without one seeing any other white man than one's partner. It has, however, some compensation in its freedom; and though the loneliness is at first almost unbearable to the man from a city or big mining centre he soon gets used to it, and in the end prefers the life to the hampered, conventional one of the towns. Then again, the small man is his own master, and though he is sadly handicapped by want of labour and capital and is, more often than not, in debt to the suppliers of his plant, once he has tasted the sweets of working for himself, he will not, if he can possibly avoid it, work again for wages. There is also the element of uncertainty in small working which appeals to the gambling instinct more or less present in all men.

The President: I would like to propose a vote of thanks to Mr. Anderson for his paper which gives us so interesting an insight into the conditions in Rhodesia.

Mr. J. Moyle Phillips (*Member*): I would like to second that. Having been for some time resident in Rhodesia I am aware of the conditions under which the tributer works.

ACCIDENTS IN TRANSVAAL MINES.

(With Special Reference to Falls of Ground.)

(*Read at November Meeting, 1911.*)

By James Chilton (Member).

REPLY TO DISCUSSION.—PART II.

Mr. James Chilton (*Member*): Mr. Tom Johnson's practical wisdom can always command attention, and his long mining experience makes his contribution specially valuable, but the writer cannot agree that the angle of dip has no effect when considering roof pressures. Generally it will be found that when the force of gravitation is acting at right angles to the bedding planes, we have roof pressures at their maximum. Professor Yates in this *Journal* for June, 1907 gives an excerpt from Principal McNair's address before the American Society for the advancement of Science, on "Deep Mining in the Lake Superior district," and in this address he says, *inter alia*, "At a dip of 38 degrees the pressure normal to the plane of the lode at 5,000 ft. is only 354 tons per sq. foot. As the dip increases, the normal pressure diminishes, at 52 degrees it is 278 tons and at 70 degrees it is 152 tons per sq. foot.

Mr. Johnson thinks an increase in the number of mining inspectors is desirable. While admitting that the Transvaal Mining Inspectors are overworked, it must not be assumed that an increase in their number will lower the accident rate, unless it is intended to put one in every mine, a procedure that would arouse high indignation on Von Brandis Square. In West Virginia, U.S.A., during 1906, only four inspectors were in charge of that district, and it was thought that an increase in their numbers would tend to reduce accidents, so the number was raised to twelve, but it was found that the increased inspectorship had little effect, for accident rates still continued to rise. An increase in the number of inspectors would remedy the dust evil in our mines but would have little effect upon the accident rate.

My critic tells us that he can see little evidence of that hustle which is alleged to influence the accident rate so unfavourably. His lines must have fallen in pleasant places and his experience must be an unusual one. The increased hustle of the last five years can be traced to the advent of the tube mill and the greater crushing capacity of our mills. When our stamps were crushing 4 tons per head, mills could be filled up by Thursday, and mine officials and miners could slacken off, but now stamps crush 8 tons per head, and the mill is never full. To get the additional tons that are necessary now, requires a good deal of energy and organization. Hustle is not necessarily synonymous with confusion and bewilderment, but it does mean the straining of every nerve to secure the desired end. Like Mr. D. Rees he does not see how big mills affect the accident rate as the rock handled per person in big mines is not greater than that handled per person in small mines. Statistics do not support this contention, for after some search the author has made out the following result for 1910—Mines with less than 1,000 employees have an average tonnage of 109 tons per employee per annum, while mines with over 3,000 employees have 151 tons per employee over the same period. This increase of tonnage in our large deep level mines partly accounts for the hustle that is associated with them.

Mr. Johnson says "There is no such thing as an air blast." The author is aware that the word is a misnomer yet he prefers it to "explosive rocks" which is the American equivalent. His assertion that it is due entirely to roof pressure is not borne out by expert evidence. Mr. M. Rzchap in his treatise on the Tauern and Wochein Tunnels says: "As these phenomena (air blasts) also appear in tunnels in which little rock has been mined out, it is evident that the neighbourhood of large empty areas upon which much stress has been laid is not a necessary condition of their appearance. There can be no doubt that the phenomena are due to the sudden release of tension in the crust of the earth." The State Mining Engineer in his report for 1907 says: "This (air blast) action is difficult to explain but is probably due to local tension or unreleased strain existing in the rock and is not directly due to roof pressure." That these effects are so often seen in our deepest mines is probably due to depth. If these strains had existed nearer the surface, probably they would have released themselves in a slight earth tremor, but being buried so far down, only mining operations gives them an opportunity to show their powers.

Mr. Johnson raises the question of the effect of hours of labour on the accident rate. While it is difficult to formulate rules that will apply to all cases, one aspect of the question may be pointed out. Where labour is accompanied by excessive mental or physical fatigue, the lessening of the working hours may reduce the accident rate, that is, if the worker has been before the end of the shift in a state of exhaustion, but if the shortening of the hours of labour leads to an acceleration of speed or hustle, the tendency will be in the other direction. The Rand miner being practically a supervisor, cannot complain of excessive wear of muscle or brain, but as the shortening of hours will accentuate the pace, its tendency will be to increase the accident rate.

Dr. J. L. Aymard raises the question of how far illness contributes to the accident list. This is a question that is difficult to solve but that it does contribute few persons will deny. In miners' phthisis and indeed in most diseases that terminate by exhaustion, there comes a time when the body becomes insensitive to outside impressions and the common dangers of the mine have no message for the benumbed brain, and even if these are received, the body is often physically incapable of responding to the call. When a miner arrives at this stage he delegates a large portion of his work to his native assistants with the results one can easily imagine. Mine officials, having nothing but pity for these poor soldiers broken in the industrial wars, hesitate to enforce the regulations, realising that it very often is a case where the spirit is willing but the body is weak. In the suffering miner, the official sees a picture of his own future condition, and with that "fellow feeling that makes us wondrous kind," deliberately shuts his eyes to breaches of the mining laws. Time after time mining inspectors speak of accidents caused by some senseless and insane actions on the part of working miners, actions so devoid of motive, so aimless, so incomprehensible that doubts are sometimes cast on the mental conditions of the transgressor. In cases like these the author has often wondered if this mental haze is due to some subtle form of phthisis that makes its presence felt in the brainy citadel long before the ramparts are aware of the presence of the enemy. In the fight against miners' phthisis it must not be forgotten that victory will have far reaching effects extending far beyond the question itself, and its results will be seen in a contented community, a greater industrial activity and a lessened accident rate.

Mr. T. G. Trevor certainly touches a vital spot when he points out that an accident is "bad business." Let us look at the question from an entirely cynical and commercial point of view, and we find that in our mines we have a number of muscular machines called natives. Each of these machines brings in a profit of £30 per annum to its company. If some of these are put out of action or completely destroyed, this source of profit at once ceases. The question is—Does it pay to allow these machines to become useless by ignorance, carelessness or neglect? There can be only one answer, yet in 1910 we lost 915 of these muscular profit-producing machines by accidents largely preventable and 1,268 were seriously disabled so as to reduce their profit earning capacity. One of our mines lost 41 "machines" during 1910, another 37. If these had been rock-drills a big outcry would have been made.

Mr. J. E. Vaughan's duty as an Inspector of Mines brings him in touch with many mine accidents and for this reason his paper is specially welcome. Too often are the valuable lessons taught by mining accidents locked up in official pigeon-holes and lost to the industry. Probably if more publicity were given to mining mishaps we should not have to complain so much of the ignorance of our miners. Mining inspectors, while quite willing to impart information privately, hesitate to give the general public the fruit of their experience and investigation. The author would be glad to see officers of the Mines Department entering into these discussions more frequently, though he realises that an almost obsolete Civil Service tradition is against this.

Mr. Vaughan does not consider it fair to make comparisons between the Rand and England, because in the latter country the underground workers are *miners*. Surely my critic cannot be unaware of the fact that 40% of the labour force in England cannot by any flexibility of language be called miners. It is made up of lads from 14 to 20 years of age, reckless, wilful, and often ignorant. Besides these there are men that drift in from other occupations, agricultural labourers, ex-soldiers and unsuccessful toilers in a thousand arts and trades. These all become underground operatives; some may in time become miners, but mining cannot be learned in a month and many remain industrial undesirables all their lives. Probably the number of miners in any English mine does not exceed 50% of the total underground force, the remainder are simply lads or amateurs at mining work.

The appointment of roof inspectors as suggested receives a cold reception from Mr. Vaughan, he preferring to leave roof inspection in the hands of shift bosses. Unfortunately these officials are already overworked and cannot always spare the time necessary to make a careful and minute examination of roof conditions. The mining regulations demand that he should visit every working place in his section, and if he does this he will not have time to make an examination that will sometimes occupy an hour or even two. A roof inspector's duties are not set out in the regulations and if the investigation of roof conditions in any particular stope consumes a large part of his working day, he cannot be accused of neglect of duty, a charge that would at once be brought against a shift boss if he did not visit his working places.

Mr. Vaughan speaks of the difficulty of proving to a magistrate that certain systems of mining are dangerous and should be stopped. Surely the time has come when a mining magistrate should be appointed to try all serious charges under the mining regulations. Against our present magistracy the author has not one word to say. Their decisions usually carry the approval of all moderate men, but no one can deny their inability to give satisfaction in mining cases. It is no disparagement of their legal knowledge to say that they are entirely at sea when dealing with mining actions. The technical language, the uncouth mining terms, the contradictions of experts, and the excursions into underground literature all tend to confuse the issue, and if by a happy chance the magistrate gives a correct verdict both prosecutor and defendant are so worn out with the laws delays, the frequent adjournments, and the wasted time, that the former vows he will never bring forward another case and the latter decides that next time he will plead guilty whatever the charge. It is not enough to have a public prosecutor who understands mining, we should also have a magistrate. On questions of law he could be advised by the clerk of the court, on questions of fact and mining procedure he ought to be so well versed that he could save both the time and the temper of mining litigants.

Mr. Vaughan after declaring that systematic timbering would not be satisfactory, concedes the point aimed at when he admits "that timber should be used more freely than it is now." This is the crux of the argument, more timber arranged on a definite plan or system. If timbering is left entirely to the stoper he invariably chooses a place where he has a natural head, and hitch, regardless of the fact that timber is intended to secure safety. If arranged on some well-thought out method timber would have to be set at definite distances and the old "prop where you like" system would be abolished. The drawing of timber from worked out areas has not been tried—so far as the writer knows—in this district, but the objections to the practice are not insuperable, and the recovery of timber from old stopes will be one of the future innovations on the Rand.

Mr. Vaughan's penultimate paragraph compresses the whole of my paper into three lines when he says "No one will contend that the natural dangers on the mines of the Witwatersrand are so great that they cannot be surmounted with care and skill." Such an expression of conviction from a cautious, experienced Inspector of Mines such as Mr. Vaughan is, is not one of the least points of interest that this paper has brought forth.

Mr. J. Moyle Phillip's long paper on general mining is interesting, but only bears slightly on the subject of accidents in mines. He has come to the conclusion that as far as safety measures are concerned we have not progressed at all in five years. While admitting that safety measures are still not receiving that consideration which they demand, the author sees some evidence of the progress that is being made along various lines to secure the safety and health of the underground employees, and in this fight our Society has played a not ignoble part. Miners' phthisis originally an obscure disease has been brought forth into the light, its causation and prevention has received attention from the highest scientific authorities, and measures have been adopted that bid fair to make this disease as rare as the Black Death. In our mines miles of water pipes have been installed and millions of gallons of water are being sent down to allay the deadly dust, and sprays, respirators, atomisers, hoses, and sprinklers are being used to accomplish the same desirable end. In the matter of miners' phthisis we have undoubtedly progressed, nor is our advancement limited to this subject only. Mine ventilation has received attention, and the huge fans that have been installed in many of our mines will have a considerable effect upon the accident rate, for no person can give his attention to safety measures if his brain is dazed and clouded by breathing in a vitiated atmosphere. Pure air may primarily be an efficiency measure, but it reacts upon safety conditions and tends to produce that mental alertness that is so desirable in underground workers. Sanitation and first-aid appliances have not escaped consideration and in these a great improvement is noticeable. In mining methods we find that back stoping and raising are being frowned upon in high technical circles, because of their dust-producing tendency; and all through there is evidence of the wish to ameliorate mining conditions. All these improvements have been made in the five years mentioned by Mr. Moyle Phillips, and though we have not yet reached the Golden Age of mining, yet this Society will keep on pointing out where improvements could be made though the results may not be seen till after many days.

Mr. M. H. Coombe speaks of the difficulty of enforcing discipline underground, and there can be no doubt that underground discipline is becoming slacker year by year. Our native worker has become so flattered by wily recruiting agents and so pampered by indulgent compound managers that the task of getting a fair day's work out of him is almost impossible. Unlike the white worker he has no dread of dismissal or disgrace. In many of our mines if a native simply puts in an appearance at his working place he must be given a wage ticket. Orders given for his safety he disregards in the most light-hearted fashion, and if any miner backs up his command by physical force he is promptly hauled before a magistrate, and fined. No person wishes to encourage brutality underground, but between brutality and namby-pambyism there is a long stretch of neutral ground. There is probably more corporal punishment administered in an English public school in one week than occurs in a Rand mine in a year. What in the English schoolmaster is necessary discipline becomes callous brutality when performed by a Rand miner. Fines and imprisonment have little effect on the native mind; in fact a long term of imprisonment is looked upon as a romantic episode in an otherwise prosaic occupation. Lack of discipline underground undoubtedly is the prolific mother of accident, and unless the miner has the power to compel obedience, this cause seems likely to neutralise efforts put forth in other directions.

From such a master of safety measures as Mr. W. Cullen we expected an enlightening contribution and we were not disappointed. Visitors to the dynamite factory always come away impressed by the extreme measures to secure safety that obtain there, and the remarkable freedom from accident enjoyed by the Modderfontein factory presents some features that might be copied by the mining industry with profit to themselves and safety to their employees.

Mr. Cullen thinks that a comparison between black and white death rates would be instructive and interesting and it originally was the author's intention to include a table giving these rates but after much thought on the subject he decided that as fatal accidents bring both white and black down to a common basis, and that death levels all distinctions, little good would result from their separation. The author hopes that in breadth of vision this Society has got somewhat beyond the stage reached by the Royal Society of England in 1800 which without a trace of irony, records an accident which not only killed twelve miners, but even "caused the death of a gentleman of quality." However, as a demand has been made for its production the author supplies it, as it may remove some false impressions and will correct an idea that seems prevalent amongst our non-mining population, viz., that the native accident rate far exceeds the European rate.

DEATH RATE PER 1,000 EMPLOYEES.

	Whites.	Coloured.	Chinese.
1905	4·01	5·32	6·82
1906	4·27	4·21	6·00
1907	3·41	3·92	5·76
1908	3·98	5·43	5·66
1909	3·49	4·40	3·76
1910	2·81	4·09	...

From the above table it will be seen that the "ignorant savage" is only slightly inferior to the civilised European in his ability to escape the dangers of mining life, and that the mining Moloch demands an almost equal number of white and black victims. It also shows that the charge often brought against white miners that they ask native assistants to perform duties which they themselves think too hazardous is based upon scanty facts.

Mr. W. Cullen and Dr. D. Macaulay raise the question of the frequent changing of both managers and miners and consider that the practice helps to swell the accident rate. No student of our mining conditions can have failed to notice the alarming frequency of these changes and their disastrous effect in a great many cases. While it is admitted that both managers and men may outlive their usefulness, it cannot be denied that excessive change breeds half-heartedness, inefficiency and accident. Men are supposed to shift because they may; managers because they must. The restlessness of the Rand miner is largely due to the lack of uniformity in the wage system that prevails here. The contract system on the mines is a never failing source of wonder and astonishment. How is it that two miners of equal ability, in the same mine, under similar conditions, differ so widely at the end of the month, one person receiving sixty pounds, and the other perhaps a cheque for 8/-? In admitting that cases like these do occur, we by no means admit that the result is obtained by withholding any part of the miner's legal earnings. Managers and surveyors are as honest as any other section of the community and the frequent charges of peculation brought against them in mining mythology are very seldom substantiated, yet with perfectly honest officials these incidents often happen. The miner receiving the small cheque naturally seeks fresh fields; no protests of innocence, no expression of polite regret; no stope plan or squared paper tracing, will convince him that he has received just treatment. The satisfied miner works on till his cheque comes down and he departs also. Mining officials dread the end of the month for it invariably means a new set of men, with greater anxiety, and more accidents. The trouble is that there are too many large wage cheques and too many small ones. The only way out is to increase the number of moderate cheques. Mining conditions in the stopes and drives change rapidly from day to day, reef bodies widen or become narrower, and stoping widths vary as the ore body changes, foot wall and hanging partings become more distinct or disappear altogether. Some days the ore is hard to drill, on others it is bad to break, faults, dikes and fracture planes make their appearance and are passed over. In the midst of these changing conditions, one thing never changes and that is the price per fathom or foot. Two rates are usually fixed, one for Main Reef conditions and the other for the South Reef, and these prices like the law of the Medes and Persians alter not. If on measuring day the miner comes out well he has worked for it, if badly he is simply unlucky There are working places in many of our mines where men never have and never will make a living wage at the ruling prices, yet they are never changed and month after month men sign on and leave with unfailing regularity. It is quite common to have eight changes in one working place in one month; this number multiplied by the number of mines will give some indication of the number of migrant miners on the fields. The lack of homogeneousness met with in the working places is the secret of the break down of the contract system. Where the conditions are uniform, and free from faults and dikes a miner can make good wages consistently month after month, but unfortunately these places are not plentiful. The author is convinced that the curse of change with its sequel of accident, will not be stayed until we deal with the system of payment underground. Flat fathomage rates should be abolished and a price fixed for every stope regardless of the rates fixed in other stopes or other mines. Let a fair average wage be agreed upon and any miner making 10% above that standard for three consecutive months should suffer such reductions as would bring him down to the average standard wage; on the other hand a miner working where he is unable to make the average wage, should receive an increase sufficient to bring him up to the standard wage. In this way the evils of high and low wages would be successfully resisted and our wandering miners would cease to roam and would identify themselves more strongly with the fortunes of their particular mines. The introduction of the ballot or "cavilling" system might be tried, for by this favouritism becomes difficult and every man gets his chance of the good working places and the bad. Restlessness and change are factors that must be considered when enquiring into our accident rate, for as Dr. Macaulay points out they do not make for industrial efficiency, nor political stability nor mining safety.

The author regrets his inability to reply to all the points raised on the discussion of this paper. To reply adequately would need a whole volume, for some of the criticisms were longer and probably more valuable than the original paper.

In summing up the discussion it is apparent that there is no royal road to the prevention of accidents. for the river of accident is a stream having many tributaries. Probably the following conclusions will represent the verdict of the majority that have joined in the discussion :—

That the accident rate in the Transvaal Mines is excessive.

That nothing in our local conditions warrants this rate.

That underground discipline needs strengthening.

That further regulations are unnecessary.

That deep levels are more dangerous than outcrops.

That mine changes effect the accident rate.

That "hustle" produces accidents.

That safety and efficiency on the mines are not incompatible.

A SYSTEM OF KEEPING UNDERGROUND COSTS AND RECORDS.

(*Read at May Meeting, 1912.*)

By G. Hildick Smith, B.Sc. (Member).

DISCUSSION.

Mr. Moyle Phillips (*Member*): The author is to be thanked for bringing forward a subject which has not heretofore been discussed by this Society, but, I would like to ask—Does he contend that such a system, or similar ones, *have not* been kept on our mines, or is he referring to one particular mine? I ask this question because I have been employed on a good many of our mines here and in different capacities, in each and every one of them without exception, I have found the custom prevalent of signing requisitions for all stores taken underground, with the number or name of the section and level and working face for which such stores were intended. If this custom was not for the purpose of checking such stores and also obtaining the costs of same, then for what purpose was it kept? I quite agree with the author that such a system as he has explained, or one similar, should be kept and I cannot understand any mine captain or underground manager not doing so. Personally, I have always, when in charge, made it a rule, if only for my own satisfaction, to check all stores, boys, tonnage, etc. underground, and I have also found it good practice to duplicate all written orders or instructions appertaining to my work both as regards the office and underground. I find this saves lots of argument and dispute. Whilst acknowledging that great benefits are derived by carrying out such a system, yet I do not think that the author will derive *all* the results which he anticipates.

For the estimating of the actual cost of a stope, or any piece of work done, it is good, as also for the sake of comparison with other stopes, etc., but because it has thus been proved that a stope or certain piece of work costs a certain sum it does not follow that another similar stope or piece of work will cost exactly the same. The conditions of our underground workings are so variable that the cost per ton of ore broken on different levels in the same mine varies to a great extent; even stopes on the same level cannot all be worked at the same cost and the two faces in one stope will often vary to the extent of five shillings or more per fathom. A good foot or hanging or the presence of faults or dikes makes all the difference in a stope, and this is where the *practical* knowledge of a shift boss comes in. The distinguishing between a good or bad stope and the reasons for the difference.

The author has referred to the "hustle" style as the *old* style. I do not agree with him in this respect. It is not an old style but a new style which has been brought into prominence by the introduction of *large* mills and tube mills so none of the old shift bosses, mine captains or underground managers can be blamed for it.

I question whether at the present time all our shift bosses do know whether or not their men require certain stores. This again is where practical experience is of great value for they certainly should know.

The author states that the tonnages are "obtained *correctly* by measurement in trucks each holding one ton of broken ore." I question whether a strictly correct tonnage can be obtained by this method, because there is a great difference in weight between a truck ton of fine and a similar one of coarse ore. A better method, in my opinion, is to let each truck pass over a weighbridge and be automatically weighed and registered. It is a rare occurrence to find even two trucks of ore to weigh the same.

Bonus Scheme.—I think the author's bonus scheme is open to abuse if I may so phrase it, but I take it that he intends the bonus for all-round efficiency in the section of the shift boss who has earned it; because if otherwise then instead of bringing about a better condition of things underground it will tend to quite the reverse. Take the following instances for example, viz.:—

1. Tonnage.
2. Timber.
3. Accidents.

In (1) we have a section where tonnage is more easily broken than in the other sections as possibly the stopes are wider and the foot and hanging more favourable, consequently the shift boss of this particular section will go in for record tonnage and the best part of his time will be devoted to this particular item. To obtain a record he will not be so careful in distinguishing between waste and ore as heretofore with the consequence that the average value of the ore in that section drops with a corresponding drop in the mill assays and output.

2. Another section is in pretty safe ground or has been kept well timbered and packed. The shift boss here will go in for the timber record, and where three props and fifteen foot packs were formerly utilised, he will make two or even one do and reduce his packs to ten feet or less, thus reducing his timber and pack costs but endangering the lives of employees in his section.

3. In this section the shift boss goes in for safety and where one pack would be sufficient he puts in two and also doubles his props, thus adding to the costs of his section without obtaining better results in the way of safety; but still this is his mode of obtaining the accident bonus. Of course these things should not be, but when a definite object is to be attained—in this case it is a bonus—every effort is liable to be made to attain that object, often to the detriment of other equally important objects; therefore I do not agree with the author's bonus scheme.

The meeting was then closed.

Obituary.

The death is recorded with regret of Mr. John Gardner Hamilton, M.V.O., the President of the Transvaal Chamber of Mines, and one of the Society's Hon. Vice-Presidents. Mr. Hamilton, who was in his 54th year, died on Sunday morning the 7th July, after a short illness, and his loss will be greatly felt in mining circles. Born in Co. Mayo in 1859, he was educated for the Civil Service and joined the staff of the British Postal Department. He made a health trip to South Africain 1885, and after two years residence at Aliwal North, C.P., returned to the Post Office in London. In 1889 Mr. Hamilton threw up his service appointment and came to Johannesburg, where he was appointed the Secretary of the United Langlaagte Co. In 1895 he became the local representative of Messrs. S. Neumann & Co. and in 1905 he was appointed the Managing Director of the Consolidated Mines Selection Co, which position he held until his death. Mr. Hamilton was long associated with the Chamber of Mines, and was Vice-President on three occasions before being elected President in 1910 and again this year. He was also a Fellow of the Institute of Accountants and Auditors (England) and had served as President of the Transvaal Society of Accountants. During the war Mr. Hamilton occupied the position of Honorary Civilian Director and Treasurer in South Africa to the Imperial Yeomanry Hospitals, for his services to which he was mentioned in despatches, and received the Queen's Medal and three clasps, King's Medal and two clasps, and made a member of the Royal Victorian Order, and a Knight of the Order of the Hospital of St. John of Jerusalem. He subsequently became the first Lieutenant Colonel commanding the Witwatersrand Rifles and on relinquishing that position became Hononary Colonel of the Regiment. In 1907 he was elected a Progressive member of the Transvaal House of Assembly for Springs and later became one of the Party Whips. He was a Fellow of both the Royal Geographical and Zoological Societies. Mr. Hamilton married in 1893 and had five children, and to his wife and family—who are at present in England—this Society tenders its sincere condolence.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ON THE ACTION OF THE "LUMINATOR" APPARATUS FOR TREATING HARD-WATER.—"In this apparatus the hard-water is allowed to flow over a polished aluminium plate, and when used for steam-raising the treated water is found to produce a soft mud instead of the hard scale obtained when untreated water is used. No difference can be detected by analysis between the untreated and treated water, and the aluminium plate does not lose in weight.

It occurred to the author that the "Luminator" treatment might alter the amount of water of crystallisation contained in the calcium sulphate, which is often responsible for the production of hard scales. A piece of polished aluminium foil was placed in a glass jar containing a saturated solution of calcium sulphate, and after an hour the liquid was boiled down to a small bulk. The crystalline residue after air-drying contained much less moisture than the 20·9% required by the formula $CaSO_4.2H_2O$. The lowest percentage of water obtained was 8·0%. No more water was lost on prolonging the time of exposure to the aluminium.

Time exposed (hours).	Water (per cent).
$\frac{3}{4}$	8·0
12	8·0
142	8·0

The substance thus obtained crystallised in small needles. On analysis the following figures were obtained :—

	Found (per cent).	Calc. for $3CaSO_4$ $2H_2O$ (per cent).
CaO	37·6	37·8
SO_3	54·0	54·1
H_2O	8·0	8·1
	99·6	100·0

It thus appears to be a definite hydrate of the formula $3CaSO_4.2H_2O$.

The action of the aluminium is entirely catalytic. After converting between 3 and 4 grms. of calcium sulphate it had lost less than a mgrm. in weight. The author has had no opportunity of analysing the mud from a boiler using "Luminator" treated water, and so cannot be certain that the production of this substance is the cause of the change in the properties of the scale."—S. SUGDEN, *Chemical News*, April 4, 1912, p. 161. (J. G.)

THE DIRECT DETERMINATION OF SMALL AMOUNTS OF PLATINUM IN ORES AND BULLION.—"By the old method of determining platinum in ores and bullion, the silver-alloy first obtained in the regular course of assay is parted in strong sulphuric acid and the residual metal weighed. This is re-alloyed with silver by a second cupellation and parted in nitric acid, the residual metal being again weighed. Any difference shown between the two weighings is assumed to be, and is called, platinum. Sometimes it is so, and if any considerable amount of platinum be present, there will be a decided difference between the two weighings ; but a slight difference is no real evidence whatever of the presence of platinum. On the other hand, the second weight may equal or possibly exceed the first, even when traces of platinum are present. Again, other members of the platinum group may go into solution in nitric acid more or less. If present, these would be called platinum and escape detection. The method does not provide any direct tests whatever as to the presence or absence of platinum. It is often indecisive, and sometimes gives erroneous results. It is, therefore, quite unsatisfactory.

Being called upon many times to determine platinum in a wide variety of materials, particularly when present in very small amounts, I have realized the disadvantages and defects of this old method.

In an article on the solubility of gold in nitric acid, I briefly outlined a method of gathering a little gold out of a solution containing much silver, which furnishes the basis of an excellent method for the direct and absolute determination of small amounts of platinum, which has the added advantage that the metal weighed may be subjected to suitable tests, to determine that it really is platinum, and to reveal the presence of other members of the platinum group.

In the regular course of assaying for the precious metals, gold is parted from silver by dissolving the silver in nitric acid. If platinum be present in small amounts only, it will readily go into solution in the nitric acid. If now a limited amount of hydrogen sulphide be added to the solution from parting, any platinum present will be precipitated as sulphide, along with some silver sulphide. On filtering off the precipitate (which generally is sufficiently washed by the operations necessary to transfer it from the precipitating-dish to the filter), the moist filter is transferred to a small porcelain crucible, dried at a low heat, and burned off by gentle ignition. This transforms the sulphide precipitate into a metallic sponge, which is wrapped in a small piece of thin lead foil and cupelled. The resulting bead is then parted in strong sulphuric acid, when the platinum will be left as a dark residue, generally collected in spongy form, even when minute in quantity. This sponge, after reboiling in fresh acid, if necessary, is suitably washed by decantation, annealed, and weighed.

Generally, the final metal speaks for itself as being platinum, but if there should be any doubt, it may be dissolved in a drop or two of aqua regia and gently evaporated. The solution obtained may be tested with potassium iodide, or a few small crystals of ammonium chloride may be added, when the characteristic precipitate will show itself. As a further test, this may be filtered off and gently ignited to produce spongy platinum. If the amount of the final metal be considerable, the platinum may be determined by the double-chloride method. Any decided difference shown would indicate the presence of other members of the platinum group, for which direct test could then be made.

For precipitating the platinum and the necessary silver from the parting-solution, a very dilute solution of hydrogen sulphide should be used. One part of a strong solution should be diluted to from 10 to 20 times with water. If the solution of silver nitrate be strongly acid, it should be largely diluted, or it may first be evaporated and then diluted. The very dilute hydrogen sulphide solution should be added very slowly to the silver nitrate solution with constant stirring. The solution is, of course, at once darkened, but there should be no immediate separation of a visible precipitate. The solution should be stirred occasionally, and in about 2 hr. flocks of precipitate should appear. It may be filtered in from 3 to 4 hr., but it is a good plan to let it stand over night if possible.

The amount of hydrogen sulphide required depends, of course, upon the amount of platinum present. If this should be roughly known or suspected, the amount used should generally be enough to precipitate the platinum and from three to five times as much silver. On an entirely unknown ore, I should at first use 1 cc. of strong hydrogen sulphide solution diluted to 15 cc., and reserve the filtrate from the sulphides for retreatment, if necessary. On an unknown bullion I should use 2 cc. of strong solution diluted to 30 cc., partly because bullions are liable to carry much more platinum than any ordinary ore, and partly because the volume of the silver nitrate solution from parting the gold must necessarily be larger. If, however, it is known that minute amounts of platinum are present, it is still necessary to use sufficient hydrogen sulphide to give a silver bead large enough to handle comfortably. For this reason I seldom use less than the equivalent of 1 cc. of strong hydrogen sulphide solution.

It may happen that the final metal shows the yellow colour of gold, due to the fact that exceedingly fine float-gold passed over in decanting the solution of silver nitrate from the gold. In such a case the metal must be re-alloyed with silver and the treatment repeated. When the proportion of gold to silver in the metal being parted is so small that the gold separates in a very finely divided state, it will often save trouble to filter the silver nitrate solution, to separate any float-gold, before adding the hydrogen sulphide.

This method has been used with the utmost satisfaction in determining very minute amounts of platinum in various silver-products directly. Much of our silver coinage, for instance, will show a few tenths of a milligram of platinum in 100 g. of coin. Recently I examined samples from two purchases of fine silver. Very large samples were dissolved in nitric acid. The acid in portions was poured upon the samples and allowed to act at a gentle heat until exhausted. Finally, a small amount of residual silver was removed from the solution and dissolved in a small amount of fresh acid, the solution being then united with the main solution, and the whole evaporated nearly to dryness. It was then diluted to about 250 cc., and 5 cc. of strong hydrogen sulphide solution diluted to 50 cc. was poured in with constant stirring.

This operation concentrated the gold and platinum of the silver into a small amount of sulphide precipitate. This precipitate was filtered off, roasted, and cupelled. The resulting bead was parted in nitric acid, and the gold was determined. The silver nitrate solution was treated with dilute hydrogen sulphide solution, equivalent to about 1 cc. of strong solution, and the platinum parted from the silver by strong sulphuric acid.

These two samples yielded the following results:

	Silver Taken. Gram.	Gold Found. Milligram.	Platinum Found. Milligram.
No. 1,	122·32	0·28	0·67
No. 2,	125·47	0·12	0·18

In case we have a material containing a considerable amount of platinum, the well-known fact that platinum alloyed with silver is not entirely soluble in nitric acid must be considered. In such a case the gold from the first parting in nitric acid must be alloyed with silver and parted in nitric acid a second, or even a third, time, before proceeding to precipitate the platinum from the parting-solutions with hydrogen sulphide.

It is also very satisfactory to use the general method of gathering gold in a precipitate of silver sulphide in determining minute quantities of gold in high-grade silver, such as that produced by electrolytic refining. It is comparatively easy to gather the gold from very large samples of silver, up to 100 g. or more, into a decigram of silver, and then part by nitric acid as usual.

Probably this method of precipitating a noble metal in solution, or removing it from suspension in a liquid, by adding hydrogen sulphide in the presence of silver in the solution, could be used to advantage in determining gold in metallic copper and similar materials."—Frederic P. Dewey, *Bulletin No. 64*, *Transactions of the American Institute of Mining Engineers*, April, 1912, p. 439. (A. R.)

Valuation of Fluorspar.—"The following method is used by the author for the commercial analysis of fluorspar. 1 grm. of the finely powdered sample is heated with 10 c.c. of 10% acetic acid in a small covered Erlenmeyer flask on the water-bath for 1 hour, with occasional agitation, and then filtered; the ashless filter paper and residue are washed four times with warm water, and the filter paper then incinerated and the residue weighed. The loss of weight minus 0·0015 grm. (correction for the solubility of calcium fluoride) is returned as calcium carbonate, but may include also small quantities of the carbonates of lead, zinc, and iron, and calcium sulphate. The residue is mixed in a platinum crucible with an aqueous emulsion of 1 grm. of yellow mercuric oxide, evaporated to dryness, heated to dull redness (in order to oxidise any lead sulphide present), cooled and weighed. The contents of the crucible are now evaporated once with 2 c.c. and twice with 1 c.c. of hydrofluoric acid, then treated with a few drops of hydrofluoric acid and some macerated filter paper, and with a few drops of ammonia solution to precipitate the iron, evaporated to dryness, heated to dull redness, cooled and weighed: the loss of weight represents the silica. The residue is treated with 2 c.c. of hydrofluoric acid and a few c.c. of nitric acid, and the crucible covered and heated on a moderately warm water-bath for 30 minutes, after which the lid is removed and the contents evaporated to dryness. If the residue is not perfectly white it is evaporated again with hydrofluoric acid. It is then digested on the water-bath for 30 minutes with a few drops of hydrofluoric acid and 10 c.c. of ammonium acetate solution (400 c.c. of 80% acetic acid neutralised with ammonia, mixed with 20 grm. of citric acid, and made up to 1 litre with strong ammonia solution), filtered, and the insoluble matter washed by decantation first with hot water containing some of the ammonium acetate solution mentioned above and next with pure hot water, then dried and ignited in the same crucible and weighed. The residue should consist of pure calcium fluoride; this is checked by converting it into sulphate. The above method was found to give satisfactory results with artificial mixtures of similar composition to commercial fluorspar."—E. Bidtel, *J. Ind. Eng. Chem.*, 1912, *4*, 201-2, *Journal Society of Chemical Industry*, Vol 31, p. 332. (J. G.)

Determination of Manganese in Steel.—"The method is a modification of that of Walters (*J. Soc. Chem. Ind.*, 1906, p. 138). 1 grm. of the steel is dissolved in 100 c.c. of nitric acid of sp. gr. 1·2, and the solution heated until brown fumes are no longer evolved, then cooled and diluted to 500 c.c. 100 c.c. of the solution (= 0·2 grm. of steel) are warmed in an Erlenmeyer flask, treated with 15 c.c. of silver nitrate solution (1⅓ grm. per litre) and 1·5 grm. of ammonium persulphate and warmed for about 1 to 2 mins. after the pink permanganate colour begins to develop. The solution is then cooled, treated with a slight excess of sodium chloride solution (6 c.c. of a solution containing 1·4 grm. per litre) and titrated with sodium arsenite solution until the white colour of the silver chloride alone remains."—J. J. Boyle, *J. Ind. Eng. Chem.* 1912, *4*, 202-3, *Journal Society of Chemical Industry*, Vol. 31, p. 338. (J. G.)

Method for the Separation of Thorium.—"Thorium in neutral solution is completely separated from accompanying rare earth metals by adding a boiling solution of sebacic acid; this gives a voluminous granular precipitate of thorium sebacate, which settles readily and is easily filtered. The method is proposed for the separation of thorium on a large scale, since the sebacic acid is readily recovered. Thorium is precipitated almost quantitatively from a netural solution by phenoxyacetic acid, completely, on boiling, by pyrotartaric acid, and from slightly acid solution, by oxanilic acid, the rare earth metals giving no precipitates under these conditions."—T. O. Smith and C. James, *Chem. News*, 1912, *105*, 109, *J.S.C.I.*, Vol. 31, p. 361. (J. G.)

Use of a Mixture of Hydrogen Peroxide and Nitric Acid in Analytical Work.—"In destroying organic matter in analytical work, a mixture of concentrated (65%) nitric acid and 15-20% hydrogen peroxide is very efficient; especially after one or two evaporations with fuming nitric acid. One or two evaporations with this mixture are more than equivalent to a great many with nitric acid alone. In using Lawrence Smith's method of destroying ammonium salts by evaporation with nitric acid, the addition of hydrogen peroxide expedites the operation, especially in the case of ammonium sulphate."—P. Jannasch, *Ber.*, 1912, *45*, 605-7, *Journal Society of Chemical Industry*, Vol. 31, p. 361. (J. G.)

An Unrecognised Source of Error in the Kjeldahl-Gunning Method for the Determination of Nitrogen.—"It is shown that loss of ammonia by volatilisation occurs when the temperature of the mixture of sulphuric acid and potassium sulphate rises above a certain point; such loss of ammonia is accelerated as the boiling mixture becomes more and more concentrated owing to the escape of acid from the flask, and this acceleration is very marked as the liquid approximates to the composition of potassium acid sulphate. In the case of a very concentrated mixture it is possible to volatilise nearly the whole of the ammonia present in a very short length of time, and the loss takes place even when care is taken to prevent the flame of the burner touching the surface of the flask above the level of its contents. To avoid loss of ammonia it is recommended that the amount of sulphuric acid employed should be such that at least 15 grm. remain in the free state at the end of the digestion process. Not more than 3·5 grm. of a cereal or other substance consisting chiefly of carbohydrates, or more than 3 grm. of a substance containing from 5 to 10% of fat, should be digested with the usual quantities of 25 c.c. of sulphuric acid and 12 grm. of potassium sulphate." P. A. W. Self, *Pharm. J.*, 1912, *88*, 384-5, *Journal Society of Chemical Industry*, Vol. 31, p. 361. (J. G.)

Solubility of Zinc Hydroxide in Alkalis.—"Zinc hydroxide is found to exist at the ordinary temperature in at least three modifications: the first is formed when a zinc solution is added to a solution of potassium hydroxide, the second is deposited, often as crystals, when a saturated solution of the first in potassium hydroxide is allowed to stand, and the third (the most insoluble) is obtained by drying the second. The first, which is the most soluble in the alkali solution and the least stable, has the formula, $H_2Zn_2O_3$, and the others have the formula, $Zn(OH)_2$. All behave as very weak dibasic acids; they dissolve in strong alkalis forming zincates, and not colloidal solutions as suggested by Hantzsch (Z. anorg. *Chem.*, *30*, 289). Equations derived from the law of mass action, and tested experimentally, are given for the equilibrium of the hydroxides in alkaline solution."—O. KLEIN, *Z. anorg. Chem.*, 1912, *74*, 157-169, *Journal Society of Chemical Industry*, Vol. 31, p. 281. (J. G.)

METALLURGY.

THEORY OF BLAST-ROASTING OF GALENA.—"The investigation was made to determine the nature of the reactions occurring during the treatment of galena by the Huntington-Heberlein, the Carmichael-Bradford, the Savelsburg, and other "blast-roasting" processes concerning which various and conflicting theories have been suggested. Two series of experiments were conducted: the first with the object of recording the thermal and chemical changes occurring during the roasting of galena in admixture with, lime, silica, litharge and lime, limestone, calcium sulphate, magnesia, ferric oxide and slaked lime, respectively; and the second to determine the rates at which sulphur dioxide was evolved from similar galena mixtures when gradually heated in a current of air, and the temperatures at which large volumes of this gas were produced. The experiments of the first series were made in open dishes heated in a muffle, those of the second being conducted in an electrically-heated tube furnace. From the data obtained, the author concludes that the older theories requiring the formation and subsequent reaction of peroxides, plumbites and plumbates are untenable, as also are those based upon the supposed interaction of calcium sulphate and lead sulphide. In the author's opinion the oxidation of lead sulphide occurs in three stages, the first being a partial oxidation to sulphate and oxide, the second a reaction between sulphide and sulphate, and the third a reaction between sulphide and oxide; in the presence of lime, limestone or magnesia, calcium or magnesium sulphate is formed in preference to lead sulphate, the roasted material then containing a much larger proportion of lead oxide. Silica and calcium sulphate act merely as diluents until the charge attains a temperature of 1000° C., when the sulphates of calcium and lead are both decomposed by silica. Ferric oxide in certain physical states appears to act as a catalysing agent, lowering the temperature of oxidation of lead sulphide and increasing the proportion of lead sulphate formed during the roast."—C. O. BANNISTER *Inst. Min. & Met.*, Feb. 8, 1912, *J.S.C.I.*, Vol. 31, p. 285. (J. G.)

SLIME: ITS DEFINITION. — "I recently came across some very interesting statements regarding the nature of clay, in a book entitled "The Soil," by A. D. Hall, a noted English authority on agricultural matters. As Mr. Hall's statements are of considerable interest, I shall quote at length, putting in italics the particular passages to which I wish to draw attention:

'From a chemical point of view, all clays are found to consist largely of kaolinite, the hydrated silicate of alumina, the other materials present consisting of extremely fine grains of quartz and other weathered minerals, together with more or less oxide of iron.

'Water in which a little clay has been rubbed up remains turbid for a long time; days and even weeks elapse before the particles settle down to the bottom—indeed, however long the liquid may be at rest, a slight haze or cloudiness may be observed within it.

'Schloesing has drawn a distinction between the part of the clay, amounting to 1 or 2% only of the whole, which persists in remaining suspended, and the portion which settles down: he has called it 'colloidal clay,' and attributes many of the typical clay properties to the jelly-like medium of colloidal matter in which the other defined particles of the clay exist. He also associates this colloid clay with such typical colloids as the highly hydrated forms of silica and organic bodies like starch and gum, which, though they appear to be truly dissolved, yet cannot diffuse through a membrane, and form, on drying, hard, non-crystalline masses with much shrinking and a characteristic fracture. *But later researches on colloids seem to show that they are not essentially different from suspended matter; they consist of particles too fine to settle down in water, or to be arrested by a filter even of porous porcelain, but still sufficiently coarse to show their presence when a strong beam of light is passed through the liquid, as is not the case with bodies truly dissolved. From this point of view the 'colloid clay' would only represent the limiting states of fineness, differing in degree, but not in kind, from the other clay particles.*

'*It seems impossible to ascribe to clay any distinctive chemical composition, or to draw a sharp line of separation where sand ends and clay begins. The characteristic 'clay' properties are due to the fineness, and not to the nature, of the properties composing the material.*

'The following considerations support this view: A sample of crude kaolinite rock, as dug in Cornwall from the surface of granite, was powdered and passed through a sieve retaining all particles above 0·2 mm. in diameter; the remainder, which consisted mainly of kaolinite with a little mica, was further separated by sedimentation with water into four fractions:

Fraction.	Approx. size of particles in mm.	% original material.
1	0·2 to 0·05	22
2	0·05 to 0·01	39
3	0·01 to 0·005	21
4	Below 0·005	20

'Of these fractions, the first contained all the mica, the others were practically pure kaolinite, yet the second fraction showed none and the third very little of the characteristic properties of clay; when dried, they fell or could easily be rubbed into a fine powder; only the fourth and finest fraction dried into a hard, coherent mass. Thus we have material which consists entirely of kaolinite, and yet is not clay; such as we see in natural deposits of Fuller's earth, which consists of kaolinite, but possesses no plasticity, and falls on drying into a fine powder.

'*Furthermore, if pure quartz be reduced to a very fine powder by grinding in an agate mortar, and then separated by sedimentation, it will be found that the finest grades of particles,* 0·01 *to* 0·005 *mm. and below* 0·005 *in diameter, show all of the characteristics of clay, and cohere strongly on drying.*

'*Thus we must conclude that clay is not necessarily kaolinite, but is rather a general term for very fine grained material: we must take an arbitrary dividing line, and it is convenient to classify all particles below* 0·005 *mm. in diameter as clay, all above that size as sand.*'

It is hardly necessary to make any comment on the foregoing statements other than to call attention to the rather remarkable claim made that finely divided quartz, under certain conditions, possesses properties identical with colloid clay. If this be true, as this writer claims it is, it would seem that the opinions generally held by engineers regarding the nature of slime will have to undergo a considerable change. Mr. Hall has opened up an interesting field for investigation, and I would suggest that some of the members of the society try to confirm his statements by laboratory experiment. It will be noticed that he defines clay as material of a certain degree of fineness. Those who are inclined to use a similar definition for slime will find a strong argument to support their side of the controversy in Mr. Hall's very interesting and instructive book"—P. C. K. O'Brien, *Mining and Scientific Press*, Feb., 1912, p. 66. (H. A. W.)

Vanadium Ores and Uses.—"Vanadium is the master weapon of the steel metallurgist declared Professor J. O. Arnold, the famous British metallurgist, ten years ago. The advanced steel makers accepted his dictum and encouraged their experimenters to fresh efforts in the commercial application of the element. But shareholders are prone to consider profits rather than superiority of product. The public was not ordering vanadium steel, though there was an ever increasing necessity for stronger and safer steel. The price of vanadium alloy was very high, its quality irregular and its supply uncertain.

Vanadium needed a genius, a man who should combine the attributes of the enthusiast with remarkable gifts as a chemist and analyst and metallurgist, and withal a practical steel maker. J. Kent Smith, of Liverpool has so eminently filled this difficult role that he is universally conceded to be the court of last resort on all matters appertaining to vanadium. In fact, the history of vanadium in the past decade is largely a story of his struggles, and triumphs, in the making of the various grades of vanadium alloy that his experiments proved to be necessary, in the production of the multitudinous types of vanadium steel and iron, and their proper heat treatment. His figure is known in all the leading steel mills of the old and new world, and he had personal charge of the initial manufacture of vanadium materials in those mills His lectures before scientific, trade and popular assemblages as his articles in the press have been potent in awakening the public to demanding from their suppliers those safe, strong and economic metals in which the use of vanadium is imperative.

The growth of the vanadium industry may be appreciated when it is understood that in 1906 the total value of vanadium alloy produced did not exceed $50,000 and that the 1911 production was worth over $1,500,000.

Like gold, vanadium is very widely distributed in nature, but as its market value is only about one third that of silver, commercial interest is confined to deposits containing more than 2% vanadium oxide, V_2O_5.

Del Rio, of Mexico, discovered the element in 1803 in the lead ores of Zimapan, but yielded to the fallacious arguments of French savants that it was an impure chromium, and the honour of discovery was given to Sefstrom, of Sweden, who found it in the Taberg ores in 1830.

The principal ores containing vanadium are patronite, carnotite, roscoelite, vanadinite and asphaltite.

Patronite is a sulphide, greenish black and slaty, containing 35% vanadium sulphide, 35% free sulphide, 5% iron sulphide, 35% free sulphur, 15% silica, and small quantities of molybdenum, nickel, alumina, lime, etc. It is found in a lens-like vein or dyke formation, with a dip of about 65°, is over 300 feet long by 4 to 6 ft. in thickness, and of unknown depth. It is situated at Minasragra, Cerro de Pasco, Peru, at an altitude of about 16,000 feet above sea-level. As it is a free-burning ore, it is easily calcined, and loses 45% of its weight in the process. The calcined ore contains about 52% vanadium oxide (V_2O_5). The earth surrounding this patronite vein is impregnated by solutions of oxidised vanadium. Possibly 5,000 tons of ore have been shipped from this deposit.

Carnotite is a complex vanadium and uranium mineral of variable content, the ordinary commercial ore from the large deposits containing about 4% vanadium oxide and 2% uranium oxide. It seems to be an impregnation of the sandstone and the stratas are flat, or blanket, and are from 1 to 4 feet in thickness. The ore is usually of a vivid canary yellow colour, but in proportion as the vanadium content predominates the colour changes to a dark yellowish green or in the calcium vanadates to a bright red, like soft iron oxide. The deposits in the Paradox Valley cover an area over 50 miles in length. Carnotite deposits are also found in Grand and Emery Counties, Utah, and in Routt and Rio Blanco Counties, Colorado, also in Portugal and in Turkestan. About 2,500 tons of ore have been shipped from the Paradox Valley, Colorado, deposits.

Roscoelite, or a material closely resembling it, occurs in blanket deposits in the Lower San Miguel mining district of San Miguel County, Colorado. The deposits are persistent, thick and easily worked. The ore averages about 3% vanadium oxide, and is milled close to the mines. The production has been close to 10,000 tons.

Vanadate of lead and vanadinite are found in various lead-ores. Near Cutter, New Mexico, are deposits averaging about 1% vanadium oxide and 5% lead, but the ore seems difficult to concentrate mechanically. Quantities of vanadinite of somewhat higher grade may be found in mines near Tucson, Arizona, but these have not been operated for vanadium. The lead ores from Zacatecas, Mexico, contain over 5% vanadium oxide, but much of the ore contains arsenic, which is a disturbing element. The lead-ores from Chihuahua, Mex., are pure and easily concentrated to 18% vanadium oxide. Hand picked ore from northern Spain yields 10% vanadium oxide.

Asphaltite or vanadiferous coal is found at Page, Oklahoma and at Palaside, Nevada. It has low specific gravity, is lustrous and brittle, and of very variable vanadium content, ashes from the same deposits ranging from 2% to 25% V_2O_5. The asphaltite deposits of Peru, notably those of Llacsacocha, are important, and it is possible they will be exploited in the near future. A ton sample of ashes received recently gave 17% vanadium oxide, while small samples have given as high as 70% V_2O_5. Asphaltite is found also near San Rafael, Argentina.

It may be observed here that comparatively few chemists are qualified to make an accurate determination of vanadium, and that it is advisable to send samples to a reputable public analyst. For a mere qualitative test, to show the presence of vanadium, the method is quite simple. Carbonaceous materials should be calcined to an ash. (This is unnecessary in materials not containing carbon). Pour a teaspoonful of dilute sulphuric acid over a small amount of ground ore. Heat over flame of a spirit, or other lamp, until fumes arise. Allow tube to cool. Colour of contents should be greenish. Pour in about 40 drops peroxide of hydrogen, which should change the colour to a dark red, or mahogany, which indicates vanadium.

It may well be that Mexico, which gave vanadium to the world a century ago, may find deposits of the mineral of such richness as to give it a leading place in the industry."—Thos. F. V. Curran, *Mexican Mining Journal*, April 1912, p. 17. (R. A.)

Defects in Metal Alloys.—"The knowledge of the properties of non-ferrous metals and alloys has not developed as fast as that of iron and steel. Several causes have contributed to produce this condition. The number of industrial metals and alloys is so large, and their characteristic properties so diversified, that conclusions arrived at from a study of one group would be of little or no value when applied to another.

Many difficulties are met with when using alloys in practice. Some of these are inherent, but some could be avoided by careful treatment. It frequently happens that a perfectly satisfactory alloy fails because of being subjected to entirely unreasonable heat treatment or to mechanical stresses, in the course of being fitted for use. Such treatment is applied in ignorance of the effect likely to be produced, but considering the care that is taken in handling steel, it is remarkable that non-ferrous alloys are often treated as if they were mere inert material, which might be ill-treated without suffering any injury. For instance, while no user of metals would quench steel from high temperature or submit it to prolonged annealing without reference to the purpose for which it was to be used, it is not an uncommon experience to find bronzes made for special purposes treated in such a way.

There are eight distinct classes of defects commonly found in alloys used in machine building. These defects may be classified as:—(1) Sponginess, (2) Brittleness, (3) Defects due to inequalities of composition, (4) Excessively coarse structure due to casting at too high a temperature, (5) Defects due to wrong thermal treatment, (6) Defects due to molecular change other than that produced by mechanical stress, (7) Shrinkage cracks, and (8) Defects due to mechanical deformation.

1. Sponginess.—This is caused by gases dissolved in the molten metal or alloy, which are released at the moment of solidification. This defect is aggravated by over-heating the metal, and may be remedied by re-melting it and pouring it at the proper temperature.

2. Brittleness.—This may be caused by oxides or dross due to lack of care in pouring, or to over-heating, which causes burning, as in the case of the zinc alloys. The oxides of different metals vary as to their ability to separate themselves from the liquid and rise to the surface, and various deoxidizers are used to facilitate their removal in the form of slag. A second source of brittleness is the presence of thin layers of non-homogeneous alloys between the crystals. The microscope is of great assistance in studying cases of this kind.

3. Defects due to inequalities of composition.—These defects are due either to improper mixing of the ingredients, to separation by gravity during solidification, or to segregation in the mould. When the ingredients are not uniformly mixed, the usual cause is the difference in specific gravity of the ingredients. Aluminium, for example, tends to float, and lead to sink. Certain metals, particularly aluminium, also possessess the property of becoming enclosed in a film of oxide as soon as in the molten state, while others have an action similar to that of oil and water and must be stirred to an emulsion and then carefully cooled to prevent separation. With alloys of tin and antimony, or mixtures containing them, a difficulty is met with in that crystals of tin and antimony form and float upward. This tendency is overcome by rapid chilling. Impurities in general, such as lead and bismuth in bronzes will accumulate at the centre of the parts and cause unsound castings.

4. Excessively coarse structure due to casting at too high a temperature.—As the strength of an alloy depends upon the degree of adhesion between neighbouring crystals, as well upon their size, and as these qualities in turn depend on the casting temperature and the degree of chilling, this becomes a complex question that requires much further investigation. A few alloys containing copper and zinc, and especially those rich in zinc, have an extraordinary tendency to form large crystals having little mutual adhesion.

5. Defects due to wrong thermal treatment.—These defects are due to a number of causes, as for instance, to quenching from too high a temperature; heating at too high a temperature or too long during annealing; burning, as in the case of copper and zinc alloys; and unequal thermal treatment of different parts of the same casting and forging. The principal effect of wrong thermal treatment is the production of coarse structure in alloys in general, and a hard, brittle metal in most bronzes. Unequal treatment also produces differences in the structure in different parts of the alloys.

6. Defects due to molecular changes other than those produced by mechanical stresses.—Some metals undergo changes at definite temperatures which involve a complete alteration of their properties. The most striking instance is that of tin, which may pass spontaneously into a gray powdery substance devoid of mechanical strength, at a low temperature. Many alloys have been known to disintigrate spontaneously from no known cause. Some of the alloys of aluminium with other metals fall to powder after a time, while an alloy of 80% of aluminium and 20% of tin breaks up into coarse crystals. It may be that these phenomena are due to impurities in the constituents. This is indicated by the behaviour of alloys of copper and manganese which have been described as disintegrating spontaneously. When prepared from pure metals, however, these alloys are permanent, and the disintegration is entirely due to the presence of non-metallic impurities

7. Shrinkage Cracks.—These may be due to a poor arrangement of the mould, to wrong casting temperature and to the great brittleness of the alloy at a temperature just below that of solidification. The latter condition is the cause of the great tendency of aluminium-zinc castings to crack during cooling.

8. Defects due to molecular changes produced by mechanical deformation.—These defects show themselves as brittleness in cold worked metals; "season-cracks" in brass and other alloys which become visible some time after rolling; and "fire-cracks," which differ from the foregoing only in that they appear during the annealing process alone. It is said that fire-cracks never appear in pure metals, but always in alloys. German silver is particularly liable to this defect. "Chilling-cracks" are produced when metals are suddenly quenched. Cracking is also due to hot working, as most alloys show an increased brittleness at some particular temperature. Many bronzes have only a small range of temperature within which they may be safely worked. This subject has as yet been but imperfectly investigated.

In addition to the defects mentioned, there are some defects which arise in the course of the use of alloys, and which do not fall under any one of the above heads. These defects are due to corrosion and erosion. The question of corrosion of alloys is now being investigated by a committee of the Institute of Metals (Great Britain), because as yet but little data has been collected on this subject. The conditions which affect the liability of alloys to mechanical erosion also demand further study. An interesting case of erosion is that of high-speed propellors acted upon by eddy currents in the water. In some cases the propeller blades are eroded to a depth of as much as 1½ in. in the middle of a blade, while the remainder of the blade is perfect. It appears that different bronzes behave very differently in regard to this action. A definite relation between the microscopic structure and the liability to erosion yet remains to be established."—C. H. Desch, *The Mexican Mining Journal* (reprinted from the Institute of Metals Proceedings, Sept., 1910), Dec., 1911, p. 31. (R. A.)

The Colburn Mill and the Clancy Process.—"The following is a brief description of the Colburn mill, more generally known as the Ajax mill, which has been built in the Cripple Creek district to treat the ores of the Ajax Gold Mining Co. by the Clancy process.

It will be remembered that the original proposals of Mr. Clancy were to prepare a complex solution consisting of calcium cyanamide, potassium cyanide, potassium sulpho-cyanide and potassium iodide, and to pass a current of electricity with a density of 50 amperes to the square foot through the solution, thereby generating, among other compounds, the halogen cyanide necessary to break up the telluride minerals. The solution had to be kept at or near the neutral point. The ore was to be 'slimed' and agitated with the chemical solvents.

It became early apparent that these proposals would have to be modified in several important particulars, notably in the question of the density of the electrical current. It was proposed to use 90 amperes per ton of daily capacity. It was found that to force this amount of current through a solution with a density of 50 amperes per square foot at least 50 volts of effective pressure would have to be used, or in the neighbourhood of 4½ kw. per ton of ore treated. An agitation of 24 hours was necessary and the cost of electricity would have been prohibitive. It was found that by using a current density of about five to seven amperes per square foot from five to seven volts would do the work. Moreover, from 15 to 20 amperes per ton of ore treated, or from 105 to 140 watts, was all that was necessary. The process was, therefore, brought within the range of commercial possibility. Next it was found that it was impossible to maintain the solutions near the neutral point. This was due to the fact that in electrolizing potassium sulpho-cyanide, which is necessary to the process, the solution rapidly became acid and the cyanide was therefore rapidly attacked. The solutions should be kept rather alkaline. It developed that in spite of the alkalinity of the solution as a whole in the neighbourhood of the anodes the solution was always acid and, therefore, the halogen cyanogen could be formed. Iodo-cyanogen is rather slowly attacked by a solution containing 3 or 4 lbs. of alkalinity per ton, and hence the effect of that compound upon telluride minerals could be obtained.

The mill is divided into two parts: the crushing and sampling mill and the ore treatment plant proper. The material is delivered to the chemical plant by means of a conveyor belt 250 ft. between centres. The crushing plant breaks the ore down to about a 10-mesh screen, the ore being delivered to the mill by a 6 ton ore car operated by two 5hp. crane motors, and receiving its electric power through a third rail. The material is dumped directly from the car into a 5-C Symons crusher, which breaks the ore to about 1¼ ins. From the crusher the ore is elevated and delivered in two conical sheet-iron ore bins, one of which is provided with a Fairbanks scale with a capacity of 35 tons. This weighing bin is for the purpose of weighing lessee ore. From the ore bins the material is passed through a magnet and is elevated to a set of Colorado Iron Works impact screens. After screening the ore is delivered to three sets of Allis-Chalmers rolls, 16 × 36 in. The screens used are ⅝, ¼ and 10-mesh. All the material that passes through a ¼-mesh screen passes over a 27 in. Snyder sampler which takes a cut of 5%. The sampler cut is delivered to a sampling plant which presents no novel features. The finely crushed material is delivered to a 16 in. rubber conveyor belt, already referred to, and is carried up a 20° slope to a vertical elevator which delivers the crushed ore to three large round steel ore bins situated at the head of the chemical treatment plant. The combined capacity of these ore bins amounts to 600 tons. Each bin will store a different character of material, and they are arranged so that they will each feed a separate classifier. The classifiers are of simple design, merely consisting of a trough, in the bottom of which a 12 in. screw is operating.

The fine slime washed out by the classifiers is delivered immediately to the agitator system and the over-size, which is diluted to a proper degree of thickness, feed a Stearns-Roger spiral fed trunnion tube mill 16 ft. long and 5 ft. in diameter. There are three tube mills, each with its own closed cycle and each crushing to a different degree of fineness adapted to the character of the ore which each tube mill is receiving.

The crushed pulp is elevated by air lifts of special design both from the delivery end of tube mill and from the slime discharge of the classifier. The ore is crushed in solution.

The slime goes to a special design agitator. There are five agitators, each of a capacity of from 80 to 100 tons of dry slime. The pulp will have a ratio of one ton of ore to 1·5 tons of solution. In the agitators there are 20 electric baskets for the purpose of electrolising the solution. Each electrode basket has an anode surface of about 4 sq. ft. and a capacity of 35 amperes. The electrodes are carried on heavy steel rail bus bars. The current is furnished by a General Electric 15-kw. generator arranged to deliver 3000 amperes at 5 volts, or 1500 amperes at 10 volts. By means of field resistances the voltage of the generator can be regulated to anything between five and ten, with corresponding amperages. The delivery of the current is regulated to each agitator by means of a Westinghouse switchboard.

The ore which will be crushed to 150-mesh screen approximately is agitated for 24 hours with low-pressure air delivered by the mill compressor, although the air system is connected with the mine compressor, giving air at 100 lb. in case of emergencies.

After agitation the pulp is delivered directly to the loading tank of the Moore filter plant with a filtering area of about 9500 sq. ft. and a capacity of about 200 tons of dry slime daily.

The Moore filter plant offers some novelty in that the crane is entirely operated by hydraulic power, both for lateral and vertical motion. The hydraulic cylinders of the crane work at a pressure of from 250 to 300 lbs. Water is forced into a 4 × 12 ft. accumulator by means of a belt-driven Deane triplex pump with plungers $2\frac{3}{4}$ ins. and making a 6 in. stroke. This pump consumes about 5 hp., which suffices for the operation of the crane. The power consumed by this filter plant compares favourably with any installation now in existence. The use of the equalised load furnished by a hydraulic pump compares favourably with electrical cranes both for economy and ease of operation.

The clear gold solution delivered by the filter system is precipitated by zinc dust fed intermittently by means of a tripping arrangement operated by the flow of gold solution. The precipitated gold is collected by two Stearns-Rogers special filter presses with 24 × 30 in. plates. Each press contains from 24 to 30 plates.

The slime delivery from the Moore filter plant will be made by a screw conveyor working in the bottom of the discharge tank. The slime will be conveyed from the mill in a large bucket carried on a cableway and moved by a small electric hoist. On the cable-way trips are provided so that automatically the bucket will dump in a different place at every trip.

It is a little early to give exact figures regarding the percentage of extraction and the cost of treatment, but it is believed that when the mill settles down to steady work the extraction will leave less than $1 per ton in the tailings, and the consumption of chemicals will amount to about 12 cts. per ton, and the cost of treatment will be $1·50 per ton." —A. W. WARWICK, *Transactions of the American Metallurgical Society*, March, 1912, p. 79. (H. A. W).

THE TREATMENT OF COMPLEX ZINC-LEAD ORES, AND THE "BI-SULPHITE PROCESS."*—"The present article cannot be considered as a premature announcement of the technics of the 'Bi-sulphite Process,' as the original patents were filed several years ago ; throughout the intervening period other patents have been taken out ; the process has been continuously developed, through the small to the large working scale, during which every step in the operations has been fully tested ; simple and effective plant items have also been devised to give practical effect to these. As most processes require to pass through a technical development of shorter or longer duration, the years taken to establish the present one cannot be considered an unduly lengthy period having regard to the ground covered.

In an article concerned with a return to chemical methods for complex ore reduction, a few words may be permitted on what is frequently termed the 'Zinc Problem.'

For present purposes the types of methods available for such ores may be broadly classified as 'Direct Smelting,' 'Concentration,' and 'Chemical.'

Direct Smelting Processes.—It is almost unnecessary to say that simple zinc ores are dealt with by distillation with carbon in a battery of cylindrical clay retorts ; whether sulphide or carbonate, such ores require preliminary calcination to ensure the presence of zinc in the form of a dense oxide. To be available for such treatment the ores handled should preferably average 50% of metallic zinc contents ; actually they may vary between 45% and 62% before being mixed to a distillation charge.

The presence of lead in proportions of more than 8% of the whole involves great difficulties in distillation, mainly in regard to the rapid destruction of the retorts ; the direct production of spelter from heavily leaded zinc ores is thus economically barred.

Similarly, if such ores be smelted for the recovery of lead-silver-bullion in cupola or reverberatory furnaces the presence of zinc to the extent of more than a few per cent. gives rise to large smelting losses. The zinc is partially volatilised as fume, carrying lead with it, while the slags become so sticky from the inclusion of zinc oxide that smelting becomes impossible or unduly costly. Both smelting operations are limited in regard to either metallic impurity by a scale of penalty charges which render them inapplicable to the usual run of complex ores.

Efforts have been directed on several occasions to overcoming the difficulties due to an excess of lead during the direct spelter distillation.

Physical or 'Concentration' Methods.—These include the ordinary ore-dressing operations and plant, such as vanning and gravity separations ; also magnetic separation, and the various types of oil (or 'surface tension') mineral recovery processes which are eminently successful within their special lines upon certain complex ores.

The limitations, as well as the advantages, of concentration are definite. No ore-crushing device can ever be sufficiently perfect to completely divide a complex mineralised ore mass into particles of the pure minerals, far less will such be of uniform size ; the particles which it is sought to classify and separate, whether of the valuable mineral or the worthless gangue, will be of every size, from the aperture of the screen used during crushing as a maximum down to a minimum of microscopic fineness. The finer portions, which resist all mechanical means for further separation of their contained mineral values, are generally called 'slime' ; it is mainly the larger particles (as a rule those which readily sink through water) or 'sharp,' which lend themselves to physical differentiation. This 'sharp,' however, is rarely composed either of clean mineral or gangue particles ; fragments or films of one mineral may still remain adherent to the other, and of both to gangue, as may be readily seen under the microscope ; whilst even the cleanest mineral particle may contain another mineral chemically combined within its mass. Thus in Broken Hill ores a proportion of the silver is in chemical combination with the galena, whilst a considerable ratio remains in combination with the ferruginous blende.

Thus the mechanical concentration of a complex ore of this type necessarily results in a series of products each of only approximate purity, and a sharp segregation of values into separate parcels is impossible.

Chemical Processes.—Chemical processes seek to avoid the incompleteness of the 'proximate' separation of minerals from each other and from gangue, by withdrawing one of the metallic constituents chemically (*i.e.*, molecularly), usually by solution methods. In the case of complex lead-zinc ores, zinc is the keystone of the arch which is usually sought to be withdrawn ; and as a blende is by no means a difficult mineral to render soluble, chemical methods have always received great attention from metallurgists interested in the reduction of ores in which zinc is the main difficulty.

Many hundreds of solution processes have been proposed and patented, and not a few have made considerable practical strides toward a working success; none have yet achieved it, if we except the method under review.

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*The Mining Journal, London.

The two most noteworthy attempts hitherto made in this direction are those of Ashcroft and Ellershausen. On both large sums of money were expended, and each nearly attained to great commercial success. It is to be regretted that these processes are too often recalled only as failures, instead of with the recognition of their close approach to practical utility, and of the ground gained to metallurgy by the researches of their inventors and those associated with them.

The point of mutual approach between concentration and chemical methods is obvious. No practical man would seek to pass an ore carrying a large proportion of gangue associated with its values through the various stages of any process, if he can previously eliminate much of the barren material at a lower relative cost. Similarly the producers of mixed mineral concentrates will be likely to welcome any cleaner separation of their metallic contents than mechanical or smelting methods will give them at present, if, whatever the treatment cost, a larger profit be realised. Both types of method may, therefore, be regarded as mutually interdependent rather than competitive, if each prove successful in its particular province. Just as concentration processes have, with simple ores, a province where further differentiation is unnecessary, so complex ores exist where chemical processes derive no help from concentration, and direct smelting is equally inapplicable. The massively mineralised yet finely intercrystallized sulphide ores of the West Coast of Tasmania, which constitute the huge deposits of the Rosebery, Mount Read, and adjacent mines, are of this character. In these the gangue is so small in amount—from 5% to 15%—that its elimination would leave the metallurgical problem practically untouched; a concentration of some ten parts into a product of nine is obviously ineffective. Whilst the Broken Hill ores are now capable of a more or less complete recovery, the even richer Tasmanian ores have not yet been economically reduced, and a rich harvest awaits a successful method. In our present state of knowledge we must look to chemical means to supply this.

In regard to such, some reservation must be made at the outset. Chemical recoveries do not necessarily predicate complete extractions of all the values, for the reason that the reactions involved may frequently be complicated, and elaboration (which usually means expense) must be avoided. A mine reduction plant cannot pretend to the refinements of a chemical works, and the degree of extraction must be regulated strictly by the local balance sheet, and not by maximum recoveries irrespective of cost. If however, we can secure the larger proportion of the otherwise objectionable zinc as a clean lead-free oxide, fit for the spelter furnace, on one hand, and a lead-silver bullion carrying the bulk of the other metals on the other hand, at a moderate treatment cost, and with no intermediate or by-products, owners of the raw material will be more than satisfied. This is what the bi-sulphite process claims to be able to do.

Sulphurous acid is absorbed by moistened zinc oxide with avidity, even from dilute gaseous mixtures; indeed, as a medium for eliminating sulphur dioxide from waste furnace gases zinc oxide has been found preferable to limestone or other bases. Further, this gas forms two compounds with zinc oxide: firstly, the mono-sulphite, $ZnSO_3$, a substance practically insoluble in water; and, secondly, the 'acid,' or 'bi' sulphite, $ZnH_2(SO_3)_2$, soluble in water. Lastly, sulphurous acid is produced in more than the requisite quantity during the oxidising calcination of ordinary raw complex ores, and is thus a solvent agent, self-produced as it were, ready for utilisation. These advantages determine the selection of the sulphite salts, in place of the sulphate, as a means for removing zinc in the process under consideration, such sulphate of zinc as is produced at the same time being allowed to accumulate until sufficient in amount to justify its removal by crystallization or evaporation for the production of zinc sulphate crystals or zinc oxide.

The solution of the bi-sulphite of zinc so obtained may be readily decomposed into insoluble mono-sulphite, which falls as a hydrous crystalline precipitate; and this mono-sulphite, in turn, when heated, undergoes complete dissociation into a dense form of zinc oxide, and into concentrated sulphur dioxide gas suitable for re-use.

These are the main reactions upon which the 'bi-sulphite process' is based. Their application may now be considered in somewhat closer detail.

The complex ore is reduced by dry crushing to a mesh sufficiently fine to calcine readily, and is then roasted in a suitable mechanical calciner to complete oxidation. The small proportion of sulphur remaining after the roast is partly due to a small amount of undecomposed zinc sulphate, but mainly to sulphate of lead. The sulphur dioxide produced is conveyed, wholly or in part, to the solution apparatus, in which the roasted ore admixed with water to form a pulp is exposed to its action.

Simple contact apparatus has been devised which permits a high utilisation of the sulphurous acid, and discharges the treated pulp to the settlement plant; the solution contains the zinc as the soluble bi-sulphite, while the residues carry the lead, silver, gold, etc., together with oxide of iron and gangue.

The zinc solution is now passed to the precipitation apparatus, where the insoluble mono-sulphite is produced and separated. The strong sulphur dioxide gas expelled during the operation is re-used. The main reactions are thus exceedingly simple, require the use of no extraneous chemical whatever, and are conducted in plant of proved types.

The zinc oxide produced is especially adapted to the requirements of the spelter distiller, having the density necessary not only to give a normal retort charge, but to permit of the slow reduction by carbon essential to good condensation into metal. This density of product is to be regarded as an important point.

The spelter produced being quite free from lead commands the extra selling price over ordinary brands due to this advantage. The company owning the British rights in this process possesses its own spelter works.

The only treatment which follows is that of the lead, silver, etc., residues which remain after the separation of the zinc dissolved by the sulphurous acid. This can now be effected by smelting in the usual manner, the resulting products of such reduction being the ordinary lead-silver and copper-iron gold, etc. mattes which are passed on to the refinery, and valueless slag. The lead-silver, etc., treatment has now, therefore, been rendered normal.

Costs.—As to the cost: the preliminary calcination necessary to convert sulphide minerals to oxides at the outset may be reasonably offset against the similar operations necessitated before simple blende ores can be reduced to their respective metals. As only fully oxidised products are sent to the smelters in this first roast, a step normal to the reduction of even simple ores is but anticipated. The succeeding process of zinc separation to obtain a smeltable oxide is light in cost—the actual solution, precipitation, and calcining (mono-sulphite) operations are covered at a low cost. The subsequent steps in reduction of the lead-silver-gold, etc., values in the residues are practically those necessitated in ores of this character largely freed from their zinc, which would have to be incurred in any case.

From careful estimates resulting from many large scale trials, it is reasonably computed that a profit of from £2 10s. to £3 per ton may be looked for from the treatment at the mines of certain hitherto unreducible ores, including shipment of the zinc oxide to Europe and its reduction to spelter.

The chief advantages claimed for this process can be summarised as follows :—

(1) The bi-sulphite process is one for dealing with complex ores, which requires the agency of no extraneous chemical. Fuel, water, and ore are the only raw materials employed.

(2) The process plant is simple, and comparatively inexpensive.

(3) No large stores of liquor are required, as the solutions are handled continuously through the process. The latter can be made automatic to a considerable extent, skilled handling being reduced to a minimum.

(4) No by-products are produced which require special markets. The outputs are lead-free zinc oxide for spelter, and lead-silver, copper-gold-matte products suitable for the refinery.

(5) Where the process is operated abroad, the zinc product can be shipped to European or American spelter producing centres in a concentrated form, immediately suitable for distillation because requiring no preliminary calcination.

(6) The working costs involved by the process itself are very moderate, whilst the subsequent reduction steps are commensurate in cost with the similar operation necessary in the case of simple ores."—*Mexican Mining Journal* (reprinted from *The Mining Journal*, London), Dec. 1911, p. 27. (R. A.)

Clarifying Cyanide Solutions.—" Probably the two worst zinc-box troubles are : (1) the deposition of fine rock particles on the zinc from 'cloudy' solutions and (2) the deposition of a zinc slime from complex zinc compounds in solution. The effect of the latter is principally to coat the zinc shavings making them inefficient, while the former not only has this effect but makes much short zinc, and a low grade precipitate. Naturally the low-grade precipitate necessitates more time and flux in melting.

To clarify cyanide solutions, it occurred to me that the process now in use for purifying city water might be useful. This method consists essentially in adding a small quantity of aluminium alum in solution to the water to be purified ; the 'lime' in solution precipitates aluminium hydroxide, which coagulates the impurities in the water. The mixture is run through a settling basin where the coagulated portion is settled out. The water is then put through a special type of sand-filter to finish the work. The process is used on river water containing the minutest clay slime and the water is so purified as to be clear and practically free from bacteria.

H. W. Miller and I made experiments to determine the effect of the alum on cyanide solution. First a sample of muddy 'box-heads' running about $2·25 in gold and silver, was treated with alum (about 2 lb. per ton, which is a comparatively large quantity). A voluminous precipitate immediately appeared and a slight odor of HCN was noticeable. The precipitate was allowed to settle. In less than an hour all had settled except for a small quantity which floated on the surface.

After settling, the solution was titrated for cyanide and was found to run slightly higher than the original. The settled precipitate gave a good test for zinc. The solution was assayed for gold and silver and was found to have lost about 5c. per ton. Not having facilities for a quantitative analysis at hand, the relative amount of zinc precipitate was not determined.

Considering the results, the following suggests itself for mill work : A long narrow reservoir containing baffle walls similar to those in zinc boxes. If the final compartment is filled with oakum, excelsior, or a similar material the sand-filter can probably be dispensed with. The other compartments should have facilities for drawing off the coagulated material at the bottom. An extra set of zinc boxes would do for experiment. Such a scheme, while it would probably require considerable storage space, would be cheap, as alum costs but $1·75 per 100 pounds."—Chester Steinem, *Engineering and Mining Journal*, April 13, 1912, p. 739. (A. R.)

MINING.

Efficiency of Engineering Applied to Mining."—The writer has been much interested in the application of scientific management to the manufacturing industries along the lines advocated by Frederick W. Taylor, who has been highly successful with his system as introduced into many large estalishments in the United States as well as in European countries.

This system which advocates high wages to workmen and low labour costs to employer. and the accurate and systematic study of time required to perform certain work, has been food for thought to the writer in connection with mining and smelting operations.

Close observations of the metallurgical industries will bring one to the conclusion that there is considerable room for improvement in operating costs, as well as the volume of output obtainable and also with labour problems.

There are few mining engineers who will admit that there are other technical professions demanding a higher training or more scientific knowledge than mining engineering. Yet, as we investigate the extremely scientific application of engineering to the manufacturing of such products as automobile engines, locomotives, etc., we find that mining is far in the background when compared from this essential standpoint.

Neither an extremely dictatorial nor lax management results in efficient operations in any mine and the harmonious relation between heads of departments is impossible under these two conditions. Did you ever notice that a manager is usually a master in some branch of the business, and that that portion is well managed often to the detriment of other branches ?

I would define good management as accomplishing expediently and cheaply exactly the result desired by the owners.

When one understands that a first-class workman can do from two to three times as much work in a given length of time as the average workman, then the possibilities of producing high wages for workman and low labour costs to employer is appreciated.

If a good workman is given an opportunity to work at greater speed and earn from 20% to 100% more money, he is usually glad to get the chance. The percentage that can be so paid should be determined by carefully kept records taken by trained men and not left in the hands of directors or labour unions.

The increase of pay must be sufficient in order to induce the employee to work under this system, and it is unwise to make changes except after having taken most careful records of time required to do certain work.

As soon as a good workman realizes that he can earn all that is consistent with his energy and intellect, he leaves the ranks of shiftlessness and becomes an ambitious workman. This awakens qualities that are often overlooked by the average foreman.

Men usually work as easily as possible, doing only what is absolutely required. If then a highly energetic workman is kept in a gang he gradually slows his pace to the average.

It does not take a close observer at most mines to notice that a workman tramming cars will usually idle away considerable time at the dump in order to rest and give the loader at the other end a rest.

This suggests systematic "soldiering" and is often found to be well organised among workmen. When each operation and the required time for performance is determined exactly, and workmen put on a just basis of minimum wages for a shift's work, and a bonus for extra work within a shift, then soldiering will stop.

As to the adaptability of this system to mine operation let us consider drilling. Under the average usual practice, let us assume that a drill operator draws $4 per shift and his helper $3. The average output being one round of say 10 holes at a depth of 4 ft. each, or 40 lin. ft. per shift at a labour cost of $7, or 17¼ cts. per foot, the crew loading and firing their own holes but doing no shovelling. All of this is usually performed leisurely in a shift. Now by taking time observations with a stop-watch it is possible to determine from a great many observations exactly the time required for each detail of the drill performance during the shift, dividing as example into time to place column, time to mount drill, time to place starter bit, time to crank forward, time to drill length of feed, time to crank back drill, etc., throughout the shift. An analysis of the time so taken will usually develop the fact that from 30% to 70% of time is spent at rest or in hunting up tools, powder, fuse, instructions for work or debating at what angle and method holes should be drilled. Now by making use of the known time required to do each detail of the work, it is possible to so plan each crew's work that from 30% to 70% of the time will be saved and speed of operations increased, resulting in greater output and larger pay for the workman and at less cost to the employer.

It is thus found possible to drill such work as above specified in a half shift, allowing the lunch hour to clear the face of gases and replenish tools so the same, or at least an efficient round of holes at slightly less depth, can be attained in the second half shift.

It is of course necessary to set a definite pace which must be attained in order to qualify at minimum wages, also to average hole depths for efficient blasting. Assuming that we pay a bonus of 12 cts. per foot and that the shift's minimum footage allowed is 50 ft.

By this method the shift has produced 80 lin. ft. of drill hole on which the workmen are to receive 12 cts. per foot for 30 ft. excess footage, or the driller receives $5·80 for his labour per shift and his helper $4·80, costing the employer for labour $10·60 per shift in place of $14, as formerly, resulting in a saving of time and $3·40 per shift, equivalent to 51% increase in wages to workmen and a reduction of 24% labour cost to employer. Equipment must necessarily be kept in excellent condition by a systematic department. It is easily seen then that there are many standards of time units which can be applied to other conditions regardless of the drilling character of the rock.

In regard to shovelling operations as a delay to drill progress, it is often possible to set drills on a stretcher bar and use the rock pile for a footage while putting in the upper holes. However, in other cases the combined effort of shovellers and trammer can clear a space for the column and be working from the outside of the pile, while the upper set of holes are being drilled. A careful study of position of holes will often make it possible to so place them as to keep the breast fairly clear of rock on blasting.

By timing shovelling operations, observations taken to segregate; time (x) in minutes, loading shovel and straightening up; (y) conveying to dumping position and dumping; (z) percentage of time resting per shift. Then the time in minutes to shovel one ton would equal,

$$\left[(x+y)\frac{\text{No. lb. in one ton.}}{\text{No. lb. in one shovelful.}}\right](1+z)$$

It is thus possible to expedite shovelling to a great degree by making use of known units of time for work and applying them judiciously.

The general principles of accurate time study and their application to increasing wages of workmen and reducing labour costs can be applied at greater range.

I will venture the statement that in 90% of metal mines generating their own steam, that a saving of not less than 25% of the combined fuel and labour expense can be eliminated therein. This is rather alarming when it is considered that steam fuel at metal mines is usually more expensive than elsewhere.

The adaption of graphic records to the operation of all machinery such as belt conveyors, Chilean mills, concentrators, etc., will show managers and directors at a glance exactly what is being done, and the detail cost merely by inspection and without elaborate figures or reference to a report too technical for the average business man to understand.

In introducing scientific management into mines, I would advocate the organisation of a planning department whose duty it will be to lay out in advance each employee's work on a card provided for that purpose. This makes it necessary for the workman to only follow instructions and it proves a great time saver. These cards properly arranged serve as a book and cost-keeping record to the exclusion of a large corps of clerks as is usually needed in a mine employing a large force. From these cards which show in detail each shift's work by every workman, graphic charts can be prepared showing the management by inspection just what is going on and being accomplished without his spending valuable time climbing ladders or entertaining foremen in lengthy conversations.

The planning department should be composed of experienced technical men and not left to bosses or officers, and it should have absolute charge of the work and be responsible for results. This department should be located as close to operations as possible.

In changing to this system I would advocate a start along the following lines:

(1) The selection of a man of wide experience, one who has a special ability to observe what is actually going on and win the confidence of all the workmen.

(2) A standardisation of all tools and methods throughout.

(3) The scientific study of time required to do unit work in all lines.

(4) Establishing a card system by which workmen can transmit the required information to the planning department.

In mining it is obvious that labour is the most important and expensive matter to deal with and should receive great study. A strong labouring force is the work of years of organisation and can not be readily replaced. Plants can be rebuilt with borrowed money in a short time.

It is evident then that efficient management must combine high wages and low labour costs. The following four principles as advocated by Frederick W. Taylor seem most necessary:

A Large Daily Task.—Each man in the establishment, high or low, should have a certain clearly-defined task laid out before him. This task should not in the least degree be vague or indefinite, but should be circumscribed carefully and completely and should not be easy to accomplish

Standard Conditions.—Each man's task should call for a full day's work, and at the same time the workman should be given such conditions and appliances as will enable him to accomplish his task with certainty.

High Wages for Success.—He should be sure of large pay when he accomplishes his task.

Loss in Case of Failure.—When he fails he should be sure that sooner or later he will be the loser by it.

In regard to the relation of this system to organisd labour: When an intelligent workman learns he can earn from 20% to 100% more money and be supplied with an opportunity commensurate with his ability, he is not at all anxious to join a union composed of men who earn less and who have nothing to offer to his advantage.

In applying this system it should be undertaken gradually, starting with one or two picked men who soon let the others know that they are getting ahead to their great advantage, which will lead others into it as fast as they are ready.

It is necessary for success with any business for the employer to be fair and fearless, and be interested in the welfare of his workmen and provide for them opportunities of expressing their ideas. This often results in much valuable data and inspires greater confidence between employer and employee.

It is impossible to discuss at length at this writing the full measure of the subject. I hope the thoughts herein expressed will lead others to give the matter further study, in view of improving conditions of both operators and workmen."—GLENVILLE A. COLLINS, *Mining and Engineering World*, April 20, 1912, p. 869. (A. R.)

SAFETY APPLIANCES IN INCLINES.—"In regard to the subject of preventing accidents from runaway cages or skips in inclines, which Mr. Ives brought up some time ago, prevention against such accidents is provided at the Gagnon incline, at Butte, Mont., in exactly the same way that such occurrences are guarded against in hoisting with cages in vertical shafts.

The Gagnon incline has a dip, I believe, of about 70°. In it skips having a capacity of 2½ tons are used, while the men are hoisted through it on cages that are taken off and put on at the proper times. Riding during shift is on the skip itself, instead of on a top deck, as is characteristic of the practice in the vertical shafts at Butte. The shaft is timbered with sets, and on these sets guide timbers are carried just as in the case of a vertical shaft. This close timbering is necessary, owing to the heavy ground that compels close timbering of vertical or incline shafts in the Butte district. Probably that is one reason that the safety catches were put on the cages and the skips at the Gagnon incline. The adding of the guide timbers was a small item of expense. Safety dogs of the cam pattern that is characteristic of cage construction in this country are provided on both the skips and the cages. These are operated by springs that are released by the relief of the pull on the draw-bar just as in the case of the cages and skips running in the vertical shafts. Further explanations are unnecessary.

In regard to the protection against runaways in incline shafts that are not closely timbered, the most feasible solution appears to be the use of cam-shaped shoes, or better, knife-like dogs to be thrown out either to grip, in the case of shoes, or to groove, in the case of dogs, the stringer on which the rails of the track are carried. I believe that a device operating on this principle, was designed by Charles Mason, for use on the inclines at the Baltic mine of the Copper Range Co., in Michigan, but it was never made. It is purely a matter of expense. There is no mechanical reason why dogs cannot be made that will stop the cage by coming in contact with the inside faces, or, for that matter, the outside faces of the rail stingers, only that the latter would require additional shaft area. In the case of the concrete stringers a timber would probably have to be imbedded in the side of the stringer for the dogs to groove; for the safety dog to work eff ctively should work on the grooving principle instead of the gripping principle that is used in most safety catches in the United States. A gripping dog comes on suddenly. It tries to grip and fails, if things be not just right; then tries again, the velocity of the cage increasing all the time. Finally the teeth of the cam catch and the guide is suddenly gripped, with the result that several sets of guides are torn out. On the other hand, with a knife, or a chisel-edge to the dog, grooving begins as soon as the dog comes in contact with the guide. This groove is rapidly but gradually deepened, with the result that the energy of the falling cage is gradually diminished, and the cage is quickly stopped before it has fallen any great distance—generally less than 8 ft. in the shafts where dogs working on this principle are used.

In the case of the safety dogs for use on inclines, it is necessary to arrange them as to shape, so that they will straddle the roller frames. This, though, is no great difficulty to overcome.

The matter of providing against runaways in inclines is simply a question of dollars and cents. It requires the installation of special apparatus at considerable expense. The mine manager would rather take the chances with runaways in his shaft than do this. Specially constructed stringers, securely anchored in place, would have to be used. No one as yet has seen fit to go to this expense, and therefore it is considered good practice in incline shafts not to use safety appliances on skips and man cars operating on inclines. On the other hand, safety dogs can be installed on cages and skips in vertical shafts with small expense, and so it early became the practice to put on the dogs, as the guides have to be there whether dogs are used or not. No one has the nerve or the niggardliness to leave them off when he thinks much of his reputation among mining men.

It is a disgrace to the mining profession that the Gagnon incline is, I believe, the only incline in the United States, and probably the only one in the world where safety appliances for catching a runaway cage or skip are used in an inclined shaft.

Further, I doubt if there is a single hoisting engine in the United States working in an inclined shaft that has any provision against overwind, any rope detaching device, or an automatic cut-off for the steam as the car approaches the surface, that are characteristic of good hoisting practice in vertical shafts, because they all cost money."—CLAUDE T. RICE, *Engineering and Mining Journal*, April 27, 1912, p. 826. (A. R.)

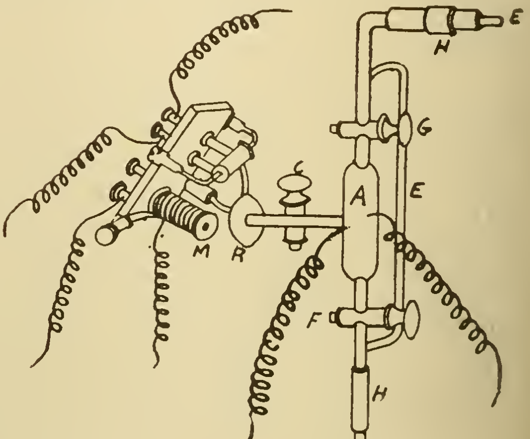
COAL FIGURES COMPARED. "In the year 1910 the coal production in Great Britain amounted to 264,417,588 tons, the number of employees at the mines being 1,049,407, of whom 1,769 (or 1·68 per thousand) lost their lives. Neglecting other minerals, by calculation it is seen that approximately 252 tons of coal were raised for each person engaged in or about the mines, whilst there were 6·7 deaths per million tons of coal raised; or say one person killed for every 149,473 tons of coal brought to the surface. In Pennsylvania the production of bituminous coal totalled 148,770,858 tons, in the raising and preparation of which 193,488 persons were engaged, of whom 539 (or 2·8 per thousand) were killed. Adopting the same method of calculation it appears that in Pennsylvanian mines approximately 774 tons of bituminous coal were raised per person employed, the deaths per million tons raised being 3·6, or say one life lost for every 276,012 tons produced. If mineral produced is taken as the standard it becomes necessary to admit that there is greater safety in Pennsylvanian bituminous mines than obtains in British mines. It may be interesting to set out the comparison in tabular form as follows:

	Great Britain.	Pennsylvanian Bituminous Mines.
Coal output in tons ...	264,417,588	148,770,858
Persons employed ...	1,049,407	193,488
Total number deaths ...	1,769	539
Death rate per 1,000 persons	1·68	2·8
Death rate per million tons raised	6·7	3·6
Tons mineral got per death	149,473	276,012
Output per person (tons)	252	774

—*Science and Art of Mining*, March 16, 1912, p. 361. (T. J.)

WINDING ROPES. "The Prussian statistics of shaft hauling ropes for 1910 which have lately been published, contain a large amount of information which is interesting and useful for rope makers as well as for mining men. They are exhaustively and critically discussed by Professor Herbst, of Aix-la-Chapelle, in a series of articles in *Glückauf.* Among the general conclusions arrived at, the following are of special interest: (1) The protective efficiency of lubrication has not been plainly proved except in dry shafts. This observation suggests the conclusion that the present lubrication processes for wet ropes leave room for improvement, though it is certain that all now known lubricative agents rapidly disintegrate in shafts with sour or salty waters. Future experiments in this direction may provide a remedy. (2) Galvanising or coating with zinc does not appear to have had a really protective efficiency in wet shafts, the reason being probably that zinc coating has but little power of resistance to salt water. It is also suggested that the wires have suffered by the galvanising process, for, though it has been proved by Winter and others that that process, when properly and carefully executed, does not unfavourably affect the ropes, it is also well known that it often reduces the tensile strength of the rope by 50 per cent. and even more. (3) The hauling efficiency of ropes in dry shafts stands in the proportion of 100 to 60 or 70 to that in wet shafts, a fact which, in view of the high prices of ropes, means a very substantial economic advantage for dry shafts. (4) Tensile strength between 160 and 180 kgs. per square mm. does not very unfavourably affect the flexibility or hauling strength of the ropes, while ropes of more than 180 kgs. per square mm. have given substantially lower efficiency figures. (5) The greater or lesser strain, as expressed by a higher or lower safety factor, put upon ropes has had no influence upon their consistency. It may, therefore, be assumed that the advantages of a higher safety factor are neutralised by its disadvantages—viz., greater rope thickness combined with reduced flexibility and greater dead weight."—*London Mining Journal*, April 20, 1912, p. 402. (A. R.)

AN EXPLOSIVE GAS TESTER.—"The gas tester shown in the sketch is intended to be located in a safe place and at a distance from localities where explosive gases may be suspected to exist.



Referring to the cut, A is an explosion pipette with safety packings of wire gauze at H.H. and provided with a by-pass B. The end D is connected by a composition tinned iron pipe to the point from which the air is to be obtained for testing, and the end E is connected with a pump so that a continuous stream of air is drawn through. The cocks F and G are closed (the current still streaming through the by-pass), and the spark from an induction coil passed between the wires. If no explosion occurs, it is because the proportion of combustible gas is above or below the explosion limit. Should it be above, then on opening for a while the cock C, enough air will diffuse in to cause an explosion on sparking again. Should it be below, then if F and G are opened and sparks passed while the gas is streaming through, flame will be seen in a darkened room with a proportion of combustible gas considerably below that necessary for explosion. If the explosion be made to take place with the cocks F and G closed and C open, and the suspended iron plate R closing the side tube. the force of the explosion will drive R against the electromagnet M, which will hold it, and electrical connections made by this contact may be made to ring a bell and thus give warning."—*Colliery Guardian*, April 12, 1912, *N. Tech. Zeitschrift für Prakt. Chem.*, p. 738. (T. D.)

DATA ON UNDERGROUND MINING.—"The data given in the accompanying tables were obtained from the records of a large mine in Nevada, where all mining is done underground. The development figures are based on records for 4,354 ft. of work, including the usual proportion of raising generally necessary in opening new ground. All shifts are of eight hours' duration.

Development Data.

	Per foot of Advance.
Average shifts of miners	0·655
Average shifts of shovellers and trammers	0·428
Pounds of dynamite	4·79
Number of blasting caps	2·74
Feet of fuse	17·45
Candles	4·28

During the same period the records for work in the stopes were :—

Stoping Data.

	Per ton of ore stoped.
Average shifts of miners	0·16
Average shifts of shovellers and trammers	0·095
Pounds of dynamite	0·446
Number of blasting caps	0·364
Feet of fuse	1·97
Candles	0·97

Based on the total tonnage of ore produced both from development and stoping for the same period, these figures are as follows :—

Mining Data.

	Per ton of ore produced.
Shifts of miners	0·271
Shifts of shovellers and trammers	0·169
Pounds of dynamite	1·46
Number of blasting caps	0·899
Feet of fuse	5·45
Candles	1·73

This would indicate that the underground workers handled 2·32 tons per man per shift, or a little more than 2 tons per man per shift, including all men employed about the mine. This average was later increased to about 3 tons per man. The number of candles used is dependent upon the distance to the working faces and draft. Approximately 14 tons of dry ore from stopes per shift of shovellers and trammers was the average at the time these records were made."—*Engineering and Mining Journal*, April 27, 1912, p. 838. (A. R.)

RATE OF BURNING OF FUSE.—"In Technical Paper 6 of the U.S. Bureau of Mines, Walter O. Snelling and Willard C. Cope discuss the Rate of Burning of Fuse as influenced by temperature and pressure. In presenting the practical conclusions arrived at as a result of a series of tests the authors say :—

It is important that time fuse should have a uniform rate of burning, and in almost all blasting operations the fuse that is used is assumed to burn in a regular and uniform manner. When fuse has been subjected to such conditions as produce acceleration or retardation in its rate of burning it becomes dangerous. Acceleration of the rate of burning increases the liability of a shot going off before the miner has left the face ; retardation increases the chance of the flame in the fuse progressing so slowly that the miner will be injured by a delayed shot when he returns to the working face. All conditions that bring about any marked change in the rate of burning of fuse are dangerous, and from a study of the list of accidents in mines and quarries each year injury and loss of life are seemingly often brought about by such conditions.

Under ordinary conditions nearly all types of fuse show great uniformity in their rate of burning. Practically all types of fuse examined in connection with the preparation of this report had a total variation in their rate of burning under normal conditions of less than 20% and all would have been passed under the allowance of 'no variation greater than 10% above or 10% below the average rate of burning.'

Under the influence of pressure practically all types of fuse are subject to wide variation in their rate of burning. Such pressure as can readily be produced by the confinement of the gases evolved by the burning fuse itself is sufficient to increase the normal rate of burning from 92·5 sec. per metre (28·2 sec. per ft.) to 21 sec. per metre (6·4 sec. per ft.). Thus, even confinement will cause fuse to burn from three to four times as rapidly as its normal rate. In experiments made with fuse confined by stemming of various kinds wide variations in the rate of burning were noted, and whenever lengths of fuse are confined by stemming or other materials impervious to gas, a sufficient length of the fuse should be used to allow for the increased rate of burning due to the pressure produced by the evolved gases.

High temperature causes a marked retardation in the rate of burning of fuse, and storage for even a short period of time near boilers, or wherever the temperature may be high, is sufficient either to cause 'missfires,' or to retard the rate of burning of the fuse so much as to greatly increase the liability to 'holdbacks,' delayed shots, etc. It is probable that many of the difficulties that are sometimes encountered in regard to fuse burning too slowly and causing delayed shots are due, in part at least, to such fuse having been kept in too warm a place. Fuse that is not intended for use in wet places (cotton fuse, etc.) does not suffer marked change in its normal rate of burning by reason of the effect of high temperatures, whereas the more completely waterproofed types of fuse show increasingly great effects from heat. Even exposure to comparatively low temperatures for considerable lengths of time causes marked retardation in the rate of burning of such fuse, and exposure to a fairly high temperature for even a short length of time may cause certain types of fuse to burn from three to five times as slowly as their usual rate. To ensure the best results, fuse should always be protected from extremes of temperature.

Climatic conditions affect to a considerable extent the rate of burning of the less waterproof type of fuse. Damp fuse burns more slowly than normal fuse, and fuse that has been wet and then thoroughly dried tends to burn at a rather slow rate, and may even cause delayed shots by smouldering for a considerable time. Fuse containing several wrappings of tape saturated with tar or asphalt resists moisture to a considerable extent, and may be used for firing shots under water, provided the fuse is not allowed to remain too long a time in contact with water before the shot is fired.

It is, of course, evident that when these water-proof types of fuse do become wet, whether through storage for a long time in a damp place or through exposure to water after the protecting layers of asphalt or gutta-percha are mechanically abraded or injured in any way, they are more difficult to dry out than are other types of fuse, and are more liable to burn at a rate slower than the normal rate.

Fuse that has been subjected to actual mechanical injury, particularly to hammering or pounding or the blows of falling rock, etc., has a greatly increased rate of burning, and sometimes burns so rapidly as to be almost instantaneous in its action. The mere bending, coiling, and twisting of fuse, such as would be brought about by forcibly placing within a bore-hole a length of fuse considerably greater than the depth of the bore-hole, does not produce any marked change in the rate of burning, but pounding or direct abrasion of fuse greatly increases that rate. Fuse that has been injured by severe abraison or by too great pressure from any cause should not be used in any work where adjustment of the rate of burning is desired.

As a final summary it may be stated that ordinary fuse may under some conditions burn as fast as 3 sec. per metre (1 sec. per ft.), and under other conditions it may burn as slowly as 745 sec. per metre (227 sec. per ft.). The former rate is more than 200 times as fast as the latter, and each is widely removed from the normal rate of burning of similar brands of fuse. Hence, the condition and past history of any roll of fuse is an important matter, and in mining and blasting operations the safety of the miner demands that only fuse that has been carefully stored and kept from unfavourable conditions shall be used."—W. O. SNELLING and WILLARD C. COPE, Technical Paper No. 6, U. S. Bureau of Mines, *Science and Art of Mining*, April 27, 1912, p. 456. (T. J.)

SHAFT SINKING RECORD.—234 FT. SUNK AT MODDER DEEP.—"The Modderfontein Deep levels were able in April to surpass the record accomplished at the State Mines in March by the narrow margin of 1 ft., the total footage being 234. The shaft was sunk in good breaking quartzite with sandstone partings, and it is considered that the possible footage that can be obtained in the same formation is 245. It will be recalled that twin-shafts are being sunk at the Modder Deep, situated 100 ft. apart. Each is of three compartments, two of 5 ft. by 6 ft. and one 6 ft. by 6 ft. Three sinkers and three assistant sinkers were employed, working eight-hour shifts.

The timber used is of similar size to that of the State Mines—*i e*, 9 in. by 9 in. The centres are 6 ft. for all but one set, that being a cross-cut set of 7 ft. 6 in. The number of sets used in timbering the distance sunk was 39, namely 38 of 6 ft and one of 7 ft. 6 in. The 39th set was put in after measuring. Two white timbermen are employed to do all the timbering work in the shafts. Bearers are put in every 50 ft. As at most sinking shafts, bratticing is used, but no lagging is required.

No special effort was made to try for extra footage until the last seven days, and it is, therefore, considered probable that this footage will be exceeded in some shaft in the district under similar favourable conditions."—*South African Mining Review*, May 1912, p. 103. (A. R.)

FRICTION IN AIR PIPES.—"The coefficients given in the accompanying tables were determined experimentally for use in calculating the friction of air in pipes by the formula

$$f = c \frac{l}{d^5} \frac{V^2}{r}$$

where f is the loss by friction in pounds per sq. in.; l, the length of pipe in feet; d, the diameter of the pipe in inches; V, the quantity of free air passing expressed in cubic feet per second; r, the ratio of compression in atmospheres, and c, the coefficient.

Table I.—Coefficients for Friction in Air Pipes.

DIAMETERS IN INCHES.		COEFFICIENT c FOR	
Nominal.	Actual.	Actual Diameter.	Nominal Diameter.
½	0·63		
¾	0·82	0·110	0·069
1	1·07	0·100	0·071
1½	1·63	0·089	0·059
2	2·07	0·082	0·069
3	2·874	0·072	0·089
4	3·937	0·066	0·071
5	4·921	0·062	0·067
6	5·960	0·059	0·061
7		0·056	
8	7·874	0·053	0·059
10		0·050	
12	11·811	0·048	0·052

For the ¾ in. and ½ in. pipes, especially the latter, the results of the experiments were so irregular, erratic and conflicting, that the coefficients given in

Table II.—Lengths of Pipe in Feet of Resistance Equal to that of Fittings.

Nominal Diameter of Pipe.	90° Elbows.	Unreamed Joints, Two Ends.	Reamed Joints, Two Ends.	Return Bends.	Globe Valves.
½	10	2 to 4	1	10·0	20
¾	7	2 to 4	1	7·0	25
1	5	2 to 4	1	5·0	40
1½	4	2 to 4	1	4·0	45
2	3	2 to 4	1	3·5	47

the tables for those sizes cannot be accepted as final. In light of these studies it is not probable that a satisfactory coefficient will ever be obtained for pipes less than 1 in. diameter, the reason being that in such pipes irregularities have relatively greater effect than in larger pipes. No doubt consistent results could have been obtained by using only pipes that had been 'plugged and reamed' and selected fittings; but the results would not have been a safe guide for practice.

A number of experiments were made to determine the resistance of 50 ft. lengths of rubber-lined, armoured hose, such as is used for connecting air tools with the supply of compressed air. These, given in Table III., include the resistance in a 50 ft. length with the metallic end couplings, where considerable contraction occurs; in ½ in. hose the end couplings are ¼ in. The excessive resistance in the ½ in. hose may have been due to these contractions or to some other obstruction. It is a further illustration of the fact that reliable coefficients cannot be obtained for pipes of less than 1 in. diameter."—E. G. Harris, *Bulletin, University of Missouri.—Engineering and Mining Journal*, April 27, 1912, p. 834. (A. R.)

Table III.—Resistance of Air Hose.

Diameter, inches	½	¾	1	1½
Resistance in 50 ft. length ...	$950\frac{V^2}{r}$	$20\frac{V^2}{r}$	$4{\cdot}5\frac{V^2}{r}$	$2{\cdot}6\frac{V^2}{r}$

Coal Mining Statistics.—"Examination of the tables issued from the Home Office shows that last year there was an increase in the output of mineral under the Coal Mines Act of 7,490,072 tons, the increase in the number of persons employed being 17,806. In the result rather more mineral per person employed was recorded than in the previous year, but the 1909 average was not reached by some six tons per person. The output of coal alone was 271,878,924 tons, an increase of 7,461,336 tons, made up as follows:—Scotland, 383,031 tons; Newcastle, 2,156,138 tons; Durham, 1,814,303 tons; York and North Midland, 1,196,546 tons; Manchester and Ireland, 25,763 tons; Liverpool and North Wales, 227,055 tons; South Wales, 1,500,745 tons; Midland and Southern, 157,755. Thus all the inspection districts contributed towards the increase, which works out at the rate of 2·82%, the increase in the number of persons employed being at the rate of 1·70%. In the Manchester district, it will be noticed fewer persons were employed although the output exhibited an upward tendency.

Regarding accidents 1911 was a record year. Compared with 1910 there were 516 fewer persons killed, and as a consequence the death rate per thousand employés fell from 1·69 to 1·18, whilst the deaths per million tons of mineral declined from 6·37 to 4·4. From the point of view of persons employed this is the lowest death rate recorded in the Blue Books for any one year, a feature that should afford considerable satisfaction to the whole community.

The following table shows the number of persons employed, the output of mineral, the total deaths from accidents, the death rate per thousand persons employed and per million tons of mineral raised, together with the tons of mineral got per person engaged during the year 1909, 1910 and 1911:—

District.	Persons Employed.			Mineral Output in Tons.			Total Deaths from Accidents.			Tons Mineral Raised per Person.			Death Rate per Thousand Persons.			Deaths per Million Tons of Mineral Raised.		
	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.
Scotland ...	133,605	137,873	138,377	44,455,555	46,205,552	46,548,384	200	209	194	332·7	335·0	336·4	1·49	1·5	1·4	4·49	4·48	4·1
Newcastle ...	112,113	119,232	121,782	30,164,214	28,962,499	31,145,939	86	242	122	269·0	242·9	255·7	0·76	2·0	1·0	2·85	8·35	3·9
Durham ..	107,283	112,881	114,850	34,246,752	32,482,057	34,152,804	276	104	121	319·2	287·7	297·3	2·57	0·91	1·05	8·05	3·17	3·5
York & North Midland ..	231,087	237,864	241,282	63,424,390	66,302,769	67,531,239	266	260	240	274·4	278·7	279·9	1·15	1·09	·99	4·19	3 92	3·5
Manchester & Ireland ...	49,747	50,851	49,963	11,479,607	11,206,503	11,254,760	58	403	54	230 7	220·3	225·2	1·17	7·9	1·08	5·05	35·88	4·8
Liverpool and North Wales	69,763	69,712	70,229	16,201,737	16,265,208	16,498,734	90	82	78	232·2	233·3	234·9	1·29	1·17	1·4	5·55	5·04	4·8
South Wales	204,984	213,252	220,887	50,583,991	48,907,880	50,356,776	355	323	309	246·7	229·3	227 9	1·7	1·5	1·3	7·01	6·54	6·1
Midland and Southern ...	105,416	107,742	109,843	27,349,813	28,121,492	28,454,396	122	152	141	259·4	261·0	259·0	1·15	1 42	1·28	4·46	5 44	4·9
Totals	1,013,998	1,049,407	1,067,213	277,906,059	278,453,960	285,943,032	1,453	1,775	1,259	274 07	265·3	268·1	1·43	1·69	1·18	5·2	6·37	4·4

Advance Tables issued from Home Office, *Science and Art of Mining*, March 16, 1912, p. 369. (T. J.)

MISCELLANEOUS.

Presidential Address.—"*Technical Education.* It is surprising to find that there are still some people, notably in America, who depreciate the value of technical schooling for those who select some special branch of engineering as their profession, while there are others who, although recognising the value of technical education, hold oddly and widely divergent opinions as to how it should be acquired. Unquestionably, however, the large majority of thinking men who have studied the subject from a disinterested point of view are agreed that thorough theoretical training is essential to the success of the young engineer of the present day, while of course unanimous in urging the importance of the student acquiring a considerable amount of practical experience, either before, during, or after his course of, say, four years at a technical college. Practical instruction, as the late Professor Huxley put it, is the heart and soul of all scientific education.

In this connection it is interesting to quote the view expressed by Mr. Edison, the world-famous American inventor, with regard to the value of college-trained students for practical work. In his opinion 'they do not seem able to begin at the beginning and give their whole heart to the work.' May not the reason for this lie in the fact that the training of the students he had in mind had been so exclusively theoretical that they were filled with the conceit of learning and an inordinate idea of their untried abilities? If their theoretical and practical training had gone hand in hand, surely any conceit of knowledge would have been largely dissipated by the hard realities of practice. Either without the other is impotent; combined they are invincible. I do not see how it can be questioned that in our or any profession a man well trained in fundamentals has a broader grasp and can more intelligently and rapidly utilise experience than one without such training.

The system of general education in Great Britain, more especially in respect of secondary schools, is even now lamentably behind that of certain other countries, notably Germany and America, chiefly for the reason that it is not organised and controlled by some central authority, such as the State, but is left in the hands of unqualified people, many of whom hold crude and mistaken ideas as to the methods to be adopted in teaching. On the other hand our chief technical colleges now compare favourably with those of other countries.

It has been claimed that the prodigious increase in the world's production of gold has not been altogether desirable. Indeed, it is unkindly asserted by some political economists that the net result of our labours, so far as gold production is concerned, has been merely to raise the price of commodities in general, and of raw materials in particular, and thereby to depreciate the purchasing power of wages. It is satisfactory, however, to be told on high authority that larger supplies of gold stimulate trade and increase the amount of work to be done, and so tend to restrict the range of their own effect on prices—a view supported by my diagrams, which show that the prodigious increase in the producton of gold has synchronised with enormous increases in the production of coal, copper and iron and steel during the past 50 years. If I had chosen to burden this address with diagrammatic statistics relative to other important industrial metals, the same feature would have been exhibited. I think, therefore, that our successful efforts to reduce the cost of obtaining gold may, after all, have been a boon and blessing to mankind in general and in particular to the wage-earner, who has not only found himself in more regular employment but, in the majority of trades dependent upon metals as their raw material, has won an increase in wages considerably more than commensurate with any increase in the cost of comfortable living.

But I am perhaps digressing a little from the question of technical education, which I cannot dismiss without referring to the regrettable fact, known to all of you, that it is open to anyone without the smallest technical training, and with little or no experience, to describe himself as a mining engineer. Nothing can be gained by shutting our eyes to the fact that, like all other professions (though I am glad to think in no greater proportion) we have black sheep within our fold, to say nothing of the wells meaning fools who sin unwittingly and bring us into disrepute and their clients to disaster. The self-styled mining experts are all the more dangerous when, having no code of business morality, but a very facile pen, they chance to have acquired more than a smattering of mining knowledge: they are the willing tools of unscrupulous company promoters, who, for a small fee or a contingent interest, desire to obtain reports that will suit their fraudulent purposes.

Yet, what action can any one of us, or even this Institution, take in such cases? I trust, nevertheless, that the day is not far distant when it will be recognised by the public and in our courts of law that, unless a man styling himself a mining engineer or metallurgist is on the roll of this or some other equally responsible technical society, he is inferentially not possessed of the necessary qualifications, and that the layman employing such a man does so at risk of losing whatever reputation he may hitherto have had for straight dealing. I do not hope, however, for such protection against charlatans as is enjoyed by the legal and medical professions. A certificate of competency, by whatever examining authority it might be granted, would not be a sufficient safeguard, our profession being peculiarly one wherein business capacity is hardly less important than technical knowledge; and business capacity is not readily tested by examinations.

Standards and Definitions.—I come now to the subject of Standardisation and Definition to which the Council of this Institution have devoted so much thought of late years. The decisions taken in this matter must tend increasingly to the advancement of our profession in the esteem of the public, who, after all, are not only our critics, but also our employers.

Why have we been at such pains to define 'ore-in-sight,' to standardise 'weights and measures,' to lay down general principles in regard to the allocation and grouping of expenditure upon mines, and so on? Primarily, I believe, in order that the investing public may be protected. The Council of this Institution have rightly decided that no ambiguity or mystery shall be tolerated in these matters, and have declared in reference to 'ore-in-sight' that such ambiguity will be regarded as an indication of dishonesty or incompetence.

Of course with regard to 'ore in sight,' in spite of our endeavours to eliminate uncertainty as far as possible, nature being the fickle jade she is, and the personal equation being the variable element it is, the valuation of a mine must sometimes err. Winzes and drifts may raise too flattering expectations; a leading stope may altogether change the aspect of a block of ground. Mining has more disagreeable surprises than any other industry I know, and no formula or definition can ensure us against them. The fixing of average metal-contents per ton to three or four places of decimals is mere pedantry. It may fool the public into crediting the mine-valuer with having done his work with scrupulous and unusual exactness; but woe betide him if his estimate should not be realised—and little sympathy will he deserve from the profession he brings into disrepute. But, having said this, I feel it is all to the good that we should know, and that the public should know, exactly what is meant, or should be meant, by 'ore-in-sight.'

Some lamentable mining fiascos have occurred within the last 10 or 15 years which must be still fresh in the memories of most of us, when unsuspecting shareholders were awakened suddenly to the fact that the 'practically inexhaustible' tonnages of 'ore-in-sight' existed for the most part only upon paper, or that the valuation of the ore-reserves was so greatly in excess of actuality as either to necessitate an immediate reduction in the profits or to remove the mine indefinitely from the list of dividend payers.

Let me quote a few cases in point:—

(*a*) A famous mine which had contributed largely to the world's production of gold, and, according to the optimistic views of the local management, was expected to maintain its output for many more years. Presumably owing to an intelligent anticipation of events, the weakness of the shares suddenly became a prominent feature of the mining market, for which the only explanation offered by the Board was that the decline in quotations was due in part to a natural reaction from an excessive valuation of the shares, although they admitted that the annual rate of production might shortly have to be reduced by a certain specified amount of quite important size. Within three months of this admission, the manager himself was obliged to agree that the annual rate of production must be reduced by a further amount more than three times that previously suggested, and it was only then explained by the Board that developments in the bottom level had not come up to expectations.

(*b*) An instance in which a manager apparently failed to adopt the well-considered methods of valuing ore-reserves advocated by our Council. An outside engineer of high repute having been called in to check the manager's estimates—which had come to be regarded with suspicion—found himself obliged to reduce the reported value of the reserves by something like a million sterling, although basing his estimate on the manager's own sampling and measurements.

(*c*) A case of sheer, muddling incompetence on the part of the management. Process after process for the treatment of the ore was recommended as suitable, and large but utterly abortive expenditure was incurred on metallurgical plants. In addition, the management had, late in the day, to admit the necessity for decreasing their estimate of ore-reserves, which failed to take proper account of the system by which the ore-body would have to be mined.

(*d*) An instance of criminal misconduct, where the manager failed to report officially to his Board certain valuable developments in the mine until it suited the books of himself and confederates in London to disclose the true position.

Now I only mention these unfortunate occurrences in order to press home the following three points: *First*, that valuations of ore-reserves should be made on the lines laid down by our Council; *second*, that a periodic checking of the mine manager's estimates of ore-reserves should be made by an independent valuator; and, *third*, that it is the bounden duty of mine managers, and should be regarded as a point of honour, to keep their Boards promptly and fully informed as to the progress and results of development and exploratory work.

In every one of the cases above quoted no outside check was kept on the manager, such as could have been obtained by the employment of an independent consulting or inspecting engineer. Due care is taken by companies, and the law compels them, to employ specialists to audit their accounts, and in many instances expert accountants are engaged to check up the stores at the mine; but how seldom does one hear of an independent engineer being asked to audit the ore-reserves, which, after all, constitute the chief asset of the company!

Technical Press.—Incidentally, I think we owe a debt of gratitude to these journals for the publicity so readily afforded for the records of notable achievements by members of our profession; for such publicity not only stimulates discussion but engenders a spirit of healthy rivalry, with material advantage not only to the profession but also to the investing public. To quote from a technical magazine edited by a distinguished member of this Institution:—

'Discussion is the whetstone of thought; by the exchange of ideas the wits of the engineer are sharpened and his methods shaped to an economic purpose.'

In speaking of publicity I deprecate, of course, the craving for the kind that may properly be described as advertising, a question which has exercised the minds of other professions besides our own. Not long ago one of our learned Judges made the remark that, 'when one man deprecated the publicity which his neighbour enjoyed, that only meant that he wished to have a large share of that publicity himself.' I trust that I shall not by these remarks lay myself open to that charge.

Professional Ethics.—In passing, I would suggest that the 'Advancement of our Profession' is not assisted by the rather reckless way in which consulting engineers and managers have sometimes been known tacitly to acquiesce in the witholding of important information from shareholders, or in the publication of misleading statements.

Of course, as engineers, our very existence depends largely upon the discovery of new mines and the provision of sufficient capital to exploit them; but company promotion has often worked to our detriment. Given a remote country, a few sensational assays or crushing returns, together with the report of someone generally described as an 'eminent mining expert,' and a certain class of company promoter is fully equipped. Naturally, when the statements of the prospectus, backed as they were by the authority of the 'eminent expert,' are proved to be unfounded, it is not the promotor but our profession that suffers.

Although I am only briefly referring to this subject, I certainly do not underrate the importance of its bearing upon our profession, but it seems to me that one can sum up all rules for professional conduct in the two words, 'Be honest;' or, as Dr. R. W. Raymond has suggested:

'Do not do what you could not tell without shame.'"—EDWARD HOOPER, *Institution of Mining and Metallurgy, Bulletin No. 92*, May 16, 1912. (A.R.)

The World's Production and Distribution of Gold.—"An extensive review of the production and distribution of gold during the 21 years beginning with 1890 and ending with 1910 is given in a report by George E. Roberts, director of the mint. These years, according to Mr. Roberts, cover the period of what has been called the 'new golden era,' beginning with the discovery of the Transvaal gold field and of the cyanide process of extraction.

Gold production is shown to mount steadily upward, save for the interruption caused by the Boer war, from $113,000,000 in 1890 to $454,000,000 in 1910. The review then undertakes to trace these new supplies of gold into use, showing the estimated consumption in the industrial arts, absorption by Asia, amounts taken by new and developing countries, and finally amounts that have entered into the bank reserves of the important commercial countries.

The 21 years are divided into two periods, the first of 10 years, 1890-1899, and the second of 11 years, 1900-1910. Production in the first period is estimated at $1,960,000,000, which is found to have been distributed approximately as follows :

Industrial arts	$570,000,000
Banks and Treasury of U. S.	260,000,000
U. S.	260,000,000
European banks	686,800,000
Canada, Australasia and S. Africa banks	59,700,000
Egypt and Asia	195,000,000
	$1,771,500,000
Other banks, circulation, private holdings, etc.	188,000,000
Total	$1,960,000,000

During the second period of 11 years, production amounted to $4,037,000,000, and distribution is estimated to have been substantially as follows :

Industrial consumption	$958,000,000
Absorption of India	433,000,000
Absorption of Egypt	146,000,000
Bank of Japan	69,000,000
Conver. funds of Argentina and Brazil	343,000,000
Banks of Mexico	28,500,000
Banks and Treasury of U. S.	726,800,000
Banks and Treasury of Canada	85,700,000
Banks of Australasia and S. Africa	95,600,000
Banks of issue Europe	863,200,000
	$3,748,800,000
Other banks, circulation, etc.	288,200,000
Total	$4,037,000,000

Industrial consumption about doubled during the period under review and for 1910 is estimated to have totalled $111,848,500. Consumption of leading countries for 1910 follows: United States, $33,756,000; Great Britain, $18,000,000; France, $16,836,000; Germany, $15,536,000; Switzerland, $5,670,000; Austria-Hungary, $5,750,000; Italy, $3,000,000; Russia, $4,000,000, and Canada, $1,850,000.

The most striking feature of the review is the showing of an increasing absorption of gold by India. That country in its relations with Europe has always had a favourable trade balance which had to be settled in the precious metals, but until recently the principal medium has been silver. Of late the trade balances have been growing larger, as a result of increasing exports and the higher prices received for them, and while India continues to take about the same amount of silver as formerly, it is also taking large sums in gold. For the 10-year period, 1890-1899, the net imports of gold into India, plus the country's own production, were $135,800,000; for the 11 years, 1900-1910, they aggregated $433,800,000; for the British fiscal year ended March 31, 1911, they amounted to $90,487,000, or about one-quarter of the world's production after the industrial consumption was provided for."—*Mining and Engineering World*, April 27, 1912, p. 896. (A. R.)

Gold Accumulation in India. — "Moreton Frewen, M.P., recently gave to the London correspondent of the New York *Times* an interview on the question of the great drain of gold to India which is going on, and which he described as a matter of the very first importance to America.

'The position is this,' said Frewen. 'The Government of India in 1910 imposed a duty of 8 cents an ounce on all silver bullion imported into India. The importation of silver previously for some years averaged 80,000,000 to 90,000,000 ounces annually. The result of the high silver duty seemed to discourage importation, and the hoarding of silver by natives, and to throw the whole demand of India for hoarding purposes upon the other metal—gold.

The position is one of great and growing danger. The drain of gold to India at the present time threatens the most serious consequences in the money markets of the western world. Whereas the importation of gold into India before the new and high silver duties didn't exceed, one year with another, £4,000,000 sterling, the importation of gold last year into India was over £20,000,000, and for the first 11 weeks of this year was at the rate of nearly a million sovereigns a week.

The Government at Calcutta has now applied to the Secretary of State for India for permission to commence the coinage in India of 10-rupee gold pieces, thus offering to natives, through the innumerable bazaars throughout India, a means of hoarding in the most convenient form. Such mintage would make gold, for the purpose of hoarding, accessible to 300,000,000 of people.

Should the Government consent to this proposal, the hemorrhage of our gold to India would further threaten western finance. It seems to me of the very first importance on the part of the great western financial centres that representations should be addressed to the Government here.

The present movement of gold to India is on such a scale that when the gold required for arts and manufactures has been deducted, comparatively little of the present immense supply of gold from the mines is left for American or European currencies. We shall revert to conditions in the '80s."—*Mining Science*, April 18, 1912, p. 344. (A. R.)

New Books.

Baly, E. C. C. Spectroscopy. New edition. Cr. 8vo., pp. 702. *Longmans.* 12s. 6d.

Barker, A. H. Barker on Heating. The Theory and Practice of Heating and Ventilation. 4to., pp. xvi.-640-lxxvi. *Carlton Press.*

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Abstracts of Patent Applications.

(C.) 350/11. William McVitty (1), Ernest William Massam (2). Improvements in cam shaft bearings. 23.6.11.

This invention relates to bearings for the cam shafts of stamp batteries with the object of minimising vibration and preventing breakage of the shaft.

The invention consists of rectangular or circular boxes bolted to the frame of the battery, and a separate bearing for the shaft carried in the boxes upon rubber and felt cushions, the bearing being able to move freely in a vertical direction within the box but not being able to move in any other direction.

The inventors state that by the use of this particular design of bearing the felt and rubber cushions will, by taking up the vibrations of the battery framing, prevent the cam shafts from vibrating and breaking.

(C.) 371/11. William Thomas Bell (1), Arthur John Wood Graham (2). Improved construction of apparatus for automatically controlling power driven constant speed air compressors. 5.7.11.

The application covers a device for governing power driven constant speed air compressors by means of a variable timed valve opening to the atmosphere during a part of the compression stroke.

(C.) 61/12. Milton Dana Morrill. Improvements in or relating to moulds for concrete walls, floors, ceilings, and the like. 26.1.12.

This invention relates to sectional moulds such as are employed in the construction, in concrete or other plastic materials, of buildings, walls, partitions, floors, ceilings, etc., etc.

The invention relates more particularly to means to be adopted to facilitate the adjustment of moulds at corners and for odd lengths.

The inventor claims for a mould for concrete walls and like structures, constructed substantially as described in the specification accompanying this application and adapted to be operated in the manner and for the purposes set forth.

(C.) 155/12. Daniel Bosqui. Process and apparatus for precipitating materials from solutions. 15.3.12.

The object of this invention is to improve the methods of using zinc dust for precipitation of gold by increasing the time of contact in an adjustable manner and especially by re-circulation of the mixed solution and precipitant with simultaneous agitation thereof without exposure to the air.

(C.) 166/12. Ernst Ritter. Improvements relating to the construction of hollow beams and the like. 22.3.12.

This application refers to the construction of hollow beams and the like of flat or shaped parts or strips.

(C.) 186/12. Botho Schwerin. Improved process for treating finely divided substances by electric osmose. 29.3.12.

This application is for a process for the separation of finely divided matter in suspension by taking advantage of the differently electrically charged bodies in a colloidal condition, which are respectively known in colloidal chemistry as "gels" which will settle and "sols" which remain in suspension. By varying the electrolyte "gels" may be made to act in the manner of "sols" or the reverse. The particular application quoted is in the purification of kaolin, whereby the iron-pyrite, iron oxide and the like, may be first separated and the colloidal clay may be obtained pure in a strongly dehydrated condition by the passage of an electric current through the suspended colloidal matter.

(C.) 201/12. John Galileo Barclay (1), Archibald Newton Barclay (2). Improvements in connection with the raising and lowering of pit cages.

This invention relates to means of the kind for preventing the overwinding of pit cages, and the object of the invention is for a mechanism to arrest the motion of the engine by

(1) Actuating the throttle valve so as to cut off the supply of steam.
(2) Actuating the brake for slowing down the engine.
(3) By centering the reversing lever.

These three objects being effected in the sequence given above.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 208/12. John George Robinson. Improved pressure release valve for use in connection with locomotive and other steam engines. 9.4.12.

(P.) 211/12. James Conning. Improvements in respirators. 10.4.12.

(P.) 212/12. Guiseppe Antonio Pietro Provay. Improvements in truck tipplers or means for discharging the contents of trucks and like vehicles. 10.4.12.

(C.) 213/12. Gogu Constantinescu. Improvements in carburettors for internal combustion engines. 12.4.12.

(C.) 214/12. Wharton Byrd McLaughlin. Improvements in and relating to a method of and apparatus for desiccating fluid substances. 12.4.12.

(C.) 215/12. Alexander Alsworth Keith. Improvements in or relating to telephone systems. 12 4.12.

(P.) 216/12. Herman Horwitz. Apparatus for measuring and distributing liquid soap and other liquids. 15.4.12.

(P.) 217/12. Frank Edward Cakes. An improved torch for lighting fuses for blasting the like. 16.4.12.

(P.) 218/12. Adam Richard Stacpoole. Improvements in the linings of tube mills, amalgam barrels and the like. 16.4.12.

(C.) 219/12. Henry Charles Herbert Braun. Improvements in means for stoppering bottles. 16.4.12.

(C.) 220/12. Antal Fodor. Apparatus for connecting, conducting wires, cables and the like. 16/4/12.

(P.) 221/12. Alfred Pickthall (1) and Daniel Roberts (2). The Pickthall patent dust arrestor. 17.4.12.

(P.) 222/12. Lewis Horwick. Improvements applicable to kettels, pans, urns, and other utensils for the heating of liquids. 18.4.12.

(C.) 223/12. William King Riley. Improvements in processes and products of forming drill steel. 19.4.12.

(C.) 224/12. William King Riley. Improvements in drill mechanism. 19.4.12.

(C.) 225/12. Whitman Symmes (1) and Walter Edwin Trent (2). Method and apparatus for precipitating cyanide solutions. 19.4.12.

(P.) 226/12. James Henry Reid (1) and George Goodwin (2). Improvements in electric arc furnaces.

(C.) 227/12. The Northern Lime Co., Ltd. (1) and Bernard MacDonald (2). Improvements in apparatus for treating ore pulp slimes and the like. 20.4.12.

(P.) 228/12. Wilhelm Mauss. Thermodynamic process and apparatus. 22.4.12.

(P.) 229/12. Arthur Leon Felix Guillaume de Groof. An improved agricultural implement for harrowing, weeding and levelling land. 22.4.12.

(C.) 230/12. Hans von Kramer. Improved means for the regulating and control of railway and like traffic. 23.4.12.

(C.) 231/12. George Babe Jackson. Improvements in or relating to detonating signals for railways. 23.4.12.

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